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MEETINGS HELD IN NEW YORK AND SAN FRANCISCO IN 1922.

PREFACE

In this volume are the papers and discussions presented at the New York meeting, February, 1922, except those on Iron and Steel (which have been printed in Volume LXVII), and the San Francisco meeting, September, 1922; also Proceedings of the San Francisco meeting.

CONTENTS

BIOGRAPHICAL NOTICES

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LEDOUX, ALBERT R., M, '95–7; V, '98–9, '19–22,
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*LESLEY, J. P., M, '72–4.
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LINDGREN, WALDEMAR, V, '12–13.
LUDLOW, EDWIN, D, '16–19, V, '19–20, P, '21.
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MATHER, W. G., D, '21–22.
MATHEWSON, E. P., D, '13–15, '21–22.
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    5.
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PROCEEDINGS

The San Francisco Meeting, 1922*

THE 126th meeting of the Institute, held in San Francisco, Cal., Sept. 25 to 29, 1922, was an unqualified success. Over 230 names were registered and all sections save three were represented.

In the technical meetings, under the leadership of Albert Burch and C. W. Merrill, the greatest interest was manifested in ore-hunting geology and modern ore-dressing methods.

Mr. Burch opened the Monday mining session with a cordial welcome to the men of the profession, and an expression of pleasure that the Institute meeting was being held in the city of San Francisco. He then introduced President Arthur S. Dwight, who spoke on the general policies of the Institute. Secretary Sharpless, who followed, spoke of the cordial reception that had been accorded the Eastern party and the spirit of good fellowship that lives in and long survives these occasions.

MINING GEOLOGY SESSION

The paper presenting the economic geology of the United Comstock mines, by Wilbur H. Grant,† was read at the Monday morning mining session. The next paper was a description of the Aztec mine, Baldy, N. M., prepared by Charles A. Chase and Douglas Muir. Three other papers of geologic interest were read by title only: "The British Columbia Batholith and Related Ore Deposits," and "Deposition of Copper Carbonate from Mine Water," both by Philip D. Wilson of Warren, Ariz., and "The Ore Deposits of Sierra Mojada, Coah., Mexico," by S. F. Shaw, Esmeralda, Mexico.

The first paper presented at the session on Monday afternoon was by John R. Reigart on the underground system of handling ore and method of stoping of the United Comstock Mines Co. Other papers were "Metal-mine Ventilation in the Southwest," by Charles A. Mitke, "Shot-firing by Electricity," by N. S. Greensfelder, "Mining Methods in the Mineville (N. Y.) District," by Earl C. Henry.

The last two papers in this session were read by title: "Sampling and Estimating Orebodies in the Warren District," by R. N. Dickson,‡

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* Detailed descriptions of the journey of the Institute party from New York to San Francisco and return and of the week in San Francisco were published in *Mining and Metallurgy* (Oct. and Nov., 1922).

† Published in *Mining and Metallurgy* (November, 1922) 39, 41, 44, 47.

‡ Distributed in mimeographed form only.

and "Methods of Sampling and Estimating Ore in Underground and Steam-shovel Mines of the Copper Queen Branch of the Phelps Dodge Corporation," by R. W. Prouty and R. T. Green.*

MILLING SESSION

The milling session was opened at 2 p.m. Monday, by Chairman C. W. Merrill. The first two papers on milling methods and metallurgy of the United Comstock Mines Co., by Walter L. Reid and A. J. Weinig,* were read by Mr. Merrill.

The paper by Ernest A. Hersam, "Factors Controlling the Capacity Rock Crushers," was read by title. "Basic Principles of Gravity Concentration—A Mathematical Study," by Theodore Simons, was read by the author, and "Surface Tension and Adsorption Phenomena in Flotation" by A. F. Taggart and A. M. Gaudin, was read by title.

The paper by John Gross and J. Walter Scott, "The Mechanism of Gold and Silver Precipitation on Charcoal from Cyanide Solution," was read by Mr. Gross.

IRON AND STEEL, PETROLEUM, AND INFORMAL DISCUSSION

The iron and steel session was scheduled to meet on Monday morning, but because of the special interest of most of the engineers in the mining meeting, there were not enough present at the iron and steel session to hold a meeting. The papers on the program for this session were: "Neumann Bands as Evidence of Action of Explosives upon Metals," by F. B. Foley and S. P. Howell; "Effect of Nickel-chromium on Cast Iron," by Richard Moldenke; "Conditions of Stable Equilibrium in Iron-carbon Alloys," by H. A. Schwartz, H. R. Payne, A. F. Gorton and M. M. Austin.

Two papers on petroleum were presented, one by Roy W. Kelley on "Industrial Relation of the California Oil Industries Syndicate,"† and one on "Colombian Oil Fields," by L. G. Huntley and Shirley Mason.

Tuesday evening was devoted to discussions on prospecting and ore-hunting geology, referring largely to Prof. Augustus Locke's paper which was presented on Monday afternoon although it was not on the list of papers scheduled on the program.

SPECIAL ENTERTAINMENTS

President Dwight and Secretary Sharpless, with other engineers, started on Tuesday morning to visit the development work on the Hetch Hetchy project and spent three days as guests of the city of San

* Published in *Mining and Metallurgy* (Nov., 1922) 51.
† Not published by the Institute.

Francisco, inspecting the operations in the Hetch Hetchy valley. While the President and Secretary were in the mountains, those who remained in the city were well entertained.

On Tuesday morning, the 26th, there was a tour through Chinatown for both the men and the women of the party, followed by a luncheon given by Mrs. Sidney Cloeman at the Burlingame Country Club. After the luncheon the party was driven through the Crocker and Grant estates.

On Wednesday the party was taken by automobile to the University of California. Small groups, each headed by a student, were taken through the Greek theater, many of the University buildings, and up to the Campanile, from which a splendid view of the surrounding country was obtained. Luncheon was served at the Faculty Club, after which the Skyline drive was taken, through Oakland and back to the university. Later tea was served at the home of T. A. Rickard. Mrs. Rickard and Professor Weeks of the university entertained the guests with music and recitals.

On Thursday, Leland Stanford Jr. University was visited. A committee, of which Prof. Theodore Hoover was chairman, conducted the party around the campus and through the various buildings. In the afternoon the women were shown the Art Gallery and Museum while the men were taken through the engineering buildings. A special organ recital was given for the visitors before they returned to the city.

On Friday the party was entertained at luncheon by Mrs. Requa and Mrs. Burch at the home of Mr. and Mrs. M. L. Requa. Madam Requa, now in her 95th year, also acted as hostess and was constantly the center of interest with her brilliant conversation. The Woman's Auxiliary formed the subject of discussion after luncheon, both Mrs. Dwight and Mrs. Burch speaking of its activities. Mrs. Rickard and Mrs. Hutton entertained with music until it was time for the distant members to say farewell to their San Francisco hosts.

Theoretically, the California meeting was a pay-as-you-go, Dutch-treat meeting. On a few occasions, the members did pay for luncheon or a street-car fare, but the hospitality of the San Francisco members seemed always to have had the better of them. More was done for the visiting members than they could possibly have expected, and the 126th meeting will long be remembered as an event in Institute history.

PAPERS

Mining Methods and Installations of Anaconda Copper Mining Co. at Butte, Mont.

By Wm. B. Daly* and C. L. Berrien,† Butte, Mont.

The problems encountered in properly operating the twenty-eight mines of this company have called for various improvements in methods and installations. In order that those not familiar with local conditions in Butte may have a better understanding of them, and to show that mining practises must vary as conditions vary, a short account of these methods is given.

Most of the present operating shafts of this company were located between fifteen and thirty-five years ago, and have attained different depths and extent of lateral development. During most of the time, the mines now operated by the Anaconda company were operated by at least nine companies.

The larger mines occupy the central part of the camp, or an area $1\frac{1}{2}$ miles square, where the complicated vein and fault structures increase the mining difficulties in many ways.

These mines are connected on almost every level either by stopes or drifts. The underground waters of all are taken from the mines through two main pumping plants and shafts. The 2800-ft. is the first level common to all mines in elevation. Each mine maintains its own air lines, electric-power lines, water supply and ventilation. The average depth of stoping is about 2000 ft., but many of the mines are working on ore up to the 200-ft. level, which was left in former years because it was too low grade, or because it was not discovered until the more extensive lateral development after the consolidation under Anaconda ownership. As a result many of the stopes in late years have been in, or near by, old workings and it has been necessary to keep open more levels than would be required in a new mine. This condition requires additional ventilation facilities and special operations; we are working to the end that as few levels as needed will be open for development and stoping to assure normal production at each mine.

* Assistant Manager of Mines.
† General Superintendent of Mines.

Increased heat with depth, additional requirements for electric power, safety considerations, better working conditions and greater efficiency have brought about many improvements in late years. This company has always operated its mines according to the best practice, but since 1914 its officials have endeavored to meet the demands of changing conditions.

The principal improvements in mining methods and installations since 1914 are described under the following headings:

Bureau of safety
Ventilation
Mine fires and hydraulic filling
Underground fire prevention
Power distributing system for deep metal mines.
Hauling ore in mines of Butte, Mont.

Electric signal installations
Electric haulage systems
Hoisting ropes and safety in hoisting
Stope cost records and mine contracts
Organization of mine sampling

Bureau of Safety.—Since the beginning, in January, 1914, the Bureau has been under the direction of C. W. Goodale, assisted by John L. Boardman. The returns, measured in welfare to the employees and benefits to mine operations in general, have been inestimable. During these seven years, it has been an ever-increasing asset well worth the individual effort given it and the financial outlay made in conducting it. The maintenance of such a department is recommended to all companies, large and small, and, although this work is slow to install in the daily working of a mine, the regulations of this Bureau are now carried out to the letter by everyone concerned. The records, reports, and general conferences of bosses not only serve their purpose in regard to safety, but keep the whole organization in closer touch with the operations.

Ventilation.—The later developments in ventilation conditions in the Anaconda company mines extend over eleven years. Up to 1910, the mines depended on natural ventilation, assisted by compressed air from pipe lines of the drilling air systems. During that period, most of the operating shafts were upcast, the downcast air entering the mine through raises or some adjoining mine.

Since that time the following fans have been purchased for installation and replacement:

1910.............. 2   1914.............. 1   1917..............22
1911.............. 7   1915.............. 5   1918.............. 5
1912.............. 3   1916.............. 5   1919..............25
1913.............. 8

From 1910 to 1916, inclusive, 31 fans of Buffalo Forge, No. 11, 13 and 15 single-inlet Sirocco type were installed and, with a few exceptions, all operating shafts were turned downcast. With these fans, approxi-

mately 1,500,000 cu. ft. of air, or 300 cu. ft. per man, were being exhausted from the mines. Although plans were made in 1914 to increase the ventilation by enlarging air shafts and replacing the single-inlet Siroccos with double-inlet reversible Siroccos, it was not until early in 1917 that this replacement program was begun. This delay was caused by the inability of plants to furnish fans and motors, and the extensive work of enlarging and lining shafts and drifts to carry the air. The installations or replacements made during the period 1917 to 1921 were as follows: 22 in 1917, 5 in 1918, and 25 in 1919 to 1921. The result has been to increase the amount of air being exhausted from all mines to over 3,000,000 cu. ft. per min. in 1921. The single-inlet fans so replaced have been transferred underground to serve as booster fans.

A. S. Richardson began his work with the Safety First Department in July, 1915, and during 1916 and 1917 made extensive reports on ventilation conditions and smooth surfacing, when it was decided to install the double-inlet Sirocco fans. In February, 1918, his work was confined to ventilation only, as ventilation engineer, and much credit is given him for the present satisfactory condition of the mines. The Anaconda company has given all the attention possible to the ventilation of its mines, realizing that this factor is of the greatest importance to successful operation.

Mine Fires and Hydraulic Filling.—During the past thirty years, mine fires in Butte have required more or less continuous attention. Previous to 1907, fire fighting, without self-contained breathing apparatus, was a particularly hazardous occupation, but since that time the development of the oxygen helmet to the present-day type has eliminated most of the danger, so that during the past ten years, fires have either been extinguished or controlled in a short time, without greatly affecting ore production. Furthermore, although this work has entailed considerable expense, the recovery of additional copper from the mine-water precipitating plants has returned a large part of the expenditure.

The fire and gas territory from 1906 to 1917 was between the 600-ft. and 1200-ft. levels and of considerable lateral extent, including parts of the Leonard, Tramway, West Colusa, and Minnie Healy mine workings, the stopes of which were connected. In the meantime, stoping was continued on all sides from the 1300-ft. level down to the 2000-ft. In February, 1917, the fire broke through the pillar of ground below the 1200-ft. level and, in a short time, it was necessary to bulkhead all levels to the 2000-ft. Realizing that unless some different fire-fighting method was adopted a pillar of several hundred feet would be required under the 2000-ft. level for protection below, it was decided to reclaim all of the fire territory by tailing filling.

The direct supervision of this work was given to C. E. Nighman and H. J. Rahilly, at different periods, in coöperation with George Moulthrop

and John Bartlett at the Tramway mine, Karl Krueger at the West Colusa mine, and George Bennett at the Leonard mine.   E. M. Norris has been assistant general superintendent at these mines most of this time.   To all of these men, and to those under them, much praise is due for the results so far attained in conducting this operation.   At this time, there is no doubt but that all of the territory affected will be reclaimed. Two-thirds of the filling is now completed and the operating officials of the company are satisfied that it will be entirely successful in solving the problem.

Underground Fire Prevention.—The necessity of carrying out the fire-prevention program is one of great importance.   Protection by fireproofing with gunite has solved the question to a great extent.   An added risk, although a necessary one, is the great volume of air required for ventilation.   In case of fire, the smoke and gases are quickly drawn through the mine, and the only safeguards are fireproofed shafts, surface buildings, tunnels and electric installations.   With fireproof downcast shafts and main stations in conjunction with reversible surface fans, much of the danger is eliminated.

Power Distributing System for Deep Metal Mines, Electric Signal Installations, and Electric Haulage Systems.—Under the supervision of C. D. Woodward, as electrical engineer, the Anaconda company has been standardizing all of its electrical equipment on the surface, in the shafts, and in the mine proper.   The improvements and new installations since March, 1918, when Mr. Woodward assumed charge, are described in three papers.   The unusual conditions to which electric equipment in mines is subject demand the most careful consideration in its proper installation and inspection.   We believe that no change is necessary in the standard established here.

Hoisting Ropes and Safety in Hoisting.—Safety in hoisting is controlled by the careful inspection of equipment and the work of competent hoisting engineers.   This company normally operates 61 hoisting engines, and, considering that serious accidents have been few over a number of years, the practices employed here must be of the best.   W. N. Tanner, chief mechanical engineer, and F. C. Jaccard, mechanical superintendent, of the company, have covered this subject thoroughly in their paper.

Stope Cost Records and Mine Contracts.—One of the most important factors of efficiency in producing ore is the recording of results in such manner that those in charge of the work may have the proper information in a concise form.   Other factors, and probably the most important, are fair dealing with the men and the devising of some method of working by which individual effort can be rewarded.

The contract system described in this paper has been satisfactory for three years and is used only in work where the conditions are fixed, and not variable.   During that period, there have been given over 69,000

contracts, which represents millions of tons of ore in production. The success of this method is due to the hearty support of those in charge of the operations and to the general satisfaction among the men.

Organization of Mine Sampling.—In no other mining district is the correlating of mining, geology, and sampling more necessary than in the Butte district, where all the variations of banded structure in the old east and west veins, the numerous fault veins, and the finely mineralized areas of granite present the problem of what is waste and what is ore. While in most of the veins values may be determined by observation alone, each mine has development drifts and stopes where accurate sampling alone can prove the presence of ore. By the system described by Messrs. Daly and Linforth each branch of the operating department is provided with daily, weekly, and annual records, to assist in properly conducting operations and recording the necessary information.

Handling Ore in Mines of Butte District.—The methods of handling ore in mine stopes, the system of underground haulage, and the size of drifts and equipment are governed by local conditions, such as the size of orebodies, percentage of waste broken with ore, the location of workings—whether in newly opened territory or in an old section of the mine—and especially on the character of the general structure of the rock mass. The latter factor governs the size of drifts, laterals and shafts and, necessarily, the equipment that can operate in openings of the size used. One condition that limits the stoping methods to those used is that the contour of the surface must be held intact because of the network of railroads, surface buildings, and roads that serve the many mines of the district.

Mr. Tunnell's paper describes the general practice. Whatever variations there may be in the different properties are being eliminated as the work of standardization advances.

Bureau of Safety of Anaconda Copper Mining Co.

By C. W. Goodale* and John L. Boardman,† Butte, Mont.

(New York Meeting, February, 1922)

THE motives behind the organization of the Bureau of Safety were twofold. First, there was the policy of the company toward its employees, which was one of fairness and consideration for their welfare and which entails the prevention of injuries; second, in order to prevent injuries, a study of their causes was essential, and this study could be made only by a thorough systematic classification of accidents, which required a special department to form and carry out the plans.

The Bureau of Safety was organized to begin operation Jan. 1, 1914, but practically all of the first year was spent in collecting data, making plans, etc. This department is semi-independent of others and reports directly to the president and general manager. It consists of the president of the company and the managers of the following departments: Mines, Washoe Reduction Works, Great Falls Reduction Works, Coal department, lumber department, B. A. & P. railway; and the chairman of the Bureau of Safety.

Working under this group is the General Safety Committee consisting of the manager of mines, as chairman, the general superintendent, two assistant general superintendents, and the heads of the following departments: mechanical, geological, purchasing, sampling works, and framing plant; and also the safety engineer.

Operating under the General Safety Committee are the group safety committees, consisting of the Anaconda group of mines, Boston & Montana group of mines, zinc group of mines, Washoe reduction works at Anaconda, B. & M. reduction works at Great Falls, coal department, lumber department and the B. A. & P. railway. Each of these group safety committees was in charge of the superintendent or manager of the group represented and the committee consisted of the foremen of the mines or works that the committee served, as well as the safety engineer.

These foremen and superintendents had met regularly for the purpose

* Chairman, Bureau of Safety, Anaconda Copper Mining Co.
† Safety Engineer, Anaconda Copper Mining Co.

of discussing costs, efficiency, and methods; therefore it was only necessary to add at these regular meetings the discussion of safety matters.

COLLECTION OF ACCIDENT DATA

On the completion of the work of organization of the new bureau, an inquiry as to the causes of accidents was made. In former times, the matter of reporting injuries had been somewhat neglected and it is thought that many cases of accidental injury occurred that were unknown to the company officials, unless the injured filed a claim for personal injury damages.

STATISTICAL BASE

It is apparent that accident-frequency statistics based on the number of men employed or on the number of tons of ore or pounds of copper produced would be inaccurate. It was thought that a base should be adopted that would show the number of accidents as compared to the actual hazard of the work. Thus, if a man works an hour he is subjected to a one-hour accident hazard and if he works a full shift he has a shift accident hazard, regardless of the number of workmen on the payroll, or whether he produces more or less than the rest.

It was decided, therefore, to adopt the shift as a basis upon which to compute accident rates and ten thousand shifts was made the unit, partly because it is an easy figure to work with in the decimal system and partly because it is probably near the maximum number of shifts that a man might work underground in a lifetime. An accident rate thus based shows the chances a man has of being injured in a lifetime of work.

CLASSIFICATION OF ACCIDENTS

In studying the causes of accidents, it was found necessary to require that every injury, regardless of how slight a nature, should be fully reported and these reports assembled and the accidents charged against the cause to which they belonged. This also enabled the compilation of frequency and severity statistics. It was evident that a large percentage of accidents would be due to falling ground and rocks; accordingly Form No. 4 (Fig. 1) was designed, in which the accident causes were divided into two general heads, Falling Ground and Other Causes. The first head is subdivided into Drilling, Barring Down, Rock Falling Down Pile or Slides, Slabs or Rock from Back or Sides, Rocks Falling Through Floors, Rocks Falling in Manways, and Rocks Falling in Shaft. The heading Other Causes is divided into sixteen subclasses with one blank column for the insertion of any new cause that may need attention. These subclasses are as follows: Men Falling Through Floors; Men Falling in Chutes; Men Falling in Manways; Loading from Chute,

FIG. 1.

ANACONDA COPPER MINING COMPANY
BUREAU OF SAFETY
CLASSIFICATION OF PERSONAL INJURIES

June 1921

FIG. 2.

or Skip Pockets; Blasting; Falling Timber, Tools, or Material; Nails and Spikes; Ore Cars and Trains; Handling Timber, Tools, Pipe, Rails, etc.; Dumping Cars; Injured on Cages; Working in Shaft; Flying Rock while Breaking; Handling Rock; Slipping; Miscellaneous. The sheet also contains columns for totals and for days lost due to injury.

When the injury is classified from the accident report, an index figure is used to indicate the days lost and for surface accidents. Thus, if an accident occurs on surface and the injured man lost no time it would be recorded with the figure one, with a small cipher above, and Sur written after (°1 Sur); or if serious, (1_1^5 Serious). At the bottom of this sheet we have space for charging the accidents against the foremen, assistant foremen, and bosses.

At the close of the month the record is totaled and transferred to Form A (Fig. 2), which contains columns for the same classification as well as Surface; Falling Ground; Other Causes; Days Lost; Lost no Time; Shifts Worked; Fatal; Serious; Slight; Total; and Serious Accidents per 10,000 Shifts. There are also forms for reporting accident, Fig. 3; for hospital identification, Fig. 4; return to work records, Fig. 5; and ambulance drivers' report, Fig. 6.

CARE OF INJURED

Injured employees are taken care of under the hospital contract system, *i.e.*, the company has a contract with two of the local hospitals whereby, in consideration of one dollar per month deducted from each employee by the company and paid by it to the hospitals, the latter agree to treat all sickness and injury to the employee during his period of employment, except venereal cases. The company furnishes transportation to the hospital, and maintains two automobile ambulances with drivers in constant attendance.

FIRST AID

It soon became apparent that much suffering and a prolongation of disability due to injuries was caused by the lack of immediate attention; for, in each case, regardless of the facilities for getting a man to the hospital some time must necessarily elapse and in many cases the injured man must be handled considerably. The danger lies chiefly in the failure to check excessive bleeding, treat shock, immobilize fractured bones, and guard against infection.

As it is impracticable to maintain a nurse, or an emergency hospital, at many of the mines the scheme of general first-aid training was adopted. In 1914 and 1915, the Bureau of Mines began the work of training first-aid men and, in 1916, the company added a mine-rescue and first-aid department to its Bureau of Safety. The Bureau of Mines and company

P..871-5-17-18-50M   ORIGINAL

ANACONDA COPPER MINING COMPANY

_____MINE _____191

Report of Personal Injuries to Employes or Other Persons, Butte Mines

1. Name. Residence, Street and Number and Occupation of injured person, and state whether married or single, and give age. Also give Card Number.	
2. How long engaged in the capacity in which he was acting when injured, and how long in employ of company.	
3. Date and hour of accident.	
4. Was it clear or stormy, daylight, dark or moonlight?	
5. Place of accident, if underground, give level, floor and number of place.	
6. Names of Foreman, Shift Boss and Superintendent in charge.	Foreman / Shift Boss / Superintendent
7. If party injured was working underground, give name of his partner, and names of men working on opposite shift.	
8. Cause of accident. If it was the fault of tools, machinery or material furnished, say so. If fault of any employe, say whom and why.	
9. Description of Injuries	
10. Name and address of surgeon called and what was done with injured person.	
11. What does injured person say was the cause of accident and who, if anyone, does he blame for it?	
12. State fully any further information you can.	

(Enter Names on Other Side of This Blank of Persons Who Witnessed Accident)

FIG. 3.—ACCIDENT REPORT BLANK.

18 NAME OCCUPATION AND ADDRESS OF EVERY PERSON WHO WITNESSED ACCIDENT. OR CAN GIVE ANY INFORMATION REGARDING IT.

NAME	OCCUPATION	RESIDENCE (Give Street and Number)

(SIGN HERE)_____

(OCCUPATION)_____

(ADDRESS)_____

DATED_____ 191_____

FIG. 3a.—BACK OF ACCIDENT REPORT BLANK.

instructors have trained more than 2000 men to give first-aid treatment to the injured. The first-aid department has established a standard equipment of first-aid materials at the mines so that it is seldom that an employee receives even the most trivial injury without receiving immediately first-aid treatment. Hospital records have shown that, in certain periods when such records have been kept, the number of cases of infection of wounds were reduced more than 50 per cent. because of first aid; and individual cases are numerous where life has undoubtedly been saved by the prompt checking of severe hemorrhage and by the use of artificial respiration.

The first-aid work has also served the purpose, to a great extent, of popularizing the safety work. In May, 1915, a first-aid contest was held in which sixteen teams of six men each took part. During the summer, a series of contests was held and the winning team was given a trip to San Francisco, Calif., to take part in the national contest in conjunction with the Panama-Pacific Exposition. The following year, other contests

F. 866 - 2-14-18—25m **HOSPITAL CERTIFICATE A No. 8946**
ST JAMES' HOSPITAL

BUTTE, MONTANA   Date_____ 191___

MR_____has been furnished
attached certificate which entitles him to treatment at St.James' Hospital.

Nature of Illness_____

Nature and Details of Injury_____

Level_____Stope_____Crosscut_____Raise_____Drift_____

Explanation_____

Names of Partners_____

Shift Boss_____Dept. Foreman_____

Names of Witnesses_____

_____Mine_____Timekeeper

Attachment to Certificate A No. 8946
St James' Hospital

NOTICE - Treatment will be given for all sickness and injuries occurring at the mines while in the employ of this Company, not arising from immoral habits or contagious diseases.

No Prescription Filled at the Hospital Unless Prescribed by a Doctor of the Staff

Butte, Montana_____191___

Mr. _____

Nature of Illness_____
or
Nature of Injury_____
_____Mine

Per_____   Timekeeper
DOCTORS' HOURS IN HOSPITAL   THOSE NEEDING IMMEDIATE
      1 to 2 P.M.   ATTENTION CALL AT ANY TIME
      6 to 7 P.M.
Subscribers wishing attention at their homes, notify the Hospital and a doctor will be sent immediately

_____Physician.

FIG. 4.—HOSPITAL IDENTIFICATION BLANK.

F-878-1-18-17-10M   ANACONDA COPPER MINING CO.

Name_____ Working No. _____

Injured_____191 , at_____Mine

Resumed Work_____191 ,_at_____Mine

Occupation When Injured_____Rate$_____

Resumed Work as_____Rate$_____

_____   Timekeeper
_____Mine_____191___

Hospital Certificate No._____

FIG. 5.—RETURN TO WORK RECORD.

F.16-1-15-20-Bks.   ANACONDA COPPER MINING COMPANY
BUREAU OF SAFETY
AMBULANCE NO.1

DATE	MINE	NAME OF INJURED	NATURE OF INJURY	HOSPITAL	TIME OUT	IN	DRIVER

FIG. 6.—AMBULANCE DRIVER'S REPORT.

were held; teams were sent to Pittsburgh, Pa., in 1919, and to Denver, Colo., in 1920, to take part in the National Contests. In 1917 no contests were held because of the extensive labor troubles and war preparations of that year. However, the mine-rescue and first-aid department conducted a school giving special instruction in work in the miners and sappers division, from which twenty men graduated, nearly all of whom entered the service as sergeants and several were later commissioned.

AWARDING OF PRIZES

On April 1, 1913, the company decided to award a prize of $1000 for accident prevention; $750 was to be given to the foreman having the lowest accident rate per 100,000 shifts, and $250 to the foreman having the second lowest rate. In this contest only such accidents as disabled the injured man for more than 30 days were counted. But this offer was discontinued, for various reasons, after two years of operation. The serious accident rates for the Butte operations, including mines and surface shops, for the year this prize was given and after the Bureau of Safety was formed, are as follows:

YEAR	SHIFTS	RATE PER 10,000 SHIFTS
Injuries Involving More than 30 Days Disability		
Apr. 1, 1913, to Mar. 31, 1914...............	2,128,463.75	1.0480
Apr. 1, 1914, to Mar. 31, 1915...............	1,697,814.00	1.0896
Injuries Involving More than 14 Days Disability		
1915....................................	2,803,156.00	1.420
1916....................................	3,810,278.00	0.989
1917....................................	3,169,720.00	1.031
1918....................................	3,543,149.00	1.117
1919....................................	1,853,143.25	1.477
1920....................................	1,879,410.50	1.583

This shows that the rate for accidents causing a disability of more than 30 days prior to the organization of the Bureau of Safety was nearly as high as that for accidents of only 14 days for later periods.

SAFETY RULES

At first the Bureau of Safety paid little attention to safety rules; the general manager compiled a four-page pamphlet of fundamental rules and a copy was given to each employee. The idea was to keep down, as much as possible, the number of rules, but to enforce those made. As the work developed and accident records began to reveal dangerous practices, placards (either pictures or printed matter) calling attention to the danger were posted about the mines and works. Many of these placards have become rules in effect. Later these rules were assembled and are now ready for publication in book form; they are as follows:

TEXT OF NEW RULE BOOK

FOREWORD

It is the sincere desire of the Anaconda Copper Mining Co. to make and keep all working places in a safe and sanitary condition.

The company is engaged in the business of producing copper and any act of employees which has helped or will help make this business profitable has been and will be reflected to the advantage of the employees. The coöperation of all employees is solicited, therefore, in order that all may pull together for a safe, efficient, and profitable business.

The following rules and suggestions are the result of recommendations made by managers, superintendents, foremen, safety inspectors, shift bosses, and miners and have been carefully edited and will be followed as closely as practical working conditions will permit.

SURFACE

1. All power-driven machinery in shops and sawmills or framing plants shall be equipped with approved guards.

2. No rubbish, oily waste, or other inflammable material shall be allowed to accumulate at any point about the surface plants. Proper receptacles shall be provided to receive such waste.

3. All boiler steam plants shall be provided with two escapeways, as remote as possible from each other.

4. All overhead driving shafts and pulleys must be made easily and safely accessible to the oiler by means of proper runways and guard rails.

5. Lumber and timber in all surface yards must be so piled that it will be safe from falling, and these piles must have sufficient clearance from all tracks and roads to prevent being struck by wagons, trucks, etc.

6. All timber chutes and slides must be provided with proper signs and safeguards to warn passers-by of danger.

7. Material in warehouses must be kept in neat and orderly manner. Rubbish, such as boards with projecting nails, will not be tolerated about the warehouse and yards.

8. The ends of waste-dump and ore-bin tracks must be provided with adequate stop blocks.

9. Ore-bins must be provided with safety ropes to be used by persons going down into bins for any reason.

10. All hoist rooms must be kept in a neat and orderly condition.

11. No person, except employees on business, shall be permitted to enter any hoist room.

12. No person shall talk to or otherwise detract the attention of a hoisting engineer while on duty.

13. All electrically driven shop machines must be provided with safety signs "DO NOT START, MEN WORKING ON MACHINE," and such signs must be displayed at the control switch of such machinery when repairs are being made, or when the machine is stopped for oiling.

14. No hoisting engineer shall move a cage upon any other signal than that of the regular shaft bell.

15. All turn sheets and floors about shaft collars must be kept in a smooth and orderly condition.

16. No tools, timber, or other loose material may be placed closer than 10 ft. to any shaft, where such material is likely to cause stumbling or is likely to be pushed into the shaft.

17. All shaft collars must be provided with proper guard rails or gates and these must be in place to guard the shaft at all times except when the cage is actually spotted at the collar.

18. Proper overhead platforms must be provided at all shafts using skips to prevent falling rocks injuring persons on the turn sheet.

Shafts

1. All hoisting ropes, cages, kingbolts, and safety catches must be regularly inspected by a competent person and must be regularly oiled and kept in proper condition.

2. All guides and shaft timbers must be inspected daily by a competent person.

3. Guard rails and electric lights must be provided at all underground stations, skip chutes, etc.

4. No tools or other loose material must be left closer than 10 ft. from the shaft on any underground station.

5. Ladders in shaft manways must be properly placed with not more than 30 ft. between landings.

6. Men working in shafts shall have a suitable covering to protect themselves from material falling down the shafts. They shall instruct all hoisting engineers on duty at the time with respect to the place and nature of their work, so that the cage will not be let down on them. They shall have their working platforms of sufficient size and strength to safely carry on their work.

7. In shafts, winzes, or raises where two or more crews are working, one crew above another, there shall be a bulkhead between each of the crews strong enough to stop any tools or other material that may fall from men working above them.

8. All cages must be equipped with doors and same must be closed when one or more persons are on cage.

9. In no case should men and material, other than that which is carried by hand, be carried on the same deck of a cage.

10. No open hook shall be used with a bucket in hoisting, but only some approved safety hook or shackle hook.

11. In no case shall a cage, skip, or bucket be lowered to the bottom of the shaft when men are working there, but must be stopped at least 15 ft. above the bottom until the signal to lower farther has been given by one of the men at the bottom of the shaft. (This rule does not apply to shafts less than 50 ft. in depth.)

12. Every underground station shall be protected by a substantial guard-rail, chain, or gate and same shall be closed at all times when the cage is not at the station.

Underground Stations

1. No person except station tenders are permitted to ring shaft bells and all underground stations shall have station signal and warning signs with reference to shaft bell conspicuously posted.

2. Scuffling, crowding, and horse-play about shaft stations is strictly prohibited.

3. Stations must be well lighted and kept in a clean and orderly condition.

4. All stations must be provided with safety screen doors of 1-in. mesh material.

5. Ore-pockets must be guarded by proper grizzlies of not more than 12 in. space and must be provided with suitable cover to be used when no dumping is being done.

DRIFTS AND CROSSCUTS

Timbering

1. All drifts used as haulageways, for motors or tramming, shall be wide enough, at least on one side of drift, to offer refuge to men while cars are passing.

2. Timbers squeezing in on track should be repaired at once.

3. The back of timber sets should be kept lagged tight and side lagging used where necessary.

4. When drifts and crosscuts are not timbered, backs and sides must be kept barred down.

Trackwork

1. All tracks must be kept in proper condition to prevent derailments, and any defect or obstruction to track repaired immediately.

2. Rock should not be allowed to accumulate along the track. All curves and turnouts must have a proper clearance.

Tramming

1. Trammers should be cautioned to pay attention to obstructions to avoid accident, and to keep their fingers off top and side of cars when tramming, where there is danger of catching fingers on projecting sides or chutes.

2. Timbers or other material should not be piled along track used for tramming closer than 2 ft. from rail.

3. No cars should be left in places where they will unduly restrict air currents or cause collisions.

4. All dumping places must be protected when not in use.

Motors and Cars

1. All underground motors shall be provided with a gong.

2. No one except those employed thereon is permitted to ride on motors or cars.

3. Do not place long tools or material on motor or cars, use a truck.

4. Powder must be trammed by hand to magazine.

5. Do not push loaded trucks ahead of motor.

6. Timber or other material must not be piled on sides of motor or along haulageways.

Electric Wiring

1. Trolley wires should be boxed in all places, no matter how high above rail. Boxing should extend to 3 in. below trolley wire.

2. Do not handle live wires, shut off the power when repairs or extensions are to be made.

3. Never close a switch without full knowledge concerning the circuit and the reason for the switch being open.

Handling Material, Loading and Dumping Cars

1. Men are forbidden to carry tools, pipe, rails, etc. upon their shoulders in any drift or crosscut where electric wires are installed.

2. Use a side board when loading from a chute and trim all loaded cars so that they will go under all chutes.

3. Where side-dumping is done, all mine cars shall be equipped with a safety attachment to prevent the cars from falling back on the dumper.

Powder Magazines and Fuse Stations

1. Must be kept dry and well ventilated and free from rubbish.   Should have electric lights wherever possible.

2. Powder and caps should be kept at least 100 ft. apart and not on a haulageway.

3. Powder sacks must be provided for and used by miners.

4. No powder, caps, or fuse shall be left in any stope, raise, drift, or other place except in the powder magazine or fuse station.

5. Powder magazine should be fireproof where possible and must have no electric wires closer than 10 ft. from the door.

6. Use only wooden mallet in opening powder boxes.

Toilet Cars and Sanitation

1. Toilet cars should not be set in a main air passageway.

2. Doors should be kept closed when not in use.

3. Toilet cars shall be sent to the surface each day for proper cleaning and disinfecting.

4. Underground stable shall be cleaned each day and drippings in waste taken to the surface.

5. Do not throw refuse food around a working place or on the sill.

6. Dump carbide lamps in a car, if possible; never practice dumping it in the same place around manways, stations, or other working places.

7. Do not allow water to accumulate and become stagnant around working places.

8. Pure drinking water shall at all times be provided.   Tanks and water kegs should be provided with locks and adequately protected from contamination by dust and from promiscuous drinking from open vessels.

9. Drinking fountains should be installed on the station of every working level where it is convenient to pipe city water.

10. All main passageways, whether for air or travel, should be kept sprinkled. By keeping down the dust, the air is purer and very often cooler.

RAISES

Timbering

1. Stand timbers as soon as possible and block carefully.   Lag sides of manway set where necessary.

2. 10-in. grizzlies should be used, with openings not over 12 in. wide.

3. Grizzlies are to be raised as fast as chute is lined and in no case should grizzlies be more than one floor below shoveler.   Grizzlies should not be removed from one floor until they can be relaid on the next.

4. When drilling from a staging in raises, the grizzlies should be lagged over.

Ventilation

1. It should be the aim of every man to help the mine ventilation in every possible way.

2. Do not leave cars or material in such a position as to obstruct needed air currents.

3. Do not tamper with or injure fans or fan motors.

4. Do not injure or otherwise derange ventilation tubing.

5. Do not blast with fan tubing too close, take down the lengths near the blast and replace them after the blast.

6. When a door is opened close it after you, unless otherwise instructed by the shift boss.

Blasting

1. Do not leave powder or primers in the stope.
2. Guard all approaches so that no one else can walk into the blast.
3. Bosses should notify men when they are close to other workings, so that men working therein will not be in danger from blasts.
4. Use tamping in every hole.
5. In tamping drill holes, use only wooden rammers. Tamping by strokes is forbidden; only direct steady pressure is permissible.
6. Miners must count their shots and report any missed holes at office when going off shift.
7. Working places with missed holes should not be entered for at least 30 min. from time of spitting and then any missed hole must be fired before any work is done around the face. Do not attempt to extract the charge from a missed hole but insert a fresh charge and fire it. Bosses should notify men when they are close to other workings so that men working in them may be warned at blasting time.
8. Do not blast without giving proper warning and guarding all approaches. In loading holes, do not leave powder and primers together at the face.

Holing

1. In holing into workings that are used for travel, all directions leading to the place to be holed shall be guarded during the blast.
2. Do not hole a raise directly under a sill, it should be offsetted one set in hanging.
3. Install a man catcher as soon as the timbering is finished.

Manways

1. Use care in passing under manways and chutes, something may drop on you.
2. Must be offset on sill when starting.
3. Must have landings or offsets in all places over 50° pitch and these must not be over 30 ft. apart.
4. Use shaft ladders in wet places.
5. Keep clean and sprinkled if possible.
6. Steel should not be thrown down manways; it should be lowered with a rope or passed from set to set.
7. Ladders must be kept in good repair, broken rungs replaced as soon as possible.
8. Guard rails must be provided at all places where it might be possible for a man to fall into manway, chute, rill, shaft, or gob.
9. Must be kept free from obstructions and safe from falls of rock, by lagging the walls whenever necessary.
10. At crossovers, timber slide should have guard rail around it.

STOPES

Drilling

1. Bar down before starting to drill.
2. Do not drill in loose ground.
3. Examine the back or face and see that there are no missed holes before drilling.
4. Do not drill from an improper staging.
5. Treat your machine as if you owned it. Do not injure it unnecessarily. Keep it oiled properly.

Mucking

1. Do not work under bad ground; if in a square-set stope, see that the floor is tight over your head; if in a rill stope, keep the safety chain or rope near at hand.
2. Be careful to trim the pile and prevent injuries to hands and feet.
3. Be sure the grizzlies on chute are well blocked and guard rails in before starting to muck.

Timbering

1. Do not work under loose ground; if in doubt, timber.
2. Do not use too small timbers in a wide heavy stope.
3. Lag over each set as you put it in.
4. When in a rill, put in a stull if the walls are unsafe.

Barring Down

Examine the back and sides before going to the breast as the blast may have loosened something that was previously considered safe. Be sure that no one is under you when barring down. Always have a sharp pinch bar handy.

Floors

1. When entering a stope, the first thing should be to examine the floors. See that all have safe bearing. Never spike them. Lay them tight and wedge between the posts.
2. Clean the working floors first before barring down.
3. Keep all tools and other working material not in use stacked neatly back out of the way while working.

Chutes

1. Line up the braces and make them tight before lining the chute.
2. Chutes should always be lined to the working floor and grizzlies securely blocked. Use standard 10 by 10-in. grizzlies.
3. In 200-ft. lifts, 3-in. chute lining should be used in first 100 ft.

Timber Slides

Use 2-in. lagging for slides. Guard rails should be installed in manways on landings to prevent any one falling into the slide.

Tools, etc.

Keep an extra powder sack for a tool bag, to be used for spikes, wrenches and other small tools. Never leave spikes on the floor. When blasting leave the tools in the clear, not in the direct line of the blast. Hang up your saw.

Filling

In square-set stopes, filling should be kept up within two floors of working floor.

First Aid and Mine Rescue

All shift bosses who are physically fit must take the mine-rescue and first-aid training.

Each shift boss should have a firebug, who is also a helmet and first-aid man. These men should carry the vest-pocket metal first-aid packet at all times so as to be able to give immediate first aid.

Supplies

The safety inspector, or foreman at a mine where there is no safety inspector, shall see that the first-aid supplies and equipment are up to standard at all times.

Exits

Whenever possible, exits shall be made to other mines and surface to provide escapeways for men when such are needed and all such exits shall be plainly marked at all times at intersections, etc.

All exits shall be kept in a safe and passable condition at all times; and when an exit is discontinued for any reason, all signs directing the exit shall be removed.

All employees on each level shall be instructed by the shift boss or safety inspector in regard to the proper exits to use in case of emergency.

RULES FOR ELECTRICAL WORKERS

Warnings

Employees whose duties do not require them to approach or handle electrical equipment and lines should keep away from such equipment or lines. Don't take chances.

Clothing

Employees should wear suitable clothing while working on or about live equipment and lines. In particular, they should keep sleeves down and should avoid wearing unnecessary metal articles, celluloid collars, celluloid or metal cap visors or similar articles. Near live or moving parts, loose clothing and shoes that slip easily on floors worked upon should not be worn.

Safety Belts

Employees should not work in elevated positions unless secured from falling by approved safety belts or by other adequate means.

Safety belts, whether owned by the company or by the individual workmen, should be periodically inspected.

When a cover is removed from a manhole, the hole must be properly guarded by railing, danger sign, or red flag. An additional man stationed at the opening is often advisable.

Tools

No imperfect or defective tool should be used.

The handles of tools should be covered with rubber tape to prevent slipping and to reduce the possibility of short circuits across them. Such taping, however, should not be relied on for protection of workmen from shock. Heads of cold chisels, center punches, etc., should be occasionally dressed and not allowed to become mushroomed. Avoid the use of measuring tapes of metal or with metal woven into the fabric, also brass-bound rules and steel scales.

Circuits

Circuits should be made dead whenever possible before work is begun. Dead circuits should be treated as if they were alive. This procedure develops a cautious nature and may sometimes prevent an accident caused by another person's error.

All circuits should be tagged or lettered so that they may be readily identified.

Whenever circuits are opened for repairs, alterations, or examination, the control switch should be locked open and, where switch construction permits, it should be

padlocked.   The disconnecting switches or cutouts should also be opened as an additional safeguard against accidental closing of circuit.   The workman responsible for having the circuit opened shall place on a controlling switch a tag bearing his name and a notice that the switch shall not be closed until the tag is removed.   No person other than the workman tagging the switch shall be allowed to close such switch. Whenever it becomes necessary for the person tagging a switch to leave before the work is completed, as may be the case in a long job, he shall go to the switch accompanied by the man who is to assume the responsibility, and remove his tag; his successor shall then attach a similar tag to the switch.

Wiring

The insulation on a wire should not be trusted for protection from shock.   While the insulation may look perfect, it may have deteriorated from age or exposure so it cannot be relied on.

Grounding

Frames of motors, switch boxes, transformers, etc., must be substantially grounded.

Operating Switches

Switches should be left wide open when in open position, and fully closed when in the closed position.

Switches should not be closed in a hesitating manner or by tapping the blades against the contacts to ascertain if the current is on, but should be closed in a firm, positive manner, using sufficient force to make full contact of blades.   A switch should not be closed without full knowledge of the condition of the circuit.

Power Plant and Motor Attendants

Do not allow oil cans, tools, dusters, or wiping cloths to catch in moving parts of machines.   When passing any switchboard or machine in operation, do not touch it unnecessarily nor allow metal tools or other metal objects to touch the apparatus or its connections.

Do not wear loose flapping clothing when working about moving machinery.

First Aid

When a man has learned five simple but practical points in first aid he may at any time be able to save the life of a fellow workman.   These five points are:
1. How to stop serious bleeding.
2. How to avoid infection or blood poisoning.
3. How to splint and otherwise treat broken bones.
4. How to recognize and treat for nervous shock.
5. How and when to give artificial respiration.

Space will not permit a thorough description of how these things are to be done, but any employee of the company in Butte can get this instruction free of charge by making application at the Tramway rescue station.   The following, however, are some of the most necessary things to do:

If you see blood spurting from a wound, you may stop it by applying pressure with your fingers in the following manner:
1. If the wound is in the scalp apply pressure just in front of the ear.
2. If the bleeding is in the arm-pit apply pressure with your thumb in the hollow between the collar-bone and the neck.
3. If the bleeding is in the arm apply pressure with your fingers along the inner seam of the coat sleeve between the large muscles of the arm.

4. If the bleeding is in the leg, at or near the groin, apply pressure in front directly over the hip joint.

5. If the bleeding is below the knee apply pressure with you fingers just back of the knee joint.

You will have to hold points 2 and 4 until the doctor comes, but you may tie a tight bandage with a pad over the pressure point to 1, 3 and 5. This is called a tourniquet.

Any time the skin is broken, no matter how slightly, there is danger of infection. To avoid this you should not wash the wound or touch it with anything except sterile gauze. Do not apply any medicine or salves, except to paint the wound with iodine, and always cover it thoroughly with sterile gauze. Do not put any iodine on the gauze and put only one light coat of iodine on the wound. Let the doctor do the cleaning.

When you see a man with a broken bone do not move him more than is absolutely necessary. Examine the injury and if the bone sticks through the skin apply a sterile gauze dressing and stop the bleeding, if any, then tie the broken bone with splints, made of pieces of wood with plenty of padding, in such a manner that the broken part cannot move. If the bone does not stick through the skin apply the splints without a sterile dressing.

Nervous shock is a condition of the nervous system caused by injury or fright and is sometimes very dangerous. When you see an injured man with a pale face, trembling or talking excitedly, lay him down with his head low, cover him with blankets, keep him warm, give him fresh air, hot coffee, hot tea, or other stimulant and get him to a doctor as soon as possible.

Artificial respiration should be given only when a man is unconscious from electric shock, gassing, drowning, or an overdose of some kind of poison, such as morphine, alcohol, etc., except in some other cases under the direction of a doctor. When a man has become unconscious from any of the above causes and is not breathing sufficiently to keep him alive you should act quickly.

Do not wait for help or to carry the man any distance, except to fresh air. Fold your coat or jumper and place it under the patient's abdomen just below the stomach. Open his mouth and clean out any loose substance that might be drawn into the windpipe, pull the tongue forward, place the patient's forehead on one wrist and extend the other arm upward from the body. Kneel straddle of the patient's thighs. Place your hands on the small of the back 2 in. above the hips. Then swing forward throwing your weight on the patient. Swing back again to position 1, then back to position 2, keeping up this alternate pressure and release at the rate of sixteen times per minute until your patient revives, or at least for 2 hr. or more. If assistance is available, you should have the patient kept warm by covering, hot-water bottles, or any other means at hand, such as rubbing the limbs toward the heart to stimulate circulation.

A good idea is for you to attend the free first-aid classes and take the complete first-aid course, as it takes some practice in all of these things to reach the maximum proficiency.

INSPECTIONS

The safety engineer and his assistant made regular inspections of all mines and surface plants and reported their findings to the manager of mines. These reports were read by the general superintendent before the meetings of the group safety committee, in which the foreman and shift boss, in whose territory a defect was shown or to whom a serious accident was charged, were asked to explain the circumstance connected

with the accident or to explain why he had disobeyed the safety rules. This kept the foremen and bosses well lined up on the safety work.

Later, and after the results of the company's extensive mine ventilation program began to be felt, it was found that many things closely allied to safety needed close attention; so a safety and ventilation inspector was employed for each mine. In March, 1920, eighteen of these inspectors were added to the force. They spend all their time underground and record in a notebook all infractions of safety rules and condition of escapeways and safety signs, they see to the proper and safe handling of explosives, and note defects in ventilation systems, compressed-air leaks, defective or improper electrical installations, places where men are found working in danger or in other ways disobeying the safety rules. They also look after the first-aid equipment and see that a crew of mine-rescue men is kept available. The defects or infractions of safety rules are reported at once to the mine foreman, who takes immediate steps to remedy the condition; the safety inspector also makes a complete report of all notes taken and what has been done to remedy the defective condition every two weeks to the chairman of the Bureau of Safety. A copy of this semimonthly report is then sent to the general superintendent with any notations the chairman desires to make. The general superintendent then reads the reports before the meeting of superintendents and foremen where further comments and suggestions are made. This provides a follow-up method that either gets the results desired or proves the safety inspector's suggestions to be infeasible.

The safety inspectors assist the ventilation engineer in carrying out the details of the mine ventilation plan. They take temperature, humidity, and velocity readings and look after the proper arrangement of ventilation doors.

The reason for employing safety inspectors is that each mine foreman is assumed to have a sincere desire to coöperate in the safety movement and these inspectors help the foreman make the work effective without interfering with his numerous other duties.

DISCIPLINE

In March, 1916, labor conditions were such that it was almost a pleasure for a man to be discharged, because it gave him at least one shift lay-off. He could, however, if discharged in the forenoon go to work at another mine that night, so that discharging a man even for the most flagrant violation of safety rules, such as ringing the shaft bell, was not a penalty.

The general superintendent suggested that a lay-off of seven or more days be given for disciplinary purposes. Then, when a man violated safety regulations and was laid off, his employment card was held in the office for 7 days and without this card he could not obtain work anywhere

in the district. This system worked very well, so far as accident prevention was concerned, but for other reasons it was abandoned about 15 months later.

THE ANODE

It was early realized that, while exact enforcement of safety rules and holding foremen responsible for accidents would do much toward the prevention of accidents, the greatest factor was the interest taken by the workmen themselves. In order to get the case before all employees in the proper light the publication of *The Anode* was begun in January, 1915. This is a sixteen-page pamphlet devoted to the interest of safety and published monthly. Tables showing accidents charged to each mine and smelter department are published in each number and the employees contribute articles containing their ideas on safety. Prizes of $15, each, are awarded for the best article submitted by employees of the mines and smelters, and $5 is paid for any article, which does not win a prize, that is published.

The Anode is printed under contract and distributed free to all employees and a large exchange and free mailing list is carried; from 6000 to 10,000 copies are issued monthly. The cost of publishing the periodical, including contract printing and delivery, postage, and prizes for articles, but not a proportion of editor's salary (the editor is also safety engineer and has charge of mine-rescue and first-aid work), is about 8 cents per copy.

SAFETY-FIRST MOTION PICTURES

Supplementing *The Anode*, a complete set of motion pictures showing the numerous operations from the mining of the ore, in Butte mines, to the making of wire bar, at the Great Falls smelter, with safe practice and safety slogans interspersed and emphasized by red film, were made and have been shown free at various times to employees. This picture consists of twelve reels and about 9000 ft. of film.

Many requests have been received for a loan of the picture to other mining companies; so to make it available to the mining public, especially at times when we desired to use it, a copy was furnished the U. S. Bureau of Mines.

MEMBERSHIP IN SAFETY SOCIETIES

The company, through the Bureau of Safety, has maintained a connection with all of the national safety movements, such as membership in the American Mine Safety Association, and in the National Safety Council; it has contributed to the American Museum of Safety, and has given the utmost coöperation to the U. S. Bureau of Mines in regard to safety matters and furnishing the Government with accident data.

local mining companies, organized the day's activities and secured from the employing companies $1500 the first year, $2500 the second year, and $3500 the third. A program of outdoor athletic contests, including a first-aid demonstration of forty teams, was arranged. All the mines were idle for the day, two brass bands were procured, and everything was free to the public. An attendance of 17,000 was recorded for the field day of 1918, 25,000 for that of 1919, and 18,000 for 1920. The drop in attendance in the last year was caused by the curtailment of mining in the Butte district; the number employed by the Anaconda Copper Mining Co. was approximately 17,000 in 1918, 10,000 in 1919, and 5000 in 1920.

SAFETY WORK AT SMELTERS

The safety work at the smelters is of a somewhat different nature than that at the mines, caused principally by the fact that there is more moving machinery, the work is principally done in a different light, and the accident hazards are less variable.

Guarding Machinery

At the smelters, one of the first steps was to guard all machinery and then to induce the men, through safety propaganda, committee meetings, etc., to use the guards provided and to exercise caution in all operations. For several months, an investigator was employed to survey the smelter plants and ascertain what safeguards should be installed. He made more than 200 recommendations for the installation of safeguards, safety placards, etc. at the Great Falls plant during the first three months of his work. One of these recommendations was that safety committees, consisting of the superintendent and foreman of each department and several representatives of the men, be organized, the employees representatives serving for a limited time so as to give, ultimately, all the men in the department an opportunity to serve on the committee. Reports of the minutes of these committee meetings are made by the safety engineer and copies sent to the manager and to the chairman of the Bureau of Safety. These committee meetings have been very successful and are held regularly.

Classification at Smelters

At the smelters, incapacitating accidents have been used on account of the difficulty formerly experienced in obtaining accurate hospital reports of the length of disability; but, beginning Jan. 1, 1921, the records have been changed at the smelters to conform to those of the mines, in which only records of fatal and serious accidents are made for publication

The records of total accidents and classification of total accidents are made only for the use of foremen, safety inspectors, and bosses, as such records are much too bulky for publication in a periodical the size of *The Anode.*

First Aid at Smelters

First-aid work at the smelters has not been given as much attention as at the mines, as an emergency hospital in charge of trained nurses is maintained. These nurses answer calls at all hours of the day and night and can reach the scene of an accident in a very few minutes. However, there are always some first-aid men about who understand how to check serious hemorrhage and administer artifical respiration.

The treatment given injured men by the nurses at the smelters is, of course, supplemented by regular hospital treatment when the case is of such a serious nature as to require it.

Costs

The costs of operation of the Bureau of Safety are rather difficult to show accurately because, while the cost at the smelters has been charged direct to safety, that in the mines has, to a great extent, been absorbed as general operating expense.

SAFETY EXPENDITURES INCLUDING ALL CHARGES AGAINST SAFETY FIRST

YEAR	ANACONDA	GREAT FALLS	BUTTE MINES	(GENERAL) BUREAU OF SAFETY	TOTAL
1914.........	$ 16,118.15	$ 3,997.69	$ 1,812.41	$ 715.65	$ 22,643.90
1915.........	18,586.18	1,508.23	12,744.32	5,263.51	38,102.24
1916.........	35,698.10	3,517.45	11,448.94	17,940.19	68,604.68
1917.........	14,124.00	4,553.48	60,890.88	19,010.10	98,578.46
1918.........	14,787.78	3,156.65	64,603.94	18,144.23	100,692.60
1919.........	9,966.19	3,239.24	75,625.01	21,353.99	110,184.43
1920.........	6,382.63	2,498.29	75,223.66	26,139.32	110,243.90
Total......	$115,663.03	$22,471.03	$302,349.16	$108,566.99	$549,050.21

This total of $549,050.21 does not include the cost of such items as, off-setting raises and manways, boxing all trolley wires, isolating and making safe underground powder magazines, maintenance of escapeways, installation of grizzlies and guard rails, off-setting ladders and providing safety platforms in manways, providing safe and sanitary toilet and drinking-water facilities underground, installing safe control devices on hoisting engines, inspections of shafts, ropes and cages, providing heavier timber for floors, chute lining, etc., guards on shop machinery at Butte mines, equipping mine cars with stop blocks to prevent their coming back on the dumper, bonus wages for firebugs and helmet men wages of helmet men while training, etc.

Conclusion

It is fortunate that the originators of the Bureau of Safety did not proceed on the basis of cost alone, for it is feared, though not definitely known, that unless the value of safety work from the standpoint of morale is considered, the balance would not show favorably. However, it can be definitely stated that considering costs and balancing against them all the various values and advantages of the safety campaign, the department has paid immeasurably, and we recommend to anyone desiring to go into the safety work that the key to success is sincerity on the part of those so engaged; not from a purely financial standpoint but rather a desire to alleviate the suffering and minimize the loss to workmen due to industrial accidents.

Ventilation of Butte Mines of Anaconda Copper Mining Co.

By A. S. RICHARDSON,* E. M., BUTTE, MONT.

(New York Meeting, February, 1922)

THE conditions that make necessary the mechanical ventilation of the Butte mines of the Anaconda Copper Mining Co. are due to a number of causes, all of which are incidental to the depth at which mining operations are now carried on. The main object to be accomplished, of course, is the reduction of the temperature and humidity of the air in the working places. Ventilation fans, both surface and underground, have been used for many years, but increasing depth has made the problem much more difficult. Further improvement in both ventilation equipment and methods, therefore, became necessary; and it is the purpose of this paper to describe the work planned and done under the improvement program.

The mines of the district have been frequently described in various technical publications, so a description is unnecessary here. From a mine-ventilation viewpoint, compared with coal-mine ventilation systems, the problem is complicated by the fact that the workings to be ventilated are situated on a large number of steeply dipping veins which frequently intersect and are faulted, and also by the fact that the extent, or existence, of orebodies is not definitely known in advance of actual development work. For these reasons ventilation work cannot be planned in advance of actual operations, nor can such regular systems of ventilation as are used in working the more uniform and continuous bodies of coal be developed.

When operations are conducted at a normal rate, about 10,000 miners are employed on two shifts, so that about 5000 men will be underground at one time. But as ventilation is necessary to reduce high temperature and humidity due, mainly, to natural causes, the quantity of air per man per minute is not a governing consideration in determining ventilation requirements. Most important among the numerous sources of heat and humidity in the mines are: heat generated during decay of mine timber; heat and humidity given off by mine rock and water; heat generated by oxidation of sulfide minerals; heat given off by electrical equipment; heat generated by mine fires (in certain localities).

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HEAT GENERATED DURING DECAY OF MINE TIMBER

The heat generated during the decay of mine timber is a well-recognized factor. Its importance, however, is often underestimated because the temperature increases slowly and at a fairly uniform rate throughout the mines, because of the presence of timber in the greater part of the workings. In certain places, notably in old workings in the upper levels where there is no air circulation, temperatures greater than 100° F. are found, which seem to be due to this cause alone. Approximately 100 million board feet of timber were used in the mines during the year 1918, and, the writer believes, a greater heating effect is due to the decay of this timber than to any other single cause.

The process by which this heat is generated is often spoken of as the "oxidation of mine timber." In downcast shafts, where there is a rapid movement of fresh air with a maximum oxygen content, little or no difficulty is experienced from the decay of the shaft timbers. The process is most rapid in places where there is high humidity and little air movement, and in the air courses through which the more impure air leaves the mines. Although absorption of oxygen and formation of carbon dioxide occur, the term oxidation does not seem appropriate as the presence of a maximum oxygen content in fresh air does not exert an active influence. In fact, the heat is generated during bacteriological changes attendant on organic decay.

HEAT AND HUMIDITY GIVEN OFF BY MINE WATER AND ROCK

On account of the cooling of the rock by ventilation, it is difficult, if not impossible, to get any record of rock temperatures that is representative of natural conditions. The water entering the drifts first opened in developing the 3200-ft. level of the Steward mine and the 3400-ft. level of the Original mine, generally had a temperature of about 104° F. (40° C.). Ground temperatures in a vein to the north of the Steward shaft, on the 3800-ft. level, taken by inserting a thermometer into drill holes, were uniformly close to 110° F. (43.5° C.), but water flowing from this vein registered 113° F. (45° C.). The rate of increase in temperature with increase in depth does not seem to be constant throughout the district, but roughly approximates 1° F. (0.55° C.) for every 100 ft. (30.5 m.) decrease in elevation.

HEAT GENERATED BY OXIDATION OF SULFIDE MINERALS

Oxidation of sulfide minerals has greatest effect where there is a circulation of air through dry filling in old stopes. Velocity of air movement may run as high as 400 ft. per min., and temperatures may be as high as 100° F., or slightly higher. The relative importance in effect on

mine ventilation is hard to estimate. In some mines, it is practically impossible to prevent such a circulation in certain areas, but only a small number of working places are affected.

HEAT GIVEN OFF BY ELECTRICAL EQUIPMENT

Electrical equipment in the Butte mines, such as pump and fan motors, consumes, approximately, 7350 kw. In addition, there are 127 haulage motors, each of 20 hp., and transformers of 4500 kv.-a. capacity. Assuming 80 per cent. efficiency for the first item, 500 hp. effective load and 50 per cent. efficiency for the second item, and 97 per cent. efficiency for the third, the heat dissipated in the mine amounts to 85,000 B.t.u. per min. This is sufficient to heat 500,000 cu. ft. of dry air by 13.7° F.

At one time when it was necessary to draw air from the downcast High Ore shaft to ventilate a pump station, the air being returned direct to the shaft, sufficient heat was given off by the pump motors to raise the temperature of the whole volume of shaft air, approximately 75,000 cu. ft. per min., 12° F. There are three main pump stations at the High Ore shaft and two at the Leonard shaft, so that the ventilation of both these mines is subject to a serious handicap.

HEAT GENERATED BY MINE FIRES

As the areas in which ore is broken are some distance from actual fire zones, the temperature and humidity of the air in the working places are not greatly affected by heat from the fires. The West Colusa mine, in which fire-filling work is being actively carried on, maintains one of the lowest temperature and humidity records of any of the larger copper mines in the district. The highest temperature obtaining in any of the working places, other than those used for fire filling, was, in February, 1921, 76° F., the relative humidity being 86 per cent. The average of all places in the mine was, of course, considerably lower.

In places where fire filling is carried on, the temperatures are generally higher. Necessity for maintaining air pressures that will not force too much fresh air toward the fire, and that will at the same time assist in controlling the escape of gas, creates a separate ventilation problem.

QUALITY OF MINE AIR

The quality of the air in the Butte mines was studied in 1913 and 1914 by H. M. Wolflin, then with the U. S. Bureau of Mines. A sample of air leaving the Mountain Con. mine through the Green Mountain shaft, analyzed at the laboratories of the Bureau, showed the following composition: CO_2, 0.23 per cent.; O_2, 20.2 per cent.; CO, none; CH_4, none; N, 79.57 per cent. A sample taken on the 1600-ft. level of the Leonard mine at the East Colusa air shaft, representative of air leaving

the mine, gave the following analysis: CO_2, 0.09 per cent.; O_2, 20.42 per cent.; CO, none; CH_4, none; N, 79.44 per cent.

In the working places, there was a wider variation in the analyses of the air. The CO_2 content of fifty-nine samples ranged from 0.06 to 0.60 per cent. in a stope under poor ventilation conditions; the average of all samples being 0.20 per cent. The average oxygen content of the same samples was 20.2 per cent. Composition of pure air is, nominally, O_2, 20.61 per cent.; N, 77.95 per cent.; CO_2, 0.04 per cent.; moisture, 1.4 per cent. Various regulations place the desirable CO_2 limit at from 0.20 per cent., for both factories and mines, up to 1.25 per cent. for coal mines in Great Britain.

Results obtained by various investigators indicate that the bad effects of a high CO_2 content in the air are not so great as has been generally supposed, provided that temperature, humidity, and air-movement conditions are such that excessive heat is removed from the body. In general, the air in the Butte mines is of sufficiently good quality, and only measures for the reduction of temperature and humidity, or increase of air movement, are necessary.

SURFACE AIR

Surface air conditions, as regards temperature and humidity, are given in Table 1, which was compiled from records kept by E. M. Norris:

TABLE 1.—*Temperature and Humidity of Surface Air*

	Average Temperatures			Average Humidities		
	Maximum	Minimum	Mean Average, Maximum and Minimum	Maximum	Minimum	Mean Average, Maximum and Minimum
1919						
October................	45.00	25.10	35.05			
November..............	37.00	17.40	27.30	70.00	49.20	59.60
December..............	30.50	3.00	16.75	82.10	71.40	76.70
1920						
January................	38.10	14.60	26.35	85.00	67.80	76.40
February	34.30	11.40	27.85	81.50	63.10	72.30
March.................	38.80	17.60	28.20	84.40	64.20	74.30
April..................	44.60	26.60	35.60	83.80	61.70	72.70
May...................	58.90	34.90	46.90	87.10	71.80	79.40
June.............	70.20	42.80	56.50	85.00	64.70	74.80
July..................	83.90	51.90	67.90	62.50	37.00	49.70
August................	79.70	49.10	64.40	51.80	27.60	39.70
September.............	68.90	41.30	55.10	54.70	27.30	41.00
Year average.........	52.40	27.90	40.15	75.20	54.40	64.80

According to this table the average surface temperature, the year round, is about 40° F., and the relative humidity at that temperature is about 65 per cent. The true significance of this condition is not realized, however, until the moisture content of the air is taken into consideration. At the given temperature and relative humidity, there are only 1.8 gr. per cu. ft. of air, which is equivalent, approximately, to the moisture content of air at 60° F. of only 30 per cent. relative humidity.

RISE IN TEMPERATURE AND HUMIDITY IN DOWNCAST SHAFTS

The low moisture content of the air is of greatest importance because it has a high evaporative effect when raised to higher temperatures within the mines. A good illustration of this is brought out by noting the changes in the condition of the air as it passes through the downcast shafts.

TABLE 2.—*Rise in Temperature and Humidity in Shafts*

Level	Mountain Con. Shaft			Never Sweat Shaft			Berkeley Shaft		
	Temperature, Degrees F.	Humidity, Per Cent.	Moisture, Gr. per Cu. Ft.	Temperature, Degrees F.	Humidity, Per Cent.	Moisture, Gr. per Cu. Ft.	Temperature, Degrees F.	Humidity, Per Cent.	Moisture, Gr. per Cu. Ft.
Surface...	29	67	1.40	23	87	1.20	32	79	1.500
200......	31	67	1.50				38	74	2.00
400......	34	63	1.60				41	75	2.20
600......	36	63	1.70				44	77	2.40
800......	38	68	1.90	41	60	1.80	46	72	2.50
1000......	41	75	2.10	43	64	2.00	47	72	2.60
1200......	43	78	2.40	44	64	2.00	51	61	2.60
1400......	47	78	2.70	51	80	3.30	53	63	2.80
1500......							55	65	3.20
1600......	49	80	3.10	50	67	2.80			
1800......	52	87	3.50	54	88	4.10			
2000......	54	76	3.50	53	70	3.20			
2200......	56	70	3.50	52	74	3.20			
2400......	60	68	4.00	61	68	4.00			

According to Table 2, in passing from the surface to the 2400-ft. level, the air has picked up 2600 gr. of moisture per 1000 cu. ft. The cooling effect due to the evaporation of this quantity of water is sufficient to lower the temperature of the whole volume of air and vapor by 22° F., though the fact is not directly noticeable because the heat given off by the rock surrounding these long air courses has more than offset the lowering of temperature due to evaporation.

MINE RESISTANCE

The general subjects of mine resistance and coefficients of frictional resistance to the flow of air have been quite fully discussed, but wide

variations occur in the figures given. By far the greater part of the determinations were made in coal mines, and almost nothing was available that had any bearing on the resistance of such entirely dissimilar workings as are found in metal mines. For these reasons, it seemed advisable to get more accurate information in regard to the mines directly affected by the work to be done, so resistance determinations were made of all important types of air courses.

In carrying out this work, the instruments used were: An inclined U-tube gage, supported on a transit head and so mounted that simple relations of gage readings to vertical head could be obtained up to the proportion of 40 to 1; various types of Pitot tubes; anemometers; and special orifices for recording static pressure. Of the types of Pitot tubes that were tested, that which seemed to give the best results was one in which the static pressure is admitted through a number of small holes on a circumference of the tube in a plane perpendicular to the direction of flow of the air. The special static orifices were made by piercing a pipe cap with $\frac{1}{32}$-in. holes, and enclosing the whole cap in a chamber to which the static pressure is admitted through a brass gauze. Pitot tubes with small static orifices are of no use in wet shafts, or drifts, because the water will clog these small holes.

In practice, the static orifices were placed at measured distances apart in the shaft, drift, or crosscut, where the determination was to be made and were then connected by air-tight tube or hose to the U gage on which the difference in static pressure was to be noted. Velocity readings were then made with both Pitot tube and anemometer, and friction coefficients figured from the data obtained. For this purpose it was, of course, necessary to use workings in which there was a high velocity of air movement.

The coefficient K, which expresses, in pounds per square foot, the pressure necessary to overcome the resistance offered by 1 sq. ft. of rubbing surface at an air velocity of 1 ft. per min., as thus determined, is given for the following types of mine workings. In all cases, the area used in making the calculations was the clear area of the air course inside main timber, and the rubbing surface was assumed to be that which encloses the clear area.

For rectangular shafts of two or more compartments with open timber framing, from 0.000,000,0072 up to 0.000,000,0096; variations are dependent mainly on the condition of the timber, such as lagging.

For rectangular shafts in which each compartment is a separate smooth-surfaced duct, from 0.000,000,0014 up to 0,000,000,0023.

For timbered drifts and crosscuts, about 5 by 7 ft. in the clear, from 0.000,000,0077 up to 0.000,000,0109; variations are dependent on whether the air course is straight or crooked.

For untimbered crosscuts, from 0.000,000,0031 up to 0.000,000,0044;

variations in this case are also dependent on whether the crosscut is straight or crooked.

For the manway compartment of raises, almost any figure from 0.000,-000,025 upward may be obtained, dependent on the condition of the raise.

These figures are based on gage readings that ranged from 3 to 15 in. (76.2 to 380.9 mm.) and are believed to be fairly accurate, considering the limitations of the instruments used and the character of the problem. It was possible to note the variations in resistance caused by the passage of a cage through the section of the shaft upon which the determination was made, also by a man standing and walking in opposite directions in a drift and by a mine car standing in a crosscut, the last being equal to about 40 ft. of the crosscut.

Nevertheless, as all determinations made with either Pitot tube or anemometer depend for their accuracy on the assumption that the flow of air takes place in straight lines normal to the plane in which the instrument is held, serious inaccuracies must occur because this assumption is unwarranted by fact. Obstructions, such as mine timber, changes of direction of air courses, and similar influences produce a swirling, or eddying, motion of the air, which varies both in direction and velocity at all points across a given plane. Nothing beyond an approximation in measurement is, therefore, possible, and accuracy is in the greatest degree dependent on the care used in making it.

The coefficients that have been given are, therefore, nothing more than approximations, and must be so considered. Further, other conditions, such as local restrictions of clear area of workings and abrupt turns in main air courses, may have such a bearing on mine resistance that the use of calculations based on friction coefficients must be governed by careful judgment.

SMOOTH-SURFACING OF AIR SHAFTS

An analysis of the component parts that made up the total resistance to the flow of air that was offered by the mine workings showed that the greater part of this resistance arises in the shafts. The reason for this is that the air must travel over 5000 ft. vertically, downwards and upwards, and only a small part of this distance horizontally, the latter averaging not much more than 1000 ft. All of the air must necessarily pass through the shafts, so that the velocities are high, and as the resistance varies with the square of the velocity, the resistance per unit of shaft length will be correspondingly high. During its horizontal travel through crosscuts and drifts and vertically through stopes, the air is split into smaller volumes so that velocity of movement is slower and the resistance relatively slight.

Increasing the volume of air circulated through the mines without the use of an excessively large amount of power, or the expenditure of

large sums of money for additional air shafts was, therefore, dependent on the reduction of the resistance of the shafts. The determinations of coefficients of frictional resistance indicated that the resistance of shafts with open timber framing might be greatly reduced by enclosing each compartment so that it would become a separate smooth-surfaced duct, and thus developed a method of accomplishing the desired result in an economical manner.

In order to develop fully the benefit that might be derived in this way it was decided, in December, 1916, to experiment with the Parnell shaft, which is an upcast air shaft used for the ventilation of parts of both
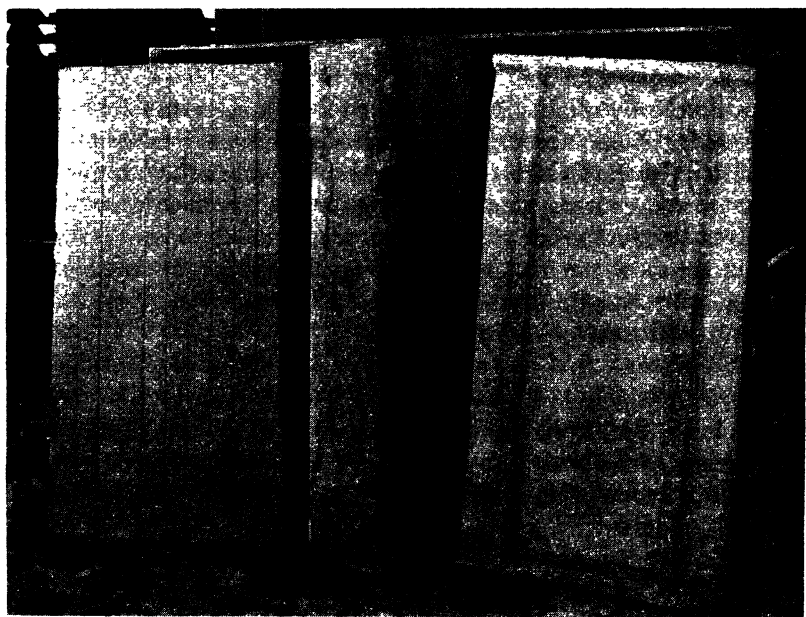
FIG. 1.—CONCRETE SLABS AND ANGULAR PIECES FOR SMOOTH-SURFACING AIR SHAFTS

High Ore and Diamond mines. This shaft has one compartment 7 by 5 ft. (2 by 1.5 m.) in the clear, and two compartments 5 by 5 ft., making a total clear area of 85 sq. ft. (7.9 sq. m.). It extends from the surface to the 2800-ft. level, and can be opened to permit the inflow of air at all stations below the 1000-ft. level.

To handle the increased volume of air that it was estimated would be obtained, the shaft was equipped with a new and larger fan, which was put into operation before any work was done toward smooth-surfacing the compartments. The results obtained before and after carrying out this work, with the fan running at the same rate of speed, and with other mine conditions essentially the same, are as follows:

	Before Surfacing	After Surfacing
Volume of air, in cubic feet per minute.........	135,000	260,000
Fan suction in inches of water.................	3.6	3.4

These results show that by smooth-surfacing all air shafts and providing adequate fan equipment, the volume of air circulated through the mines could be practically doubled. The air-carrying capacity of
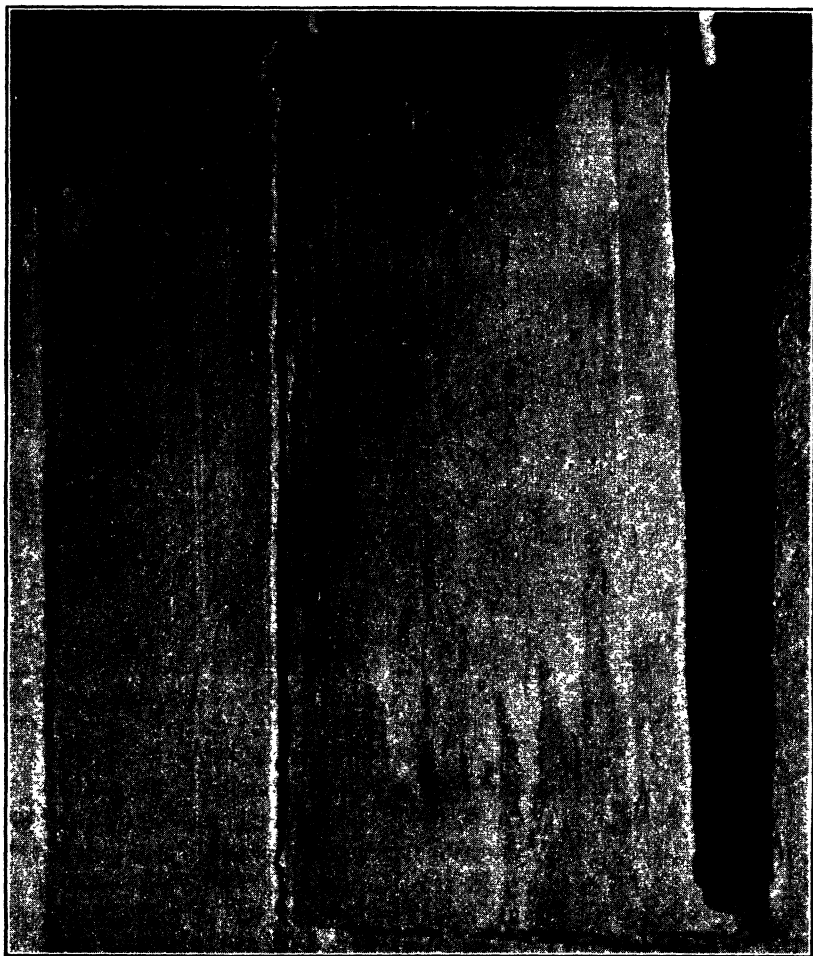
FIG. 2.—CONCRETE SLABS AND ANGULAR PIECES IN POSITION ON SHAFT TIMBERS, BUT NOT "POINTED-UP" WITH MORTAR. ANGULAR PIECES ARE NECESSARY TO FORM SUPPORT FOR SLABS, AS TOPS OF WALL AND END PLATES ARE CUT AWAY BY FALLING ROCK.

more than twenty air shafts was thereby affected, and, to that extent, a saving equivalent to opening that additional number of shafts could be accomplished. It was decided, therefore, to proceed with this work, and an entire new equipment of fans was ordered.

Originally, it was intended to do the smooth-surfacing with 1-in. sheeting, surfaced on one side, but later, in accordance with the general fireproofing plans, it was decided to use concrete flat slabs and angular pieces, precast at the surface and held in position on the shaft timbers by wedges and cement mortar. The essential features of both slabs and angular pieces, as well as their positions in relation to the shaft timbers, are shown in Figs. 1, 2, and 3. The wooden wedges serve, mainly, to hold the slabs in position until the mortar sets. In shafts surfaced with sheeting nailed to the face of the shaft timbers, the nails are destroyed by copper water so that often the sheeting falls across the shaft and obstructs the flow of air. As no metal is used in fastening the concrete slabs in position, this danger has been eliminated. Both slabs and angular pieces are manufactured at a small plant that was originally built for smooth surfacing the Parnell shaft.
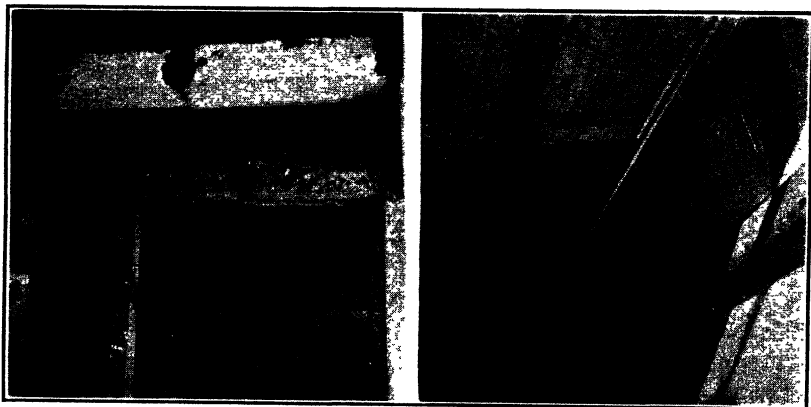
FIG. 3.—ST. LAWRENCE YARD SHAFT BEFORE AND AFTER SMOOTH-SURFACING WITH CONCRETE SLABS.

Two men working together on one scaffold will set in position the material to surface from 10 to 30 ft. of one shaft compartment in an 8-hr. shift, dependent on shaft conditions; whenever possible two scaffolds are used. One station tender and one surface laborer to mix mortar and move materials are required to keep the four men on the scaffolds supplied with materials. Since the summer of 1918, many thousands of concrete pieces have been made at the Parnell plant; six air shafts have been smooth-surfaced, and work on four more has been undertaken. No renewals or repairs to the work have so far been necessary.

A number of air shafts have wooden water boxes, which obstructed the flow of air because they were supported by bearing pieces that extended from one side of the shaft to the other. The highly acid copper water constantly destroyed the nails and repair costs were high.

Some water carries considerable iron salt, which coats the sides of the boxes and rapidly reduces their carrying capacity, and is removed only with difficulty.

To eliminate the resistance to the flow of air caused by the bearers, and the destructive action of the copper water, as well as to reduce the difficulties of removing the iron precipitate, a triangular concrete water box that fits into the corner of a shaft compartment was made. On account of photographic difficulties it was not possible to get a photograph of the box in place in a shaft, so a framing was erected to represent similar conditions, and to illustrate the manner of supporting the boxes. The upper box shown, Fig. 4, is cast with extension supports, which are a part of the box. The lower box is supported by a flange bearing on a separate bracket; this is the type used in combination with the concrete slabs. Each box fits loosely into the box below, and may be removed by simply lifting it off its support and then lowering it clear of the box below. In service these boxes have lasted several times as long as the wooden boxes and show no signs of deterioration.

Connections from air shafts to surface fans are made through concrete ducts that open into the shafts through either wall or end timber framing, thus leaving all compartments open at the collar and directly accessible for use in case of necessity. All surface fans, with three exceptions, are reversible, and are operated as exhaust fans to develop a vacuum system of ventilation. The working shafts, through which men are lowered and hoisted, are the main inlet air courses and will, therefore, remain clear of smoke in case of any fire occurring within the mine. Not only is a safer exit from the

Fig. 4. — CONCRETE WATER BOXES FOR AIR SHAFTS, WHICH FIT INTO CORNER OF SHAFT COMPARTMENT AND OFFER MINIMUM RESISTANCE TO FLOW OF AIR.

mine assured by this method of ventilation, but cables, skips, and cages are not subjected to the destructive action of the acid-laden moisture

FIG. 5.—FAN AT COLLAR OF GREEN MOUNTAIN SHAFT.
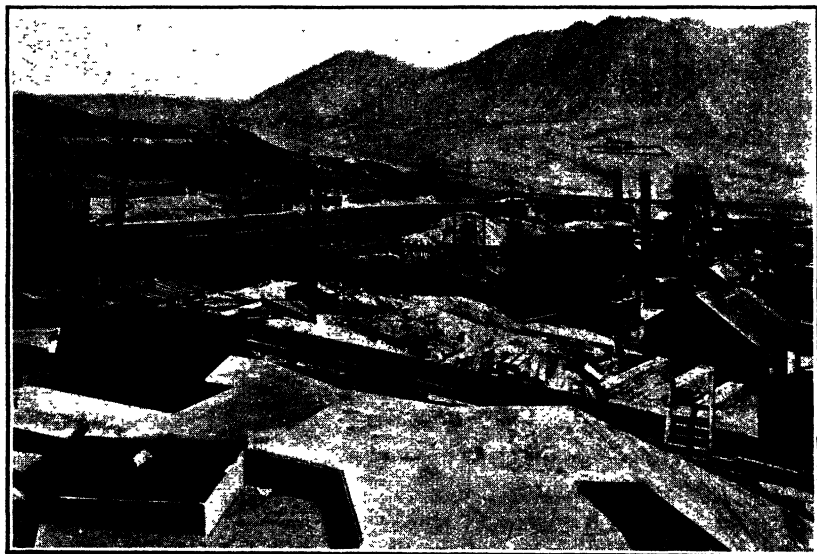
FIG. 6.—FAN AT WEST COLUSA MINE. AIR CAN BE DRAWN THROUGH EITHER, OR
BOTH, OF TWO SEPARATE AIR COURSES.

precipitated during the cooling of exhaust mine air, and working conditions on the stations are greatly improved.

TABLE 3.—*Volume of Air in Circulation in Different Mines*

Shaft	Volumes of Air in Cubic Feet per Minute			
	Old	Estimated	Present	Remarks
Emily...	35,000	60,000	45,000	Work incomplete
Moose.................	40,000	100,000	100,000	Work incomplete
Cora....................	83,000	150,000	83,000	Work incomplete
East Gray Rock..........	37,000	60,000	45,000	Work incomplete
Green Mountain..........	65,000	150,000	180,000	
Parnell..................	130,000	260,000	260,000	
Mat.....................	40,000	70,000	30,000	Work incomplete
East Steward............	38,000	38,000	38,000	
Parrot..................	54,000	200,000	180,000	Work incomplete
West Gagnon.............	100,000	175,000	80,000	Work incomplete
East End Line.......... .	60,000	60,000	60,000	
St. Lawrence Yard........	95,000	100,000	120,000	
Molly Murphy............	25,000	100,000	125,000	Work incomplete
Anaconda...............	35,000	60,000	35,000	Work incomplete
Never Sweat..............	30,000			Changed to downcast
Ramsdell Parrot..........		100,000	110,000	
Mountain View raises, No. 116...................	46,000	70,000	75,000	
Mountain View raises, No. 270...................	47,000	70,000	80,000	
West Colusa air raises......	65,000	150,000	120,000	Work incomplete
East Colusa air shaft......	47,000	125,000	85,000	Work incomplete
Mitchell............... ..	45,000	125,000	40,000	Abandoned
Liquidator...............		250,000		Work incomplete
Tramway No. 4 shaft.....	60,000	125,000	70,000	Work incomplete
Pennsylvania air shaft.....	75,000	200,000	75,000	Work incomplete
Silver Bow, No. 1........	35,000			Changed to downcast
Silver Bow, No. 2........		60,000	80,000	Work incomplete
Silver Bow, No. 3........		70,000	50,000	Work incomplete
Snohomish...............	43,000	60,000	70,000	
Nettie, No. 3............	28,000	40,000	55,000	
Emma....................	20,000	60,000	70,000	
Josephine................		30,000	30,000	
Tropic	20,000	20,000	20,000	
Poulin..................	30,000	30,000	30,000	
Alice....................	30,000	30,000	30,000	
Total air..............	1,463,000	3,198,000	2,481,000	
Total motor horsepower..	2,175	3,820		

. Table 3 shows the volume of air circulated by the old surface fans before any of the work contemplated under the present improvement

program had been done, together with the volume it is estimated will be circulated when the work is complete, and the results obtained up to the time of writing.

DISTRIBUTION OF AIR

The success of a ventilation system depends very much on the proper regulation of the air distribution; and as the location of operations is constantly changing, this is not always easy to work out to best advantage.
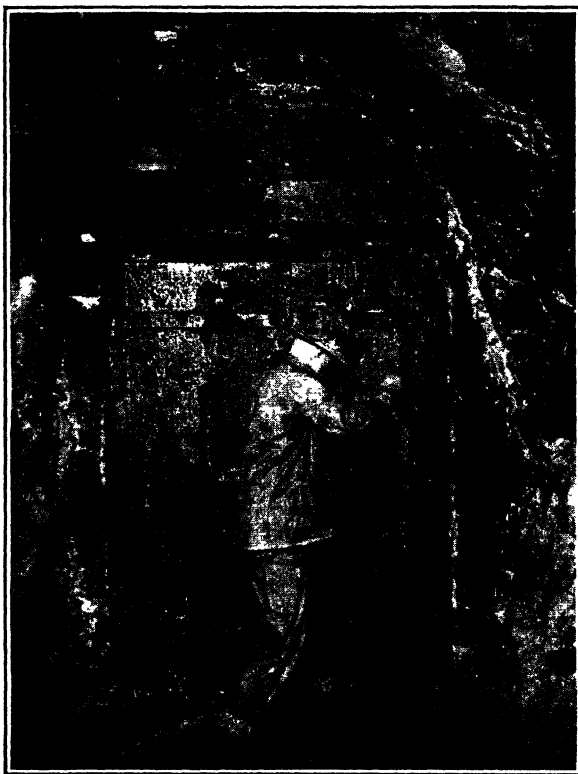
FIG. 7.—SHEET-STEEL DOOR WITH CONCRETE FRAME USED AT INTER-MINE CONNECTIONS; CAN BE OPENED FROM EITHER SIDE.

In the Butte mines, with their complex systems of veins and faults, it is necessary to provide for the ventilation of working places that are widely scattered both as to elevation and situation on different veins. In one mine, work is being carried on from the 800-ft. to the 3800-ft. levels, in all directions from the downcast working shaft. The developments of each day become the ventilation problem of the near future, and all plans are governed by the immediate necessity.

Control of the air in its flow through the mines depends, mainly, on the use of ventilation doors, which prevent circulation along lines of least

resistance, leaving as dead ends many places where ventilation is most necessary. Auxiliary to the use of doors, and as an effective means of creating circulation on the lower levels, are the so-called "booster" fans, which are generally installed near the bottom of both downcast and upcast shafts. The fans at the bottom of the downcast shafts force the air from the shaft into the lower levels, where it is generally most needed, while those at the bottom of the upcast shafts collect the air from the mine and help to boost it up the shaft.

As service requirements vary at different places, a number of different ventilation doors are used in the Butte mines. At intermine connections the standard type, shown in Fig. 7, is one in which the frame is made of

FIG. 8.—WOODEN DOORS USED ON MOTOR HAULAGEWAYS. WHEN RELEASED FROM LATCH, DOOR IS PULLED OPEN BY COUNTERWEIGHT.

FIG. 9.—THE "PENNSYLVANIA" DOOR.

concrete while the door itself is made of sheet steel. The latches are connected to operating levers on both sides of the door so that it can be opened from either side.

On motor haulageways, where there is a heavy air pressure, the common type is a single door, Fig. 8, counterbalanced by a weight and fastened positively by a latch, which can be operated by pulling a rope carried along the side of the crosscut for the convenience of motormen. Two doors, spaced about 100 ft. (30.5 m.) apart, are used to form an air lock sufficiently long to accommodate a train of ore cars.

Another type used on motor haulageways, termed, locally, the "Pennsylvania" door, Fig. 9, is a set of heavy double doors hung from inclined posts, which tend to close as the weight of each door causes it to swing downwards and toward the center. This door is practically indestructible as the impact of a motor merely serves to open it, no matter in which direction the motor may be traveling. Unfortunately, a great deal of air leaks through and under heavy air pressure it is almost impossible to prevent the doors from swinging open under surges in the air pressure.
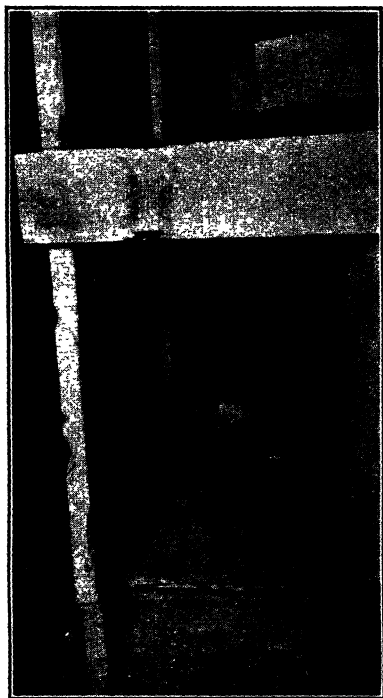
FIG. 10.—TRAP DOOR AT HEAD OF MANWAY TO BLOCK FLOW OF AIR THROUGH STOPE WHERE NOT NEEDED.

FIG. 11.—GRIZZLIES AND LATTICED DOOR DESIGNED TO OFFER MINIMUM RESISTANCE TO FLOW OF AIR THROUGH RAISE.

Ventilation doors used on levels that are not haulageways are generally of the type first mentioned, except that the rope tackle is omitted. Openings in the doors, covered by a movable slide, are used when regulators are required.

To regulate the distribution of air from the sill into different stopes manway brattices are used at the bottom of manways leading to square-set stopes, while at other places a solid trap door may be used at the head of the manway, as shown in Fig. 10. Where a large volume of air is to be passed through a raise, protection may be retained and resistance

reduced to a minimum by the use of grizzlies made of rails and a latticed door at the head of the manway, as shown in Fig. 11.

The volumes of air handled by the booster fans range from 30,000 to 70,000 cu. ft. per min., and the resistance measures from 1 to $3\frac{1}{2}$ in. of water. All of these fans are belt driven by motors of from 30 to 100 hp. A typical installation is shown in Fig. 12. Table 4 shows the location of present booster installations, and also the motor horsepower and volumes of air circulated by the fans.
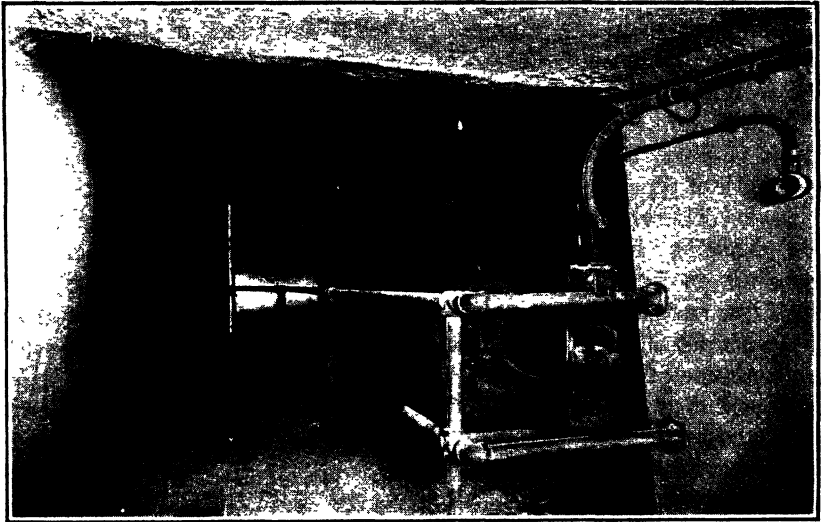
FIG. 12.—BOOSTER FAN ON 2800-FT. LEVEL OF HIGH ORE MINE.

TABLE 4.—*Booster Installations*

Location	Fan	Motor Horsepower	Volume of Air in Cubic Feet
2000 West Gagnon............	No. 11 Sirocco	75	50,000
2800 West Gagnon............	No. 11 Sirocco	75	70,000
3400 Original................	No. 11 Sirocco	75	70,000
3200 Steward................	No. 11 Sirocco	75	60,000
2800 Diamond...............	No. 11 Sirocco	100	60,000
2800 High Ore...............	No. 11 Sirocco	75	60,000
2800 Belmont................	66-in. Jeffrey	75	75,000
1500 Berkeley...............	No. 11 Sirocco	75	65,000
1000 Berkeley...............	No. 8 Sirocco	30	35,000
2200 Tramway...............	66-in. Jeffrey	75	70,000
1700 Tramway...............	66-in. Jeffrey	75	70,000
1400 Tramway...............	60-in. Buffalo	50	50,000
Total....................		855	735,000

VENTILATION OF DEAD ENDS

In confined places, such as dead ends on crosscuts, drifts, or raises, the heat generated by blasting and lights and that given off by men's bodies raise the temperature of the air. Ventilation of such places is necessary to drive out powder smoke and maintain a supply of good air and to reduce temperature and humidity. For this service small blowers are used to force fresh air from the nearest source of supply, through canvas pipes, to the working face.

Standardization of equipment has led to the selection of the following pipe sizes: For the ventilation of raises, where the space available for pipe is small and where the length of pipe is not over 250 ft. (76 m.), 8-in. (203 mm.) diameter pipe; for crosscuts and drifts, where the length of pipe is not greater than 500 ft., 12-in. pipe; for places where the length of pipe is greater than 500 ft. and to carry air into stopes, 16-in. pipe. The blower equipment for use with these pipes is: For 8-in. pipe, No. 2½ Sirocco, wheel one-third width of standard wheel, direct connected to 3-hp., a.c. motor; for 12-in. pipe, No. 2½ Sirocco, wheel two-thirds width of standard wheel, direct connected to 5-hp. a.c. motor; for 16-in. pipe, No. 4 Sirocco, wheel one-half width of standard wheel, direct connected to 10-hp. a.c. motor.

Selection of blowers was governed by the fact that, within certain limits, the pressure exerted by a blower to force air through the pipe is a function of the rim speed of the wheel, while the volume of air delivered against a given pressure is dependent on the width of the wheel. Under a given pressure, a canvas pipe will carry only a limited quantity of air. The problem then is to proportion the width of blower wheel so as to obtain maximum efficiency, considering the volume of air that the pipe will carry at the pressure developed by the blower when running at constant motor speed. Power required to drive the blower, running at constant speed, decreases with each additional length of pipe added, or increase in resistance to the flow of air and increases to a maximum when there is no pipe in place. Obviously, when the wheel is unnecessarily wide and the pipe does not carry all the air that the blower is designed to deliver at the pressure developed, there is an excessive consumption of power. In case the pipe should accidentally be torn off the blower and the overload release does not act, an unnecessarily large motor has to be provided in order to avoid burning up the motor, possibly starting a mine fire.

The combinations of blowers and pipes that have been mentioned are designed to give air velocities of 2000 ft. per min. (609.6 m.) at the pipe ends under reasonably good pipe conditions. Effectiveness of the equipment is, however, in the greatest degree dependent on the manner in which the canvas pipe is installed and maintained in good repair. No

canvas pipe that is pulled around a cap or post, and thereby had its clear area reduced to about 10 per cent. of the nominal clear area, will deliver much air at the end of the pipe; and a number of kinks produce much the same effect. Under these circumstances, the natural tendency is to blame the blower equipment; but when it is remembered that increase in volume will vary only as the square root of the pressures developed by the blowers it must be evident that little is to be gained by extra heavy pressures.

Where pipe lines are exceptionally long and correspondingly high resistance is offered to the flow of air, it is the practice to use two or more blowers, connected in series, at suitable positions along the pipe line. Under normal operating conditions about 300 blowers will be in use.

VENTILATION RECORDS AND MAPS

Inspection of ventilation conditions, together with the duty of making recommendations for their improvement, is entrusted to a ventilation and safety engineer at each mine. He is employed under the direction of the mine foreman, but acts in ventilation work in coöperation with the ventilation engineer. Repairs and renewals of canvas pipe and blowers, and the installation of doors, or other details necessary to carry out the ventilation system, are made on his recommendation to the mine foreman. Monthly reports made to the ventilation engineer include: A record of the temperature and humidity of the air at every working place in the mine; the conditions existing at all places ventilated by blowers and canvas pipe, with a description of the equipment used and its state of repair; recommendations in regard to improvements or other matters needing attention.

Details of air distribution are, of course, subject to frequent change because every raise that is holed will affect the flow of air on the levels to which it is connected, and also because the location of working places is constantly changing. Ventilation records and maps, therefore, must be adapted to meet these conditions without requiring an unduly large amount of work.

The method that has been developed to accomplish this work involves the use of blueprints of all levels of the mine, and loose sheets of paper upon which are typewritten all items of information that cannot conveniently be noted on the blueprint. The tracings from which the blueprints of the levels are made will generally be on a scale of 1 in. equals 200 ft., and will give only a bare outline of the workings. With prints at hand, details of ventilation conditions are recorded with white ink, or a solution of potassium oxalate, and the record may be brought up to date at any time merely by taking another print from the tracing and going over the distribution of the air.

As a matter of convenience in recording information on the print, the following conventional signs have been adopted:

⊙   Air moving upwards, entering the level.

⊗   Air moving upwards, leaving the level.

▣   Air moving downwards, entering the level.

⊠   Air moving downwards, leaving the level.

ō̄   Bars over the foregoing signs indicate that the air is not coursed directly to adjoining levels. The mine numbers of raises, or other working places, are given when necessary.

          Filling, timber bulkheads, or other conditions permitting the air to escape from the sill.

↠―→
18,000   Direction of flow of air, and volume in cubic feet per minute.

5-Ω-1   The first numeral is an abbreviation of the number of the level, the letter indicates that the stopping is a door, and the last number is the number of the door in reference to the particular level. Bulkheads, and brattices, may be similarly denoted by a difference in the letter.

A ventilation map, or print, of the 500-ft. level of the Nettie mine is shown in Fig. 13.

The typewritten notes on the loose leaf accompanying the prints of each level include such information as the temperature, psychrometer wet bulb, and relative humidity of the air in all working places on the level, taken monthly, and also a general description of such features as the construction of ventilation doors, or stoppings, and the purposes they serve. Typewritten sheets and blueprints of each level are kept together, and all of these records for a given mine are bound together in a loose-leaf notebook. The system has not yet been fully worked out, or applied to all mines, and modifications will probably be found necessary.

COST OF VENTILATION WORK 1916–1920 INCLUSIVE

YEAR	OPERATION	CONSTRUCTION	TOTAL
1916..............	$433,536.21	$22,123.29	$455,659.50
1917..............	459,295.45	54,430.93	513,726.38
1918..............	802,566.13	54,672.65	857,238.78
1919..............	531,308.18	223,983.30	805,291.48
1920..............	624,216.78	110,288.16	734,504.94
	$2,900,922.75	$465,498.33	$3,366,421.08
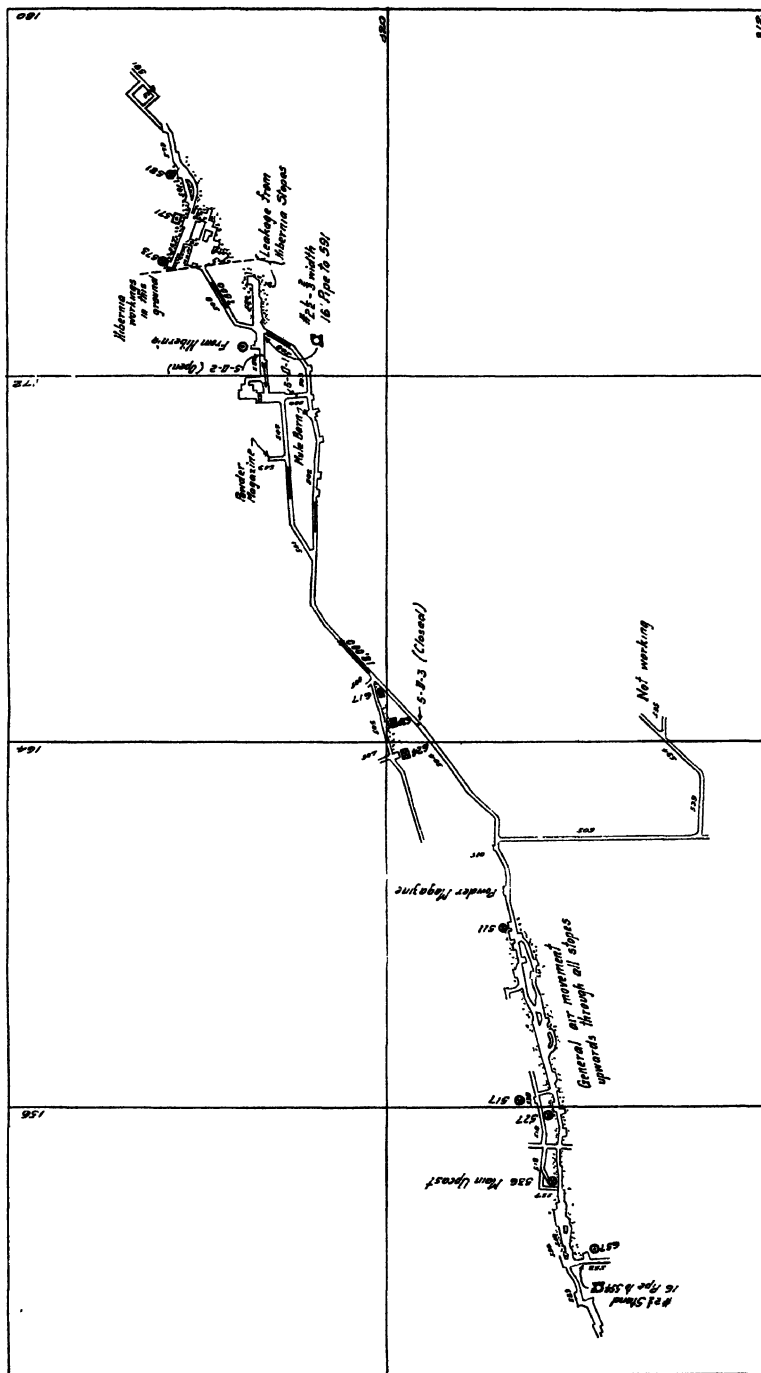
Fig. 13.—Ventilation map of 500-ft. level of Nettie mine.

In this table, operating includes all work on air shafts such as sinking, smooth-surfacing, and repairs; opening crosscuts, drifts, and raises required for use as air courses; excavation necessary for the installation of booster fans; cost of blowers and canvas pipe and installation charges for this equipment.

Construction includes fans, motors, construction charges covering new equipment for twenty-four air shafts; fans, motors, and installation charges covering ten booster fans; cost of plant and equipment for making concrete slabs. No charge is counted for the mechanical equipment of seven air shafts for which new equipment was not needed, or upon which it has not yet been installed.

RESULTS

In considering the results obtained from the ventilation work that has been outlined, it should be remembered that mechanical ventilation of the mines has been carried on for a number of years. The comparisons that will be given are, therefore, not relative to mines in which mechanical ventilation is used for the first time, but are a measure, only, of recent improvement. Unfortunately, the abnormal conditions of wartime operations, followed by the subsequent industrial depression, has delayed much of the work. Considerable time is required to carry out the work of smooth-surfacing a shaft, and at least one year seems necessary to develop the full benefit to be derived from an increased flow of air. A few instances of the results obtained in mines where the improvement program has been nearly completed can be given, however, the standard being the average of the temperature and humidity conditions of the air in the working places.

West Colusa mine: In May, 1919, the average temperature and humidity conditions of forty-six working places were: Temperature, 80.1° F.; psychrometer wet bulb, 77.3° F.; relative humidity, 88 per cent. In February, 1921, the average conditions of thirty-five working places were: Temperature, 69.3° F.; psychrometer wet bulb, 66.4° F.; relative humidity, 86 per cent. Work on a second upcast air course is not yet complete; it is expected that when this is done the volume of air circulating through the mine will be increased nearly 50 per cent.

Mountain Con. mine: In October, 1917, the average temperature and humidity conditions of fifteen working places were: Temperature, 85.8° F.; psychrometer wet bulb, 83.0° F.; relative humidity, 88 per cent. In November, 1920, the average conditions of thirty-one working places were: Temperature, 76.2° F; psychrometer wet bulb, 67.0° F.; relative humidity, 62 per cent. In October, 1917, the mine was operating at only part capacity, due to labor shortage, and in November, 1920, it was shut down. The record taken at the latter date did not include certain

places used temporarily to carry air leaving the Steward mine. It is fairly representative of average conditions, but the low humidity indicates a further probable lowering of temperature in the future. A large booster fan will probably be installed in the mine when operations are resumed.

Berkeley mine: In May, 1919, the average temperature and humidity conditions of the working places were: Temperature, 81.5° F.; psychrometer wet bulb, 80.0° F.; relative humidity, 93 per cent. In December, 1920, the average conditions were: Temperature, 75.8° F.; psychrometer wet bulb, 70.3° F.; relative humidity, 76 per cent. The work planned for this mine is not yet complete, and the mine was shut down at the time the last record was made.

Improvement of ventilation conditions is not measured by reduction in temperature and humidity alone, it depends also on other conditions. In fact, reference to relative humidity alone is misleading because 100 per cent. relative humidity is not oppressive at low temperatures although it is decidedly oppressive at high temperatures. At wet-bulb temperatures below that of the body, say 98° F., an increase in velocity of air movement is the most effective means of reducing the bad effects of high temperatures. So long as the air is not saturated, this increases the rate at which perspiration is evaporated on the skin, and as the latent heat of evaporation is high the cooling effect due to it is generally greater than that caused in any other way. The present ventilation program provides for the circulation of practically double the volume of air formerly circulated through the mines, so that there will be a corresponding increase in the velocity of air movement, and dilution of the CO_2 content.

DISCUSSION

D. HARRINGTON, Denver, Colo. (written discussion).—On page 34, the author objects to the term "oxidation" in connection with absorption of oxygen with attendant increase of surrounding air temperature in timbered regions, as there appears to be comparatively little of this oxidation where there is the maximum amount of oxygen; he believes that in timbered regions the heat is generated during bacteriological changes attendant on organic decay. Nevertheless, oxidation is undoubtedly a proper term. The action is almost analogous to spontaneous combustion of coal in piles or in mines, for which oxidation is accepted as being responsible; yet there is comparatively little, if any, bacteriological action and frequently there is a maximum supply of oxygen, though usually little air movement to carry away the temperature increase.

The coefficients of friction given on pages 38 and 39 indicate the desirability of smooth-lining main air courses in which large quantities of air must be carried at high velocity. Nevertheless, the coefficients given include not only frictional coefficients, but air resistance caused by

eddy currents from projecting timbers and other obstructions; while they are of undoubted practical utility, the point is emphasized that they are only approximations. It would have increased the value of the paper had some of the experimental data obtained by the Anaconda Copper Co., regarding frictional coefficients in various kinds and sizes of tubing, been given; also reasons why the Anaconda Copper Co. adopted flexible tubes for ventilating blind ends; and the data obtained from the experiments on flow of air in shaft models. There is widespread interest in experimental work on frictional coefficients in mine openings and in tubing used in mine ventilation, and in suitable fans for various kinds of mine service. The data collected, however, is conflicting; for instance, some experimental work on flow of air in tubes gave frictional coefficients for flexible tubing as less for similar service with rigid tubes, while other experiments gave coefficients that are greater; these uncertainties should be cleared. Experimental work on ventilation is being done by the U. S. Bureau of Mines at the Experimental Mine, Pittsburgh, Pa., by the University of Illinois at Urbana, Ill., and by other agencies; in the interest of the mining industry, it would seem advisable that these various agencies adopt standard testing apparatus and methods, and thus avoid the issuance of conflicting data.

On pages 42 and 43, it would have been interesting had costs of lining with slabs and with chicken wire and burlap with the cement gun been given, with data as to the relative effectiveness of the two methods. It is stated that working shafts are to be intakes; this is the desirable method but it is impracticable in certain regions because of the accumulation of ice in the winter time; how is the formation of ice in the downcast working shafts in Butte prevented? The author states that "all surface fans, with three exceptions, are reversible and are operated as exhaust fans to develop a vacuum system of ventilation." The mining public would like to know what considerations led to the adoption of reversible fans and of vacuum instead of force units.

The doors illustrated in Figs. 8 and 9 are dangerous, especially in case of a fire, inasmuch as they are not tight, are not held closed, and would be much more likely to be open than closed in case of fire, when it would be extremely desirable that they should remain closed. All doors should have a latch that will hold the door closed even when the direction of the air current may change, and automatic doors should be self-closing after cars or men have passed through; moreover, all mine doors should be made as nearly leak-proof as possible; if it is necessary for a certain amount of air to go past a door, a regulator should be placed in the door or in the frame adjacent to the door.

The description of the use of booster fans shows that while booster fans may be, and generally are, dangerous in coal mines, they are extremely desirable in metal mines.

The weakest point in present-day metal mining ventilation practice is bringing the air currents to the places where men are working. In the discussion on page 50 it would have been interesting had the saving of power produced by the fractional width fans been given; the writer understands that there was a decided reduction in power consumption when the smaller width fans were adopted. On page 51 the length of piping or tubing that would necessitate the installation of a second fan or blower in the series should have been given; also the kind of tubing used.

Under the heading Ventilation Records and Maps, it is stated that the man in charge of ventilation at each mine is employed under the direction of the mine foreman and is expected to act in ventilation work in coöperation with the ventilation engineer. If the man in charge of ventilation must report to that foreman, who is responsible for present practices, he will probably not hold his job very long if he makes very drastic recommendations as to needed changes, as the foreman would probably resent those recommendations as criticisms. Until such time as metal mine foremen become more sympathetic with ventilation work, the men in charge of ventilation work should report only to higher officials and be responsible only to them.

The cost of ventilation work given would be more interesting if it were reduced to the amount per ton of ore hoisted or per pound of copper produced. It would also be interesting if a statement could be made as to whether this immense expenditure of money has been justified and, if so, in what manner both as to savings in dollars and cents and as to health, welfare and general attitude of the workers.

CHAS. A. MITKE,* Bisbee, Arizona (written discussion).—Having visited Butte every few years for some time past, I have been able to note the improvements that the Anaconda Copper Co. has been making in the ventilation of its mines. This subject is of great interest to me, as I have worked out and supervised the installation of a number of the large ventilating systems in the Southwest, where, except that the Anaconda mines are much deeper, the difficulties to be overcome in providing efficient ventilation are very similar.

In the deeper mines in the Southwest, in addition to the regular temperature and relative humidity problems, as well as the heat from fire areas, we have the excessive heat generated by friction and oxidation of ore carrying from 10 to 45 per cent. sulfur. This must be taken care of by the ventilating system, as additional quantities of heat may issue from such a district at any time, depending on the crushing effect of moving ground over old stoped areas. It is the practice to force large

* Consulting Mining Engineer.

volumes of fresh air into such areas in order to cool the ore before it takes fire, provision, of course, always being made that such air does not return and flow through the live workings. After a fire is once started, however, a different method is used, depending on the problem at hand.

A. S. RICHARDSON (author's reply to discussion).—Mr. Harrington refers to the writer's objection to the use of the word "oxidation" in reference to the processes occurring during the decay of mine timber. While the correctness of the terminology is not a matter of great importance the writer is not satisfied that Mr. Harrington's argument in support of the word is well founded. The comparison to the action that causes spontaneous combustion of coal contains the statement that in this case no bacteriological action occurs, and as fundamental differences exist the analogy is by no means evident.

Special measures for the ventilation of pump stations are necessary on account of the heat generated by the electrical equipment, and are practiced. Most frequently the fresh air that is drawn from the shaft for this purpose is diverted directly to the outlet air courses and does not pass through other working places.

The reference to the low temperature record at the West Colusa mine, where fire fighting was being carried on, was intended to bring out the fact that, in itself, the fire had not greatly affected the temperature of the main body of mine workings. Mr. Harrington's deduction that the ventilation of the mine is efficient is believed to be correct. However, the presence of a mine fire frequently causes complications that render efficient ventilation decidedly difficult.

The formation of ice in the downcast shafts has never been a serious obstacle in Butte, even though the winters are severe, but some difficulty occasionally occurs. Elimination of the trouble is accomplished by prevention, by catching up and diverting the flow of water, and by removing the ice as soon as it forms. In at least one instance the main surface fan was reversed and the ice melted by the warm air from the mine.

The reason for having main surface fans reversible is in order that the direction of flow of air through the mine may be reversed in case this should become desirable because of a mine fire, or any other contingency. It is believed that the most important reasons for selecting vacuum units for main surface fans were given in the original paper; no other selection would be practicable so long as the hoisting shafts are main inlet air courses.

Many difficulties have to be overcome in designing efficient ventilation doors for metal mines having heavy ground and strongly acid copper water. Those now used in Butte were developed as the result of years of experience during which many types were tried out under operating

conditions. It is not thought that they are necessarily the best that may possibly be designed, and work is now being done to improve and to standardize this equipment.

Use of fractional width fans for use with canvas tubing permitted the use of motors of about one-half the horsepower of those required for the standard width fans. For the No. 4 standard Sirocco blower a 20-hp. motor is used under these conditions, while for the No. 4½ width blower a 10-hp. motor is used. For the No. 2½ standard blower a 7½-hp. motor is used, while on the No. 2½, one-third and two-thirds width blowers the motors are 3-hp. and 5-hp., respectively. On an order for 300 blowers the saving in total motor horsepower was 1,837 horsepower.

Use of canvas tubing in preference to galvanized-iron pipe is made because of lower cost of canvas pipe, flexibility of the material permitting greater ease and consequent lower cost of installation, greater efficiency when maintained in good repair due to smaller number of joints and lower leakage losses. To a certain extent there are limitations to the effectiveness of both kinds of tubing because the air flowing through the tube becomes heated by the hot air surrounding it. In this respect the canvas tubing has also a slight advantage because of the lower heat conductivity of the material.

The first canvas tubing used in the A. C. M. Co.'s mines on a large scale was a rather inferior article which was purchased from an outside manufacturer. The service secured from it was so poor that it became necessary to get something better, and that now used is made in the company's own shops. The material is canvas of what is ordinarily called 9-oz. weight, and it is coated with a roofing compound. Other manufacturers have now produced tubing that is much more serviceable than that formerly obtainable, but, considering the large proportion of tubing destroyed by purely mechanical causes and the greater cost of the more thoroughly protected material, it has not so far seemed advisable to make any change.

No exact rule can be given for the installation of two or more blowers on a pipe line because the resistance of these lines varies a great deal irrespective of length. Lines as long as 2000 and 3000 ft. have been carried satisfactorily in fairly cool places with the use of only one blower, but, in general, additional power seems necessary when the length of pipe is greater than 1200 feet.

Complete treatment of such subjects as the resistance of tubing, different types of shafts, and other components of mine resistance would be quite lengthy, and unduly prolong the discussion of a paper which was written solely as an account of the ventilation work recently done, or undertaken, by the Anaconda Copper Mining Co., and not as a technical treatise. This can more appropriately be done in a separate paper, and

as a considerable amount of such information has been gained a paper on the subject will be offered.

Justification of expenditures for ventilation work cannot be exactly defined in terms of dollars and cents. Numerous other factors affect labor efficiency, and it is not possible to estimate the benefits derived from the improved health and welfare of the miners. At one mine the tons per man per day for miners employed in breaking ore in stopes, on a contract basis, was increased approximately 50 per cent. during the period of ventilation improvement, and the substantial drop in average temperatures that has been effected cannot be without great benefit to all concerned.

Mine Fires and Hydraulic Filling

By H. J. Rahilly,* Butte, Mont.

Mine fires, in the Butte district, have been a source of trouble and expense for the past thirty years, for while the actual fire area in most of the mines has been comparatively small, the handling of the gas has generally been a much larger problem than the actual putting out of the fire. The chief causes of these fires have been: Defective electrical equipment, incendiarism or carelessness, and spontaneous combustion.

While fires from the first two causes have been numerous, they have generally been discovered and extinguished before they became very serious. Spontaneous combustion has been the cause of most of the large fires. Sufficient heat has been generated by ground movement, high rock temperatures, and decomposition to set fire to inflammable materials (tarred ropes, canvas, dry timber, manure, and hay) left in the gobs some years ago. Also, the heat resulting from the oxidation of the fine broken sulfide ore is sometimes sufficient to set fire to the timbers.

In 1906, a fire was discovered on the 1100-ft. level of the Minnie Healy mine about 400 ft. (122 m.) west of the Minnie Healy shaft, which presumably originated, in one of the upper levels, from spontaneous combustion. The fire was walled off by concrete bulkheads and a pillar of ground 100 ft. (30 m.) thick was left below and mining resumed underneath. During subsequent operations, this pillar became cracked and broken and the fire passed into the new workings. It travelled downward and spread rapidly until, in 1917, it had become necessary to wall off portions of the Tramway, Leonard, and West Colusa mines on the 1300, 1400, 1500, 1600, 1800, 1900, and 2000-ft. levels, and the tops of the 2200-ft. raises were sealed. The fire area now consists roughly of a block 1000 by 1200 ft. and from the 600-ft. to the 2200-ft. level. This block is composed largely of a series of parallel northwest-southeast nearly vertical veins and stringers that form a large orebody, with no well-defined walls but badly fractured and faulted, and known locally as "horsetail ore." These veins, or orebodies, are mined by the square-set or the rill method.

' Mine Fire Superintendent, Anaconda Copper Mining Co.

Prior to this time (February, 1917) mine fires had been combatted by fighting the fire until it was extinguished or if, as so often happened, the fire was in inaccessible territory, the region was sealed and water introduced through diamond-drill holes put in from levels above. It became apparent that to fight so large a fire as this, any method of permanent sealing would be ineffective, as the ground was too badly broken, and the use of water would be futile, because it would form water courses in the stopes and would not spread sufficiently to be of much value; therefore it was decided to fill this walled-off region with mill tailings from the Black Rock mill of the Butte & Superior Copper Co., Ltd. Prior to Dec. 1, 1920, the tailing was taken direct from the mill; but since the mill has been closed, tailings have been obtained by hydraulicking the tailing pond. This hydraulic filling was planned and started in February, 1917, by C. L. Berrien, then assistant general superintendent of mines, with C. E. Nighman, who had charge of fire work in these mines, in charge of the operation. The author became assistant to Mr. Nighman in April, and was in charge from July, 1918, to February, 1919, when Mr. Nighman returned from the army. Since April, 1920, the author has had charge of the work, Mr. Nighman having been transferred to the Anaconda phosphate property.

According to Charles Enzian,[1] "The first instance of the application of hydraulic mine filling is reported to have been at a coal mine in the Schuylkill region of Pennsylvania in 1884 for the purpose of extinguishing a fire in the main haulage slope. About 1886 and 1887, hydraulic filling was introduced in the Lehigh and Schuylkill regions for the purpose of sealing off mine fires, arresting squeezes, and supporting the surface." In these instances culm and not sand was used as the filling material. Hydraulic sand filling has been used for some time in France, Belgium, western Australia, Hungary, Westphalia, Silesia, and, more recently, on the Rand, South Africa; its principal use has been for surface support.

PRELIMINARY WORK

Preparations for sand filling in Butte, Mont., were begun in the fall of 1917. Diamond-drill drainage holes were drilled from the 1200-ft. level of the Leonard mine to the Reins shaft, the shaft unwatered, and a crosscut driven on the course of the drill holes. The shaft and connecting crosscut to the Leonard were put in condition for permanent use of the tailing pipe. A six-spigot spitzkasten was installed at the Black Rock tailings pond and a flume 23 in. wide by 8 in. deep by 3650 ft. long on a 2 per cent. grade was constructed from the tailings pond to the Reins shaft. A 6-in., extra heavy, cast-iron, flanged, pipe line was installed in the Reins shaft from the surface to the 1200-ft. level (1040 ft.) where

[1] U. S. Bureau of Mines *Bull.* 60.

it was divided, one branch going to the 1100-ft. level of the Tramway (which is at the same elevation as the 1200-ft. level of the Leonard) a distance of about 2000 ft. and down the Tramway shaft to the 1700-ft. level (660 ft.); the other branch was run to the 1200-ft. level of the East Colusa (a distance of 700 ft.) and down the shaft to the 1900-ft. level (680 ft.).

METHOD OF FILLING

It was decided that the filling should be built up in layers, starting at the lowest level of the fire region. Therefore, it was first necessary to reenforce all bulkheads on the 2000-ft. level, so they were made 4 ft. thick and provided with two 2-in. copper drain pipes and valves, also to dig ditches for conveying the drain water to the diamond-drill drainage holes that conducted it to the 2200-ft. level, from where it is sent through raises to the pumps on the 2800-ft. level of the Leonard mine.

On the 1900-ft. level of the Leonard (119 ft. above the 2000-ft. level) the main lateral and those crosscuts and drifts that could be utilized were opened and repaired. At the same time new openings were driven so as to get to the tops of raises, or within close proximity to the stopes. Where conditions prevented the driving of these crosscuts until they actually intersected the workings, they were advanced as near as possible and diamond-drill holes were drilled to the stopes below. Some of these holes were inclined so that they hit the stopes about 60 ft. below the 1900-ft. sill, or about 60 ft. above the 2000-ft. sill; the other holes were drilled practically horizontal. The tailings pipe was installed from the East Colusa shaft and connected to these drill holes and these stopes were the first to be filled.

All concrete bulkheads in drifts and crosscuts connected with the fire region, that were not opened, were reenforced with 4 ft. of concrete and provided with two 2-in. copper drains and a 4-in. discharge pipe. The discharge pipe was connected to the tailings pipe and the filling was introduced to this sill and stopes below in this manner or through the diamond-drill holes.

At the same time, the 1700-ft. level of the Tramway (146 ft. above the 2000-ft. level) was opened by crosscuts, drifts, and diamond-drill holes, and the tailings discharged into the stopes through these holes or through concrete bulkheads, the same as on the 1900-ft. level of the Leonard. All bulkheads were placed as near the stopes as possible so as to fill no more of the workings than was necessary.

This work was performed generally under abnormal conditions; such as high temperatures, gas, and broken and caved ground. Oxygen-breathing apparatus had to be used constantly, particularly in blasting through the old concrete bulkheads, after which fans were used to hold back the gas and heat. On the 1900-ft. level of the Leonard, over 4000

ft. of drifts and crosscuts were either repaired or driven. The main supply tailings pipe on both the 1900-ft. level of the Leonard and the 1700-ft. level of the Tramway was 6-in. extra heavy, cast-iron pipe and the lateral lines 4-in. cast-iron and wrought-iron pipe.

As the filling progressed on the 1900-ft. level of the Leonard and the 1700-ft. level of the Tramway, the 1800-ft. level of the Leonard and the 1600-ft. level of the Tramway were being opened by repairing the old drifts and crosscuts, driving new openings, and diamond drilling; and, in turn, the succeeding levels above were opened in the same manner until at the present time (July, 1921) the stopes between the 1600-ft. and 2000-ft. levels of the Leonard are filled and the 1400-ft. and 1500-ft. levels are being opened by crosscuts and diamond-drill holes and are partly completed. The Tramway stopes between the 1500 and 2000-ft. levels are nearly filled. On the 1100, 1200, 1300, and 1400-ft. levels of the Tramway, the stopes immediately north of the shaft have been filled, thereby cutting off the gas that was coming out and interfering with operations in the shaft and other parts of the mine. In the West Colusa, the stopes below the 1600-ft. level are filled, the stopes between the 1500-ft. and 1600-ft. levels are nearly filled, and the 1300-ft. and 1400-ft. levels are being opened by crosscuts and diamond-drill holes.

The mill tailings from the mill contained about 16 to 18 per cent. solids by weight and had approximately the following screen analysis:

	OPENING, INCH	MESH	PER CENT.	ACCUMULATIVE, PER CENT.
Retained on...............	0.0137	40	0.34	0.34
	0.0072	80	7.37	7.71
	0.0058	100	4.27	11.98
	0.0041	150	9.03	21.01
	0.0029	200	19.38	40.39
	0.0025	220	3.89	44.28
	0.0021	240	3.68	47.96
	0.0019	260	3.55	51.51
	0.0018	280	1.29	52.80
		minus 280	47.20	100.00

The spitzkasten thickened the pulp to about 25 to 30 per cent. solids, by weight, having the following approximate screen analysis:

	OPENING, INCH	MESH	PER CENT.	ACCUMULATIVE, PER CENT.
Retained on...............	0.0137	40	0.70	0.70
	0.0072	80	10.90	11.60
	0.0058	100	5.55	17.15
	0.0041	150	12.75	29.90
	0.0029	200	18.00	47.90
	0.0025	220	2.65	50.55
	0.0021	240	3.35	53.90
	0.0019	260	3.50	57.40
	0.0018	280	1.00	58.40
		minus 280	· 41.60	100.00

These tables show that a large part of the slime material and water was removed in the spitzkasten. This was deemed advisable because the tailings frequently leak through the slips and cracks in the ground, especially when there is an excess of water or fine material. When this condition arises, several coats of gunite are applied and the cracks sealed, small pipes being cemented into the cracks to drain off the water.

The 6-in. extra-heavy cast-iron flanged pipe line in the shafts originally was offset every 200 ft. and wooden boxes put in at these offsets to break the fall of the tailings; but these boxes are being removed as they wear out and the pipe line is connected up continuously. The boxes are a constant source of trouble as considerable air is drawn into the pipe line with the tailings, which causes splashing at the boxes if the amount of tailings varies. It has been determined that for each 100 ft. in head in the shaft pipe lines, the tailings can be transported about 1000 ft. on the levels; however as the drifts and crosscuts in many places have settled as much as 10 to 15 ft., the tailings pipe lines have sags or traps in them so that, for each 100 ft. of head, horizontal transportation of slimes for about 800 ft. is considered good practice. There has been little or no trouble with settlement of material at sags. Sometimes, pressure heads as high as 500 ft. were used.

The 90° curves in the pipe are made up in four 22½° sections on a 4-ft. radius. Two of these sections have connections for 2-in. water lines, the other two have handholes, provided with bolted covers, so that the inside of the curve can be examined from time to time. All curves in the pipe lines are made with a 22½° section, a 30° section, or with wedges that can be adjusted to any angle from 0° to 10°, or by a combination of several of these curve pieces. The lateral lines are 4-in. cast iron or wrought iron Much of the latter is used in inaccessible places or where the lines are to be used only for a short time, as second-hand wrought-iron pipe can be more easily obtained, is less liable to breakage, lighter in weight, and more easily installed. Two-inch water lines follow the tailings pipe, connections with 2-in. valves being made at all curves and junction points, in order to flush the tailings pipe with clear water after shutting off the tailings, because if the solid particles are allowed to settle they will tend to plug the pipe line.

Air Locks

When opening up crosscuts or drifts that were sealed with concrete at the time of the fire, a concrete bulkhead 18 in. thick with a tight-fitting iron door is built about 10 ft. outside of the bulkhead that is to be blasted open. Outside of the iron door, two brattices are built about 10 ft. apart with tight wooden doors; a No. 2½, No. 3½ or No. 4 Sirocco fan is placed outside these brattices and 12-in. or 16-in. canvas fan pipe is put

through the brattices above the doors and made to discharge inside, but outside of the iron door. In the second brattice, a flapper valve is inserted in the fan pipe which is held open by the air pressure, but closing down the fan automatically shuts it, thereby allowing little or no gas to escape from within the brattice. The flapper valve is a piece of sheet iron hinged at the top forming a tight seal over the end of a piece of 16-in. wrought-iron pipe. The bulkhead is then blasted open and the fan made to hold back the gases so that the men can work in good air. The brattices are made of 1-in. boards, covered with metal lath and coated with gunite, or of burlap over which 1-in. mesh chicken wire is stretched and then gunited. These brattices are built very rapidly—digging out the loose ground, building the wooden frame, putting on the burlap and wire, hanging the door, and guniting rarely take over two 8-hr. shifts of three men on a shift. The writer favors the latter construction because the time and cost required to build such a brattice is materially less than when boards are used. The air pressure maintained by the fans and air-lock doors is adjusted so as barely to hold back the gas, this being done by regulating the size of the fan intake. Water-gage readings at several of the doors have shown as much as 30-lb. pressure per square foot.

Rock Temperature and Humidity

In repairing the old workings, the air temperature is seldom under 85° F. with relative humidity from 70 to 100 per cent. In driving new openings, particularly near the stopes, the temperature of the rock generally becomes higher and, in a few instances, was so high that men were unable to work; also there was considerable danger in blasting a round of holes, galvanized-iron blasting pipes 1⅜ in. in diameter being used and only two or three holes fired at one time. Rock temperatures as high as 287° F. were recorded. Fans and air blowers are used constantly for ventilation and for making the places workable. Table 1 shows some of the air and rock temperatures encountered in opening the 1800-ft. level of the Leonard mine.

Use of Diamond Drilling

Diamond drilling is resorted to where it is too hot to drive crosscuts, or where a crosscut will allow too much air to reach the fire, or where it is cheaper to drill than to drive a long opening. The drill holes are 2½ in. in diameter and cased with 2-in. extra heavy pipe, made up in 5-ft. or 6-ft. lengths with flush or casing joints. These holes are connected to the 4-in. lateral lines by short pieces of hose equipped with suitable couplings. Hose is used because of the ease with which the connections can be made and the ease with which it is possible to determine whether the pipe line is plugged with tailings.

TABLE 1.—*Temperature and Relative Humidity on 1800-ft. level of Leonard*

Place	Date	Temperature, Degrees F.		Relative Humidity, Per Cent.
		Dry Bulb	Wet Bulb	
1800 and 1801...............	10/2/19	90	89	96.0
1895A breast.................	1/28/20	92	92	100.0
1894 breast..................	1/28/20	121	99	45.5
1750E, 100 ft. W1893..........	1/28/20	99	90	70.5
1756A breast................	1/28/20	122	96½	38.0
1752A.......................	1/28/20	105	93	63.0
1750W and 1754A............	1/28/20	92	82	65.5
1754 SS No. 1...............	1/28/20	98	93½	81.0
1825 and 1803...............	1/28/20	107	102	84.0
1847....................	10/10/19	118	105	64.0
	1/28/20	117	114	91.0
1847 compressed air..........	1/28/20	114	108	82.0
1847 fan discharge....	1/28/20	108	94	59.0
1847 rock...................	1/28/20	195		
1758A breast........	10/19/19	138	100	63.0
	10/23/19	131	87	63.0
	10/25/19	120		
	1/28/20	113	100	63.0
	9/28/20	110	87	50.0
	11/18/20	104		
Rock.......................	10/19/19	287		
Rock.............	10/23/19	247		
Rock.......................	10/25/20	234		

Six diamond-drill machines are used constantly on the work, five being operated daily on three 8-hr. shifts; the sixth machine is used as an extra so that no time will be lost in case one of the machines is shut down for repairs or moving. Up to July, 1921, approximately 80,000 ft. of drilling had been done with 1150 karats of carbon; the average length of hole is 60 feet.

TELEPHONE INSTALLATION

Telephones are installed at points convenient to each discharge place so that communication can be established between any part of the mine or surface. This is very important because:

1. All valves must be set properly before changing the tailings from one stope to another, otherwise the pipe line will be plugged.

2. The tailings pipe must be flushed thoroughly with water after filling a stope, otherwise the sand material may settle and plug the pipe.

.3. The flow of tailings can be changed quickly to places unaffected should leaks through the ground or breaks in the pipe line occur. The

tailings are run into any one stope from 4 to 8 hr. continuously and then the flow is changed to another stope, enough places being kept available for continuous operation. Tailings are run into no stope for more than eight in each 48 hours.

After a stope has apparently been completely filled, a few holes are drilled into the stope to test out and make certain that no openings are left and that no gas or fire exist; also an accurate account of the amount of tailings flushed into any stope is kept and an approximation of the amount of material that the stope should take is made from the stope maps.

BREATHING APPARATUS AND CEMENT GUNS

Oxygen breathing apparatus is used to a large extent in opening up old crosscuts, particularly in building air locks, blasting bulkheads, sealing doors, and making exploration trips to determine the amount of work necessary to introduce the tailing. From January, 1917, to July, 1921, 26,000 sets of apparatus were used. Deficiency of oxygen and the common mine-fire gases, as carbon dioxide, carbon monoxide, sulfur dioxide, and steam, are encountered.

Three to six cement guns are constantly used in building bulkheads, brattices, guniting cracked and broken ground to stop gas leaks, and fireproofing fan stations. Up to July 1, 1921, approximately 20,000 cu. yd. of sand and 125,000 sacks of cement had been used.

PUMPING INSTALLATIONS

In November, 1920, the Black Rock mill of the Butte & Superior Copper Co., Ltd., was closed down so that no more filling material could be obtained from that source. As there were several hundred thousand tons of tailings in the tailings pond, it was decided to utilize these. The part of the pond first attacked was that south of the north end of our 3600-ft. flume. This comprises a block roughly 300 by 800 ft. and varying from 20 to 50 ft. in thickness, which contained 252,000 tons of available material. On account of the possible semiliquid condition of the center of the pond and because severe rain storms are not uncommon, it was decided to leave around the edge a rim or bank of tailings at least 30 ft. thick on top, so that by no possible chance could the tailings be washed down Silver Bow Creek or southward toward the dwelling houses.

No. 1 shaft, consisting of two compartments, was sunk 30 ft. through the tailings to bedrock. This shaft is 5 by 10 ft. with the 4 by 10-in. timbers laid skin to skin. The lower 10 ft. of one compartment was concreted, both walls and floor, so as to be watertight and a 4-in. Krogh centrifugal pump installed, driven by a G.E., 25-hp., three-phase, 60-cycle, 440-volt, 1150-r.p.m., K motor. The other compartment was used as a sump and was connected to the intake of the pump by a 4-in.

pipe. The discharge from the pump is a 5-in. pipe 600 ft. long with a static head of 36.67 feet.

The tailings are sluiced by water obtained from the Black Rock mine and by water from Dixie and Yankee Doodle Creeks, about 450 gal. per min. being available. This water is flumed and piped to a 2800-gal. water tank, from which it is drawn to one of the two $2\frac{1}{2}$-in. Worthington centrifugal water pumps, running 1500 r.p.m. driven by a G.E. three-phase, 60-cycle, 440-volt, 15-hp., 1120-r.p.m. IK motor. One of these pumps is always kept in reserve. From the pumps the water is conveyed through pipes and $2\frac{1}{2}$-in. hose to the working place, where it discharges through a 1-in. nozzle at about 30 lb. pressure.

The tailings were first worked out around the shaft and down to bedrock; and as the solid material had a tendency to settle out in the sump, an agitator was installed. This agitator is a wooden tank 8 ft. 8 in. in diameter and 4 ft. high in which four revolving arms, running at 22 r.p.m. are driven by a 15-hp., back-geared, three-phase, 60-cycle, 440-volt, 1140-r.p.m., CS Westinghouse motor.

No. 2 shaft, consisting of two compartments, was next sunk 52 ft. to bedrock. This shaft is 14 by 24 ft., lined with 4 by 10-in. planks placed skin to skin. The lower 16 ft. of one compartment was made watertight with 6-in. concrete walls and floor and two 4-in. Krogh pumps and motors similar to the installation in No. 1 shaft were installed. In the other compartment, an agitator of the same design as No. 1 plant was installed. The discharge pipe is 5 in. in diameter and 300 ft. long with a static head of 55.54 ft. on the pumps.

At present, either plant can be operated; and as the sluicing operations are continuous, changing pump liners and runners, or other necessary repairs, are made by shifting from one pump to another. Pump liners and runners will pump about 7000 tons of solids in the No. 1 plant and about 5000 tons in the No. 2 plant before they must be renewed.

The volume of tailings delivered to the mine is measured by a weir and hydrochronograph at the Reins, just before the pulp enters the shaft. Samples are taken every $\frac{1}{2}$ hr. and the tons of solids and gallons of water are computed. The daily capacity varies from 500 to 1500 tons. Before the Black Rock mill closed, 600 tons a day was the average capacity; but since sluicing operations were installed the capacity has been increased so that the average is over 1000 tons per day. The amount of solids varies from 20 to 50 per cent., by weight, depending on the size of the individual particles handled. From Dec. 1, 1920, to July 1, 1921, 154,000 tons of solids and 72,000,000 gal. of water were pumped into the mine.

The tailings have been run through the surface flume with the atmospheric temperature as low as 40° below zero, with no trouble because of freezing (the temperature of tailings entering the mine was 42° F.). Less

trouble is experienced, in winter, by running continuously than when the runs are intermittent, as in the latter case the launder gets cold and trouble from freezing is experienced when starting up. With about 300 gal. per min., about 50 per cent. solids, by weight, is the maximum of solids that will flow down the flume without blocking.

No slimy water is allowed to go to the pumps on the 2800-ft. level of the Leonard, as these are of the plunger type and the sand particles tend to eat out the valve seats. All water pumped out of the mines is measured by a weir and hydrochronograph, so that by a comparison with the amount of water being put into the mine, any large reservoirs of water accumulating in stopes can be detected and steps taken to drain them. All water drained off carries copper in the form of copper sulfate, the sulfides of the ores having been oxidized by the fire. It is sent to the precipitating tanks, where the copper is precipitated and recovered.

FIRE FILLING DEPARTMENT

The work of extinguishing the fire with tailings is carried on by the Fire Filling department. The work consists in opening up the old workings, driving new openings, diamond drilling, installing pipe lines, putting in concrete bulkheads, brattices and doors, running the tailings into the stopes and caring for the drainage. The department employs about 250 men, who are distributed as follows: 1 foreman or superintendent, 2 assistant foremen, 1 office man, 9 shift bosses, 33 diamond-drill men, 130 miners, muckers and timbermen, of which seventy-five are trained to wear oxygen apparatus for performing the necessary work in the gas, 12 pipe men, 18 cement-gun men, 3 discharge men, 6 hoisting engineers, 1 oiler, 6 station tenders, 6 tool nippers, 6 drain men, and 20 laborers. All work on three 8-hr. shifts, seven days per week; as much of the work as possible is done on contract. Considerable of the work of opening up the old workings is done by the crews of the Tramway, Leonard, and West Colusa mines under the supervision of the Fire Filling department. From 75 to 125 men are employed daily in this capacity. In the sluicing operations at the tailings pond, a boss and six men are required on each shift; they perform the actual sluicing, changing of pump liners and runners, installing launders, and other incidental work. In addition, a man at the Reins takes care of the weir, hydrochronograph, and the taking of samples. Another man is located at the discharge end or the place where the tailing enters the stope.

From 1917 to 1921, approximately 155,000 shifts were worked. Approximately 90,000 shifts will be worked this year if present conditions continue. From September, 1918, to July, 1921, about 460,000 tons of solids and 280,000,000 gal. of water have been utilized; it is estimated that a total of about 800,000 tons will be required for the job.

The accident record for the actual fire filling work as charged by the A. C. M. Co. Safety Department to the Reins mine, shows that, for the years 1919 and 1920, out of a total of 106,200 shifts worked there were six major accidents—one was fatal and the other five occasioned incapacity for over two weeks.

The effect of the filling on the mine conditions has been marked. The levels where the filling is completed are free of gas and cooler, and practically all settlement of the ground has stopped. Table 2 shows the change of temperature that has taken place since the work of filling was begun.

TABLE 2.—*Temperature and Relative Humidity of Air Before and After Filling*

| Place | Date | Temperature, Degrees F. | | Relative Humidity, Per Cent. |
		Dry Bulb	Wet Bulb	
1701 Tramway and 1772.......	9/29/19	108	108	100.0
	9/30/20	102	102	100.0
1799 Raise and 1772..........	9/29/19	101	101	100.0
	9/30/20	88	86	92.0
1653 Leonard and 1601..	1/28/20	88	86	92.0
	9/28/20	81	78	87.5
1620 Leonard and 1601........	1/28/20	90	87	88.5
	9/28/20	87	85	92.0
1894A.......................	1/28/20	121	99	45.0
	11/19/20	89	81	71 0
1750E.......................	1/28/20	99	90	70.0
	11/18/20	88	81	74.0
1892-93 corner	9/28/20	86	83	88.0
	11/18/20	79	74	79.0

Several crosscuts and drifts that were filled have been opened and in each case the tailings were found to have filled the opening completely, were well drained, and compact enough to pick to a vertical face. One of the best examples was observed in driving A1629 drift in the Tramway through 1601 crosscut, which had been filled 9 or 10 mo. before. The gas and steam known to exist farther north (tests being made through a 4-in. pipe) were entirely sealed off and the rock temperature had been reduced to normal.

CONCLUSIONS

The following conclusions have been arrived at by the writer as to the operation and results of the filling:

1. The tailings will make possible the recovery of several million tons of ore.

2. It will extinguish the fire.

3. It will tend to eliminate high rock temperature.

4. It will eliminate gas.

5. It will help materially to increase ventilation.

6. It will tend to stop movement of the ground.

7. Material used is mill and flotation tailings consisting of 75 to 85 per cent. silica, and crushed so that 50 per cent. will pass 280-mesh screen. Little trouble is experienced in sluicing this material down a 3600-ft. flume on 2 per cent. grade, in 250 to 600 gal. of water per minute carrying from 20 to 50 per cent. of solids, by weight.

8. The wear on the pipe has been negligible, and wrought-iron pipe for distribution laterals underground would serve better than cast-iron pipe, as it is cheaper and easier to handle, but the writer would prefer cast-iron pipe in shafts and main distribution pipes.

9. Water drains through bulkheads and rock pillars, so that no great pressures are built up.

10. A rubber-lined pump has been tried at the tailings pond, but because of nails and small pieces of metal in the tailings, the rubber lining would not stand the service; at present, the use of this pump is still in the experimental stage.

11. The success of the filling has depended on the use of fans, breathing apparatus, and the cement gun; without these the work would be difficult.

12. Filling of this character is particularly applicable to mine fires of large extent and long duration, where temperatures are high and the actual fire is inaccessible.

13. Under certain conditions, the filling might be used as ordinary mine filling and to fill stopes that have been mined out, thereby excluding any possible chance of their catching fire.

DISCUSSION

(Includes also the paper of E. M. Norris, pages 77–85)

R. DAWSON HALL,* New York, N. Y.—Why was the airshaft lined with slabs rather than gunited? The slabs would not seem to give as fireproof a job. Did you decide against gunite because movement of the ground would be apt to break it?

* Engineering Editor, *Coal Age.*

WM. B. DALY, Butte, Mont.—The airshafts of the Anaconda Copper Mining Co., in the Butte district, have a fixed cross-sectional area. Because of the great depths with increased temperature and humidity it was necessary to increase the volume of air taken into and out of the mines. The water temperature at 3400 ft. varies from 104° to 107° F.; so that the air absorbs moisture readily. For that reason, without ventilation, the humidity, in the bottom levels, would come close to 100 per cent. saturation.

When we calculated the cross-sectional area of all the outtake shafts, we found that we were exhausting from those shafts as much air as we economically could with reasonable horsepower. We were passing about 1,500,000 cu. ft. of air per minute through the twenty-six properties. To double that, of course, meant doubling the cross-sectional area, with the same velocity in the shaft. We decided that the only way we could increase the volume of air passing through the mines was to smooth the surface of the outtake shafts or double the number of air shafts. To double the number of air shafts would cost between $10,000,000 and $15,000,000.

We decided to smooth-surface the shafts by lining them with 1-in. board; but that increased the fire hazard, so we developed concrete slabs to smooth-surface and at the same time fireproof the shafts.

The test made to determine our policy was at the Parnell shaft, which has a cross-sectional area of 85 sq. ft. A No. 17 double-intake Sirocco fan with a 300-hp. motor was installed, when it was found that we could exhaust 135,000 cu. ft. a min. with 262 hp.; I think the water gage was 3.6. After smooth-surfacing with slabs, we exhausted 262,000 cu. ft. with 242 hp., with a water gage of 3.4. We doubled the quantity of air passing through the mines and used slightly less power.

The downcast shafts, which are the operating shafts, were gunited. I would not care to be on the cage at 3400 ft. when a rock struck the bonnet in a smooth-surfaced operating shaft; but by guniting the timber, the fall of a rock is broken by bounding from set to set.

FRANK HAAS, Fairmount, W. Va.—What is the minimum velocity in the airshafts?

WM. B. DALY.—In the stopes, drifts and crosscuts, about from 300 to 400 ft.; 200 ft. is sufficient. In our shafts we must have a velocity of from 2500 to 3000 ft. in order to get out the quantity of air we desire; the different levels leading to the shafts make so many feeding points, that the movement of the air is greatly reduced. We regulate the velocity of the air in the shafts to correspond with the velocity of the movement of air in the workings.

FRANK HAAS.—In airshafts of coal mining, we try not to exceed 1300 ft.

WM. B. DALY.—You have smooth-surfaced shafts?

FRANK HAAS.—Yes.

D. HARRINGTON,* Denver, Colo. (written discussion).—According to the paper of Mr. Norris, fires are due largely to conditions from electrical installations; and if this is true in Butte it is true in both coal and metal mines throughout the United States. It is significant, too, that the author pays considerable attention to gunite for fireproofing. His plan of removing lagging at intervals of 100 ft. in order to bring the gunite in direct contact with the rock and eliminate the possible creeping of fire behind the gunite should be considered by all users of gunite for fireproofing, both in drifts and in shafts.

While the author does not touch upon this point, it is my understanding that since the guniting of many of the drifts in the Leonard and adjacent mines there has been a great decrease in the amount of repair work because of decaying timber. Fig. 4 is an excellent example of what should be done by all users of electric motors underground, even when the motors or other electrically driven machinery are to be used only temporarily. The cost of fireproofing these small stations is negligible, when weighed against the loss caused by fire.

The fire-patrol system described is excellent. The members of the fire-patrol crew are trained in the use of mine-rescue apparatus, and to encourage them the company gives a bonus of 10 per cent. over the miner's wages. This practice might be adopted to good advantage by companies that have difficulty in retaining the services of men trained in the use of mine-rescue apparatus.

The suggestion that the mine bosses should become thoroughly familiar with the ventilation of their mine and that the foremen should give bosses theoretical problems on ventilation based on supposition of a fire should be adopted in every metal mine in the United States.

Mr. Rahilly's paper describes one of the most interesting mining problems in the United States at the present time. The origin of this fire was probably spontaneous combustion of hay, manure, and like materials thrown into abandoned workings, a practice that still exists in metal mines in some parts of the United States. After attempting to control this fire by the usual methods of sealing and, later, by diamond drilling with introduction of water, without success, filling the open spaces by the slimes and draining away the water soon extinguished the fire.

An interesting feature is that for years a fan was used to force the fumes out of the region in order to prevent the surrounding mines from being flooded with gases. After this method was abandoned, it was found that

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* Supervising Mining Engineer, U. S. Bureau of Mines.

the gases could be controlled by sealing, though occasionally they would break forth. The fact that apparatus was used 26,000 times from January, 1917, to July, 1921, without the loss of life shows the efficiency of the apparatus and the care with which the apparatus was used. The work was done largely in air with a temperature from 90° to 130° or 140° F., frequently in smoke so dense that the apparatus wearers could not see each other, and in caving, broken ground. It is probable that oxygen breathing apparatus has been used to a greater extent and with greater efficiency in the fighting of this fire than in any similar occurrence in the United States.

Probably the cement gun has been used to a greater extent and with greater utility in fighting this fire than in any other region in the United States. When it is noted that 18 men are using the cement gun practically every day in the week every week in the year, the immensity of this practice can be realized. Similarly, the use of diamond drills underground is notable. As 33 diamond-drill men are continually engaged on the work, though no figures are given, the diamond-drill footage is tens of thousands of feet; much of this work is done where the rock and air temperatures are largely in excess of 100° F.

One of the problems that worried the management at first was the handling of the extremely fine material, it being thought that this material would not settle from the slimes and would accumulate with attendant extremely high pressures, so an attempt was made to remove some of the fines. But, later, when the material was sent direct from the mill practically no trouble was experienced because of the method of handling the slimes. That the method finally adopted is successful is shown by the fact that the plunger pumps in the Leonard mine, which handle the liquid material that drains from the slime-filled area, would quickly be worn out by the fine siliceous material if any went through. There has been comparatively little difficulty with the pumps and the water drained from the bulkheads in the lower part of the mine is clear, as far as solids are concerned, though impregnated with the bluish or greenish copper sulfate.

While costs are not given, the cost of handling is very small per ton of material. The paper states that from September, 1918, to July, 1921, about 460,000 tons of solids and 280,000,000 gal. of water were sent into the fire region; up to the present time well over 600,000 tons of solids have been placed there.

To those who are slow in taking precautions against fires in metal mines, this paper should convey a warning. There are engaged in this work over 250 men daily; from 1917 to 1921, approximately 155,000 shifts were worked; the cost, including labor and material, is probably over $1,000,000. Most mining companies would not be as fortunate as the Anaconda Copper Mining Co. is in this affair. The fire oxidizes copper sulfide, which is leached out by the fluid from the slimes, and the copper

sulfate from the water pumped from the mine returns a large amount of copper, which tends materially to reduce the expense of fighting the fire. One of the remarkable features in connection with this work has been the fact that although the work is of an extremely dangerous nature with high temperatures, poisonous gases, hot rock, broken and caving territory, and difficulty of ventilation, out of 106,200 shifts worked in 1919 and 1920, there were but six major accidents; one of which was fatal while the other five occasioned incapacity for over two weeks. This is a record of which any company might be proud and indicates that accidents can be prevented if proper precautions are taken, for the work is being done under probably the most dangerous conditions in metal mining in the United States.

The fire fighting at the Reins mine constitutes one of the most interesting problems in present-day metal mining and neither this discussion nor Mr. Rahilly's paper has brought out a number of interesting features that have been encountered and handled.

ROBERT E. TALLY, Jerome, Ariz. (written discussion).—The methods used by the Anaconda company for the prevention of mine fires are most effective. Mine fires may occur in any timbered workings and their results are often so serious that we should use not only preventative measures, but control and fire-fighting methods. Plans should be made to fight a fire in any portion of the mine, and the organization responsible for handling the fire should be thoroughly familiar with them, in order that no time will be lost should a fire occur. Perfect control of air currents is most essential, for these currents not only increase the extent of the fire, but endanger the men in the path of the currents to suffocation from smoke.

Underground Fire Prevention by the Anaconda Copper Mining Co.

By E. M. Norris,* Butte, Mont.

During the winter and spring of 1917, an unprecedented number of underground fires occurred in the Butte district. With one exception, these fires were caused by the failure of electrical equipment, and called attention to the increased fire hazard in underground mining operations occasioned by the extensive use of electricity.

Five local conditions that contributed toward a heavy fire risk were: (1) The heavy continuous timbering necessary for mine supports; (2) the strong ventilating pressures in the shafts and main airways; (3) the subsidence and faulting movements in the country rock, which ruptured cables and displaced trolley and lighting wires; (4) the strongly acid mine waters, which quickly developed any weak points in the electrical insulation; (5) oxidation in old stopes containing large amounts of timber.

In the summer of 1917, a comprehensive plan of fire prevention was begun by the Anaconda Copper Mining Co., which involved a thorough and intensive development of the preventive measures then in use. These were as follows: Fireproofing, remodeling and strengthening electrical insulations, extension of underground water system, control of ventilation, maintaining efficiency of fire-fighting crews, and reorganization of fire patrol.

In considering the problems of fireproofing, it was immediately apparent that some method of applying a durable coat of fireproof material to the mine timbers was needed. The first experiments were made with a plaster composed of one part cement, one-half part fireclay, and three parts sand. This was applied by hand to a thickness of 1 in. (2.5 cm.) over a reenforcement of herringbone metal lath. It made a very satisfactory covering for fan stations, but was too soft and friable for traveling ways, where it was subject to abrasion. The cement gun was tried next with a three to one mixture of sand and cement. A ½-in. coating of gunite applied over poultry netting or metal-lath reenforcement proved

*Assistant General Superintendent of Mines, A. C. M. Co.

to be durable and fire resisting. Preparations were then made to fire-
proof the Tramway No. 1 hoisting shaft.

This was a four-compartment shaft, 2475 ft. (754 m.) in depth; it was
lined with framed shaft sets of 12-in. (30 cm.) fir timber; the sets were
7 ft. by 18 ft. 10 in. over all, and were spaced 5 ft. on centers. The sets
were lagged with 2-in. pine plank, placed against the rock walls. The
timbers and lagging were covered with a ½-in. coating of cement, applied
with the cement gun over 27-gage diamond-mesh metal lath. The fire-
proofing operation was completed[1] in the fall of 1917. Because of the
success of this operation, it was decided that the twenty-four main
hoisting shafts operated by the Anaconda company in the Butte dis-
trict, not already so, should be made downcast shafts and fireproofed
along these lines; also that the same treatment should be given all surface
tunnels carrying steam, air, and electric power from plant to mines.

Twelve type N-I cement guns were procured and the work pushed
as fast as operating conditions would allow. One or more shafts at a
time were withdrawn from ore production, retimbered where necessary,
and fireproofed under the direction of the mine foreman. Each foreman
selected a crew, from among his shift bosses and shaftmen, who were
trained at one of the mines where fireproofing work was in progress.
The work was carried on continuously by three 8-hr. shifts. The crew
for each shift consisted of one shift boss; six shaftmen, who nailed on the
metal-lath reenforcement; two nozzlemen for the cement gun; two runners
for the cement gun; station tenders; topmen; etc. The fireproofing work
has been completed at the High Ore, Moonlight, Tramway, Rarus,
Mountain View, Leonard, and Belmont mines. It is in progress at the
Pennsylvania and Diamond mines, while preparations are being made at
the Original and East Colusa mines.

Regarding the details of the gunite work, some general conclusions
may be summarized: Poultry netting makes satisfactory reenforcement
for covering wooden surface structures of continuous plane surfaces,
such as buildings and ore-bins; expanded-metal lathing is to be preferred
where much fitting is necessary, as on underground timbering. A gunite
covering ½ to ¾ in. thick on wooden surfaces is sufficient for fireproofing
purposes. The sand should be clean and should be screened through a
screen four or five meshes to the inch. If the sand is very dry, it does not
mix well with the cement; 6 to 10 per cent. moisture is best. Moisture
in excess of 15 per cent. is liable to block the material hose. A mixture of
three parts of sand to one of cement is best. A water pressure of at
least 40 lb. per sq. in. is desirable; the air pressure may be from 40 to 100
lb. per sq. in., but the air should be dry. The working air pressure in the
cement gun must vary with the distance from the gun at which the nozzle
is being used, but should be kept as low as will keep an even flow of

[1] E. M. Norris: Fireproofing Mine Shafts. *Trans.* (1919) **61**, 201.

material through the hose. Using a 50-ft. material hose, the pressure should be from 5 to 7 lb. per sq. in. The limit at which the nozzle can be used, with satisfactory results, is about 200 ft. vertical or 500 ft. horizontal distance from the cement gun. Under these conditions, the air pressure in the gun will be from 25 to 35 lb. per sq. in. It is important that the mixing chamber of the cement gun be cleaned out every second or third day when the gun is used continuously; the cement finds enough moisture to set on the rotating plate, where it clogs the feeding channels. A $\frac{1}{2}$-in. mesh screen fastened over the hopper of the cement gun obviates many delays. There should be at least one experienced man in a cement-
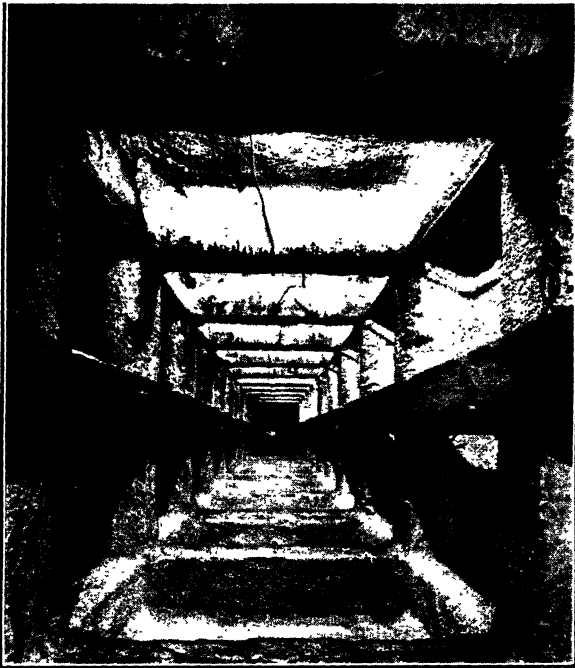
FIG. 1.—RARUS SHAFT 18 MO. AFTER GUNITING WAS COMPLETED.

gun crew; new men are quickly trained, but an inexperienced crew gets very poor results. The figure used in making estimates for materials was 1 cu. ft. of sand to $7\frac{1}{2}$ sq. ft. of surface gunited $\frac{3}{4}$ in. thick.

The Tramway shaft has had $3\frac{1}{2}$ yr. continuous service in ore production since the fireproofing work was completed. A recent inspection of the shaft showed that the timbers from the surface to the 1600-ft. level were 92.5 per cent. covered with gunite; from the 1600-ft. to the 2800-ft. level, the timbers were 85 per cent. covered with gunite. The depreciation of the gunite is due to falling rocks from overloaded skips. In wet parts of the shaft, where there has been sufficient ground pressure

to buckle the timber, there is a tendency for the acid mine water to run between the timber and the gunite covering. This eats away the nail fastenings and loosens the gunite. The damage from this cause has been slight up to date. It can be controlled by proper attention to drainage.

Fig. 1 shows the Rarus shaft, taken 18 mo. after the guniting had been completed. The cages have been operated about one third of the time; the shaft is dry and there has been but slight movement of the
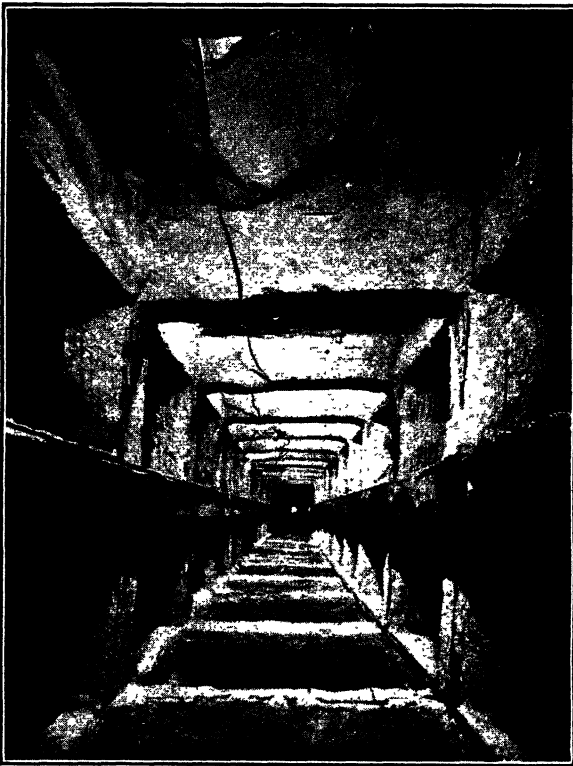
FIG. 2.—TRAMWAY NO. 1 SHAFT AFTER 3 YR. HARD HOISTING SERVICE.

timbers. The depreciation of the gunite is hardly noticeable. Fig. 2 shows the Tramway No. 1 shaft after 3 yr. hard hoisting service. There have been occasional deluges of "copper water" in the shaft from fire-fighting operations; ground pressure is heavy at several points. The effect of these conditions on the gunite can be seen by comparing Figs. 1 and 2. It will be noticed that the gunite is cracked and that the timbers are exposed in small patches; this, however, does not seriously affect the fireproof qualities of the covering as a whole. A timbered

surface tunnel under a railroad fill had been gunited, but the gunite covering was damaged by workmen, who found it necessary to remove numerous small portions of gunite. The flames from an adjacent surface fire swept through the tunnel for 30 min., melting the lead packing from the joints of a 16-in. cast-iron air line. But an inspection of the tunnel showed that though the timbers had been burned away to a depth of 1 or 2 in. wherever the gunite had been removed, there was no tendency for the fire to spread behind the gunite covering.
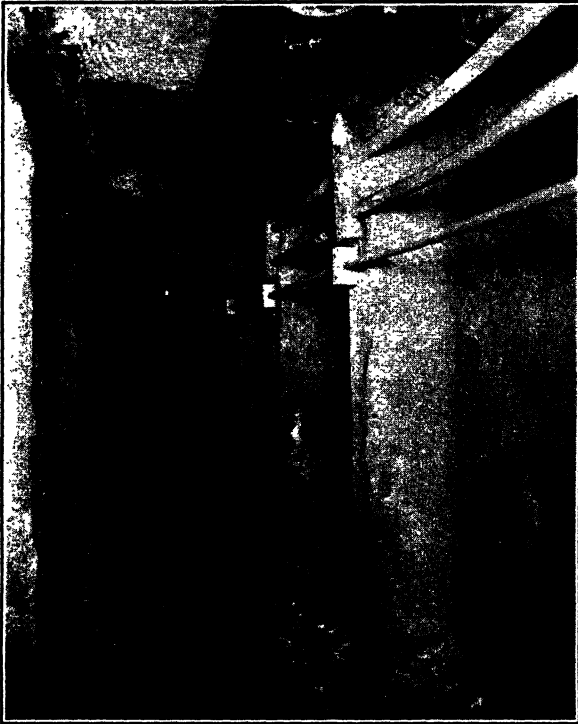
FIG. 3.—GUNITED DRIFT CONTAINING POWER CABLES.

Gunite has been applied to all wooden structures adjoining the mine shafts; including ore-bins, subways, etc. The timbers and lagging in the shafts, skip chutes, and stations have been covered with gunite; at intervals of 100 ft., the lagging has been removed and the concrete covering carried back to the rock walls, so as to hermetically seal the timbers into sections (see upper foreground, Fig. 2). This should prevent spread of fire in the timber between the rock walls and the fireproof covering. In the underground pump stations, the timbers have been gunited as have also the approaches for a distance of 50 ft. or more.

Rooms for switch panels have been cut in the walls adjacent to the pump stations; the supports for the roof and walls of the switch rooms are made of structural steel and concrete. Transformer and motor-generator sets are also isolated in rooms supported in the same manner. All crosscuts or drifts carrying power cables are gunited for the entire length of the contained cables, see Fig. 3. Where such crosscuts are untimbered, the rock walls are covered with a 1-in. coat of gunite; this prevents slacking of the walls and roof. The underground fan stations are fireproofed with gunite. Where possible, timbers are left out and the walls covered with gunite. The air passages leading from the shaft to the large booster fans are gunited throughout. The guniting is carried for 75 ft. beyond the exhaust of the fans. The smaller fans, Nos. 2½,
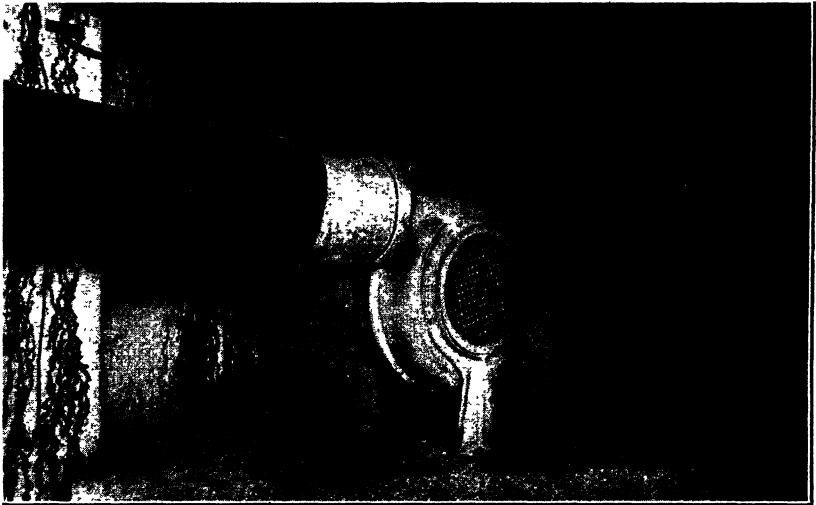
FIG. 4.—GUNITED FAN CHAMBER

4, and 6, Sirocco, are enclosed in a fireproof covering, and one or more sets of adjoining timbers are covered with gunite, see Fig. 4 The timbers are gunited wherever oil or blasting powder are stored. The total amount of surface gunited to date is as follows:

	SQUARE YARDS
Shafts	227,503
Stations	46,426
Pump stations	5,179
Fan stations	3,072
Transformer stations	777
Ore-bins and surface buildings	14,624
Subways or surface tunnels	22,369
Total	319,950

The electrical equipment of the mines is being thoroughly overhauled. The older power cables have been replaced by new steel-armored cables; the lighting and signal cables have been renewed; especial attention has been given to the manner of making taps in the power lines and to the junction boxes and wiring on the stations. This work is described in detail by C. D. Woodward. [2]

The underground water system has been extended so that, in all the larger mines, $1\frac{1}{2}$ or 1-in. water lines have been carried into each drift, crosscut, and stope. Water is obtained from the city water system; it is carried down the shaft in 3 or 4-in. water columns. At each underground station, connections on the water columns are provided for standard $2\frac{1}{4}$-in. fire hose. Hydrostatic pressure in the water columns is kept down by means of reducing valves or tanks, placed at intervals of 400 ft. difference in elevation. In the larger installations, tanks are preferable, as they provide a reserve water supply. They are circular wooden tanks 9 ft. in diameter by 7 ft. in height, and hold 3000 gal. of water. They are equipped with a float-and-valve arrangement which automatically regulates the supply and maintains the water level within fixed limits. A hose house is located on the surface close to each shaft, where a supply of about 1500 ft. of $2\frac{1}{4}$-in. hose is kept available for underground use. Pyrene extinguishers are placed on each station and near all electrical equipment.

Control of ventilation is obtained by making surface fan installations reversible and by iron or cement doors placed in the main crosscuts and in the connections between mines. These doors are set in concrete frames and can be made gastight by chinking with fireclay or cement.

Two rescue stations are maintained by the Anaconda Copper Mining Co. in the Butte district. No. 1 rescue station, situated in the Anaconda mine yard, is equipped with twenty-one sets of Paul breathing apparatus and serves the mines in the central and western portions of the district. No. 2 rescue station, situated in the Tramway mine yard, serves the mines on the eastern slope of the Butte hill; it is equipped with fifty sets of Paul breathing apparatus. It is also used as a training station; see Fig. 5. Some nearby surface workings available for training work make it possible to give training under approximately fire conditions. A crew of from six to thirty helmet men is maintained at each mine; these men work as miners or timbermen when not actively engaged in fire fighting. Recruits are trained by the instructor, who is in charge of the station; the course prescribed by the U. S. Bureau of Mines is used. Men who have been previously trained are given 6 hr. practice once every 6 mo. About 400 trained helmet men are in the Butte district.

. [2] Power Distributing System for Deep Metal Mines. See p. 86.

The older system of fire patrol was organized upon the principle of having from two to six underground watchmen (firebugs) at each mine. These men patroled the workings continuously while on shift; their hours were arranged so as to include the intervals between the operations of the two 8-hr. mining crews. The beats were necessarily long and several hours elapsed before they could cover the workings deserted by the day or night shift. Under the present system, each shift boss is responsible for the fire patrol of his beat. With the aid of an assistant, chosen from among the repairmen of his crew, he covers his beat immediately after the shift goes off; the assistants are trained helmet men. They go on shift 1 hr. later than the regular crew and receive a bonus of 10 per cent. over miner's wages. Under the operation of this system, a force of
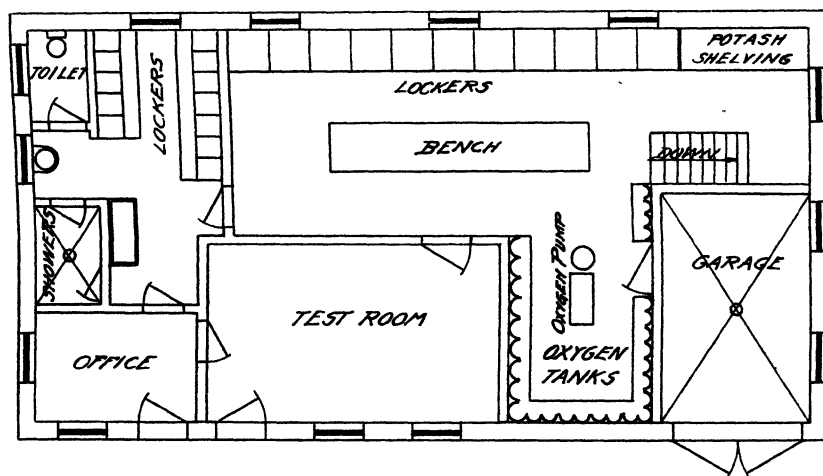
[.Fig. 5.—Plan of Tramway rescue and training station; storeroom for supplies is in basement and classroom is on second floor.

from eight to twelve men patrols each mine immediately after the shift goes off. The entire mine is covered within an hour.

The foregoing preventive measures have proved their efficacy. The gunite coating has stood up under several tests. In the Leonard shaft in July, 1920, the insulation of an old power cable broke down and an arc set fire to the shaft ladder; several lengths of the ladder were burned before the fire was put out. The shaft timbers, however, which had been gunited, were not damaged. A number of motors on small ventilating fans, which were surrounded by gunited timbers, have been short circuited and burned up without damage to the mine timbers. On numerous occasions, the advantage of having an underground water supply available has been demonstrated. Fires have been put out before they had time to spread and get out of control. In 1917, at the Tramway mine, the prompt closing of the proper firedoors turned a rush of gas into

the upcast shaft and enabled the men on shift to be removed from the mine without a single casualty; without such action, a serious loss of life might have resulted. The mine bosses should become thoroughly familiar with the ventilation of their mine. It is good practice for a mine foreman to give his bosses theoretical problems in ventilation, based on the supposition of a fire in different parts of the mine. The methodical training of Butte fire-fighting crews has resulted in high efficiency, and they have been frequently called upon for service in neighboring states, where they have done most satisfactory work.

Power Distributing System for Deep Metal Mines

BY C. D. WOODWARD, BUTTE, MONT.

THE Anaconda Copper Mining Co. purchases 25,000 kw. of electric power for its mining operations at Butte, Mont. This power is delivered, over duplicate feeders, in the form of 60-cycle, 2400-volt, three-phase current, to five compressor plants, where most of the energy is used for driving air compressors. At the compressor plants are located the switching apparatus for controlling the 2400-volt power and distributing it to the various mines. At these points, also, there are a number of motor-generator sets for converting the alternating current to 275-volt direct current for the haulage systems. Pole lines convey the power from the compressor plants to outdoor transformer substations at most of the mines. When it is not economical to distribute the power at the lower voltages in the mines, transformers and motor-generator sets are placed at strategic points underground.

All power is purchased on a maximum-demand basis; therefore there is installed at one of the compressor plants an eight-element totalizing graphic wattmeter. This instrument records simultaneously the value of all power taken at the numerous points of delivery, which not only eliminates the diversity factor but permits load dispatching between compressor and pumping plants, and thus materially reduces the maximum demand from the power system. As most of the mine hoisting is done by 90-lb. compressed-air winding engines, the air-compressor equipment for the winding engines and mining operations is quite extensive. The standard air compressor has a rating of 7500 cu. ft. per min. of free air, and most of the compressors are directly connected to synchronous motors. The synchronous motors are designed for 80 per cent. loading power factor, thus facilitating power-factor correction of total mine load. Consequently by coöperation with the power company, the voltage of the power received at the compressor plants is constant.

In the mines there are installed an induction motor load of approximately 10,000 hp., and 4500 kv.-a., in transformer capacity. It is the practice to install 2200-volt motors in the mines, except those rated below 50 hp., hence there are in operation underground about 5000 hp., in the

higher voltage motors, and approximately the same horsepower in motors wound for 440 volts. Approximately 50 per cent. of the motor load in the mines is required for pumping operations; the remainder is used for mine ventilation and haulage purposes.

When the mines were shallow, pumping and ventilation did not present a serious problem, so the underground electrical installations were comparatively simple. As the mines were developed and deepened, greater pumping facilities were essential. Steam pumps were replaced by those driven by electric motors; and the installation of a number of fans underground made necessary the distribution of large quantities of electric energy.

In the early electrical installations, lead or cotton braided armored cables were clamped by wooden cleats to the timbers of the mine shafts. A number of cable failures and fires resulted from this practice, because the cable armor did not have sufficient mechanical strength to support the cable or resist the impact when struck by material falling through the shaft. The cable armor was frequently pulled apart or punctured, allowing water to enter; in time this caused insulation failure and, consequently, destroyed a section of the cable or ignited adjacent combustible material. The safety and reliability of the electrical service in the mines is dependent, to a great extent, on the electrical cable installations; therefore a number of improvements have been made.

At all mines where power is required underground, the operating shafts have a gunited pump compartment provided with ladders; these compartments are used for the feeder cable ducts. At each level where the cables are tapped, an offset or fireproof room is constructed and furnished with latticed iron doors. This room contains the junction boxes, switches and transformers. To provide sufficient mechanical strength for the electric cables, when they are supported in a vertical position, a lead-jacketed, steel-armored cable has been adopted. In shafts that are upcast or where the cable is subjected to acidulous waters, each strand of the steel armor is lead covered. This type of cable is supported by cast-iron cleats covered with cement, or a hardwood clamp fixed to the mine timbers with bronze lagscrews, set approximately 10 ft. apart.

Cables installed in the pump compartments are always exposed and liable to an accident in the shaft. Where greater safety and reliability is required, the electric cables should be put down through diamond-drill holes. Considerable care must be exercised in locating these holes, to avoid passing through "bad" ground. The Anaconda Copper Mining Co. at Butte has placed the drilling of these under the supervision of the geological department and this department has been successful in locating the holes so that they pass through fresh and unaltered granite; when a fault is encountered, a transfer across the fault is made on the mine level.

A typical modern electrical underground installation is located at the High Ore mine. Here the switching apparatus and cable installation is particularly interesting, because a departure has been made from the usual practice. As shown in the one-wire diagram, Fig. 1, six main circuits from the compressor plant are taken down the mine and terminate on the 2800-ft. and 3400-ft. levels. The pumping plants are on three

LEGEND
A-600 Amp. Oil Circuit Breaker, 7000 V., Overload Trip, Remote Elec. Control, Westinghouse
B- 4 in Orangeburg Fiber Conduit
C- 3 in "   "   "
D- G E Cable Support
E -3-Conductor-V C-400,000 C M-5000V Lead jacketed-Steel Armored Cable
F- 3-Conductor-V C- No.0000 5000V-Lead jacketed Steel Armored Cable
G- 3-Conductor-V C-No. O-5000V Lead jacketed-Steel Armored Cable
H-2-Conductor-V C-250,000 C M-1200V Lead jacketed-Steel Armored Cable
J- Home made cable supports
K-Johns-Pratt Junction Box
L- " - " " "
M- D. C. Circuit Breaker
N- G E 600 A-7500V-Truck Safety Panel
O-300 Hp Auto Starter
P- 300 Amp. G.E. FK-20 Oil Switch-2500V
Q- Pot-Head
R- Knife-Blade Disconnecting Switches

Busses in Butte Hoist Comp P.t Basement. C.T.
Mont. Power Co's. Feeders

Protective Reactance
D Drill No. 1
D. Drill No. 2
Surface Fans

Surface Splicing Chamber

500'
700'
900'
1100'
1200'
1400'
1600'
1800'
2000'
2200'
2400'
2600'
2800'
3000'
3400'

Alice
Lexington
Paulin
Belmont
Corra
Future

M R P
Shaft No.1
Shaft No.2

lead covered Cable pump Transformer

1200
20 kw
Lts. 3ф
100 kva Fans

Five 7½"-600 Gal-A C M Co-60 R P M Pumps-Lift 1000'
2-150 Hp 2200V 514 R P M Allis-Chalmers Motors per Pump
2200

20 kv-a
Lts. 3ф
200 kv-a Fans

2800
20 kva. Lts
3ф-200 kv-a Fans
Feeds 2400-3800'

four 7½"-600 Gal-A C M Co
60 R P M Pumps-Lift 600'
Two 150 H.P.-2200V-514 R P M
Allis-Chalmers Motors per Pump

3400
One 6½"-10"-300 Gal-
Deane Pump 60 R P M Pump
One 100Hp-2200Volt-590 R P M
Allis-Chalmers Motor

LIST OF 400,000 C.M.-5000V-V.C.-STEEL ARMORED CABLE

10 Reels each containing 560'-2 for drill hole surface to 500-5600 Ft.	
2 " " " 135'-2 " " " 500 to 700	270
2 " " " 215'-2 " " " 700 to 900	430
2 " " " 215'-Transfer on 900 Level	430
2 " " " 230'-2 for drill hole 900 to 1100	460
2 " " " 125'-2 " " " 1100 to 1200	250
2 " " " 215'-2 " " " 1200 to 1400	430
2 " " " 130'-Transfer on 1400 Level	260
2 " " " 160'-2 for drill hole 1400 to 1600	320
2 " " " 210'-2 " " " 1600 to 1800	420
2 " " " 230'-2 " " " 1800 to 2000	460
2 " " " 105'-Transfer on 2000 Level	210
2 " " " 230'-2 for drill hole 2000 to 2200	460
	Total 10,000 ft

4/0 5000V-VC-Steel Armored Cable
For Shaft and diam drill holes from 2200 to 2800

2-Special reels of 410' each	820'
2-Reels of 250' each	500'
2-Reels of 840' each with limit to reel diameter,	1680'
Limit of size which may be taken down shaft-40½"56"	
	Total 3000'

Joint Couplings for 400,000 C.M., 250,000 C M, and 4/0 Cables
are ordered from Standard Underground Co

FIG. 1.—ONE LINE DIAGRAM OF HIGH ORE MINE POWER CABLES AND CIRCUITS.

levels, approximately 1000-ft. apart. The five 2400-volt, steel-armored cables are tapped at these points and branches carried into the pump stations. Four of the cables used for transmitting power to the pumps in each pumping station are connected to a common station busbar. The fifth cable extends into each pump room to the transformer equipment, which delivers power for operating the fans. The power for operating the 275-volt haulage system is taken from the sixth steel-armored cable, which is tapped at the mine levels where the mining locomotives are used.

In the basement of the compressor plant there is a current-limiting reactance connected in series with busbars that feed all 2400-volt circuits entering the mine.   On the structure supporting the busbar in the com-

FIG. 2.—OIL SWITCHES AND CURRENT-LIMITING REACTANCE FOR 2400-VOLT MINE CABLES.

FIG. 3.—SPLICING CHAMBER, SHOWING POTHEAD SUPPORTING A DIAMOND-DRILL HOLE CABLE AT SURFACE.

pressor plant are the disconnecting switches and electrically operated oil circuit breakers, Fig. 2.   This apparatus is controlled from a switchboard

on the main floor of the compressor plant; the six mine circuits are connected to lead-armored cables through inverted potheads; the cables are then run through an underground fiber conduit system that ends in a splicing chamber, Fig. 3, near the mine shaft. At this point, two cables are taken down diamond-drill holes, while the other four enter a gunited cable compartment and are taken down the mine shaft. The diamond-drill holes are $3\frac{3}{32}$-in. (7.85 cm.) bore and extend to the 2800-ft. (853 m.)
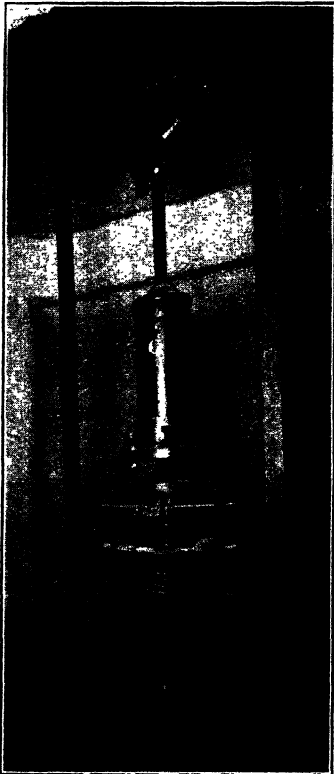
FIG. 4.—DIAMOND-DRILL HOLE CABLE
SUPPORT ON 2200-FT. LEVEL.

FIG. 5.—METHOD OF SUPPORTING SHORT
LENGTHS OF CABLE IN DIAMOND-DRILL
HOLE.

level. The cables installed in the diamond-drill holes are 2.92 in. (7.31 cm.) in diameter and made up of three 400,000 cir. mil stranded conductors, insulated with varnished cambric, for a working pressure of 5000 volts, lead jacketed, over which is a covering of galvanized steel armor bedded on asphalted jute. The cable has sufficient strength to support itself in 600-ft. (182 m.) lengths when hung vertically. For supporting the long lengths of cable hung in the diamond-drill holes, a special pothead has been used, Fig. 4; this supports the cable conductors as well as each

strand of the steel armor. To hold the short lengths of cable, a clamp like that shown in Fig. 5 was used.

Inasmuch as it was necessary to tap all the cables on various levels, junction boxes were installed and a means provided therein for disconnecting the conductors should it be desirable to test or keep a section of a cable in service. Fig. 6 shows four of the junction boxes installed on the gunited shaft timbers outside of the cable compartment, and tapped for branch circuits leading into the pump station on the 1200-ft. level. In the cable compartment, the cables are supported by cast-iron clamps set
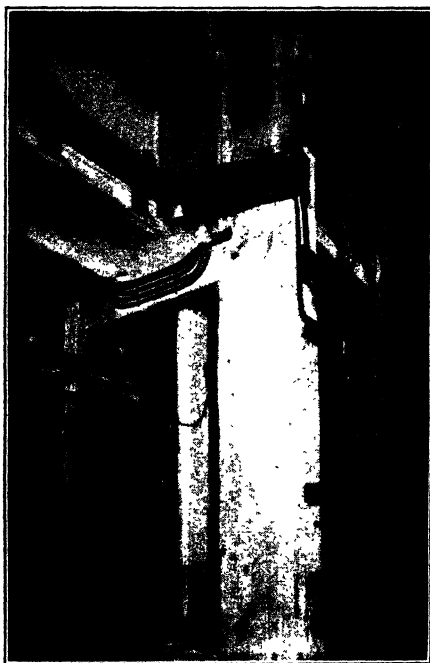
FIG. 6.—JUNCTION BOXES SUPPORTED ON SHAFT TIMBERS ON 1200-FT. LEVEL.

on 15-ft. centers. During the winter months, ice accumulates on the cables from the collar of the shaft to a depth of about 300 ft.; in this part of the shaft the cables are supported every 5 ft., to withstand the additional ice load.

On the first two pumping levels of the High Ore mine, there are installed five, quintuplet, vertical, 600-gal. (2271 l.) 1000-lb. (453 kg.) pumps. The third, or 2800-ft. level, as shown in Fig. 7, has an installation of four pumps. Each pump is gear driven by two 150-hp., 2200-volt, induction motors. For the control of the circuits supplying the pumps, a simple switching arrangement at each station permits power to be supplied from any one or all of the four cables. Should all

four circuits be operated in parallel and a disturbance occur on any one circuit, directional relays are provided at various points to select and disconnect the defective cable.

Normally the four cables are charged, one circuit furnishing power for each pumping level, with a spare circuit for any one of the pumping stations, if a cable should fail. However, two cables will transmit the present pumping load indefinitely. The switching apparatus is installed in concrete rooms that open into each pumping plant. The switch-boards are the safety-first or removable-truck type. Fig. 8 illustrates one installation on the 2200-ft. level. Each switchboard panel is mounted on a four-wheeled truck, together with the main switch, instru-
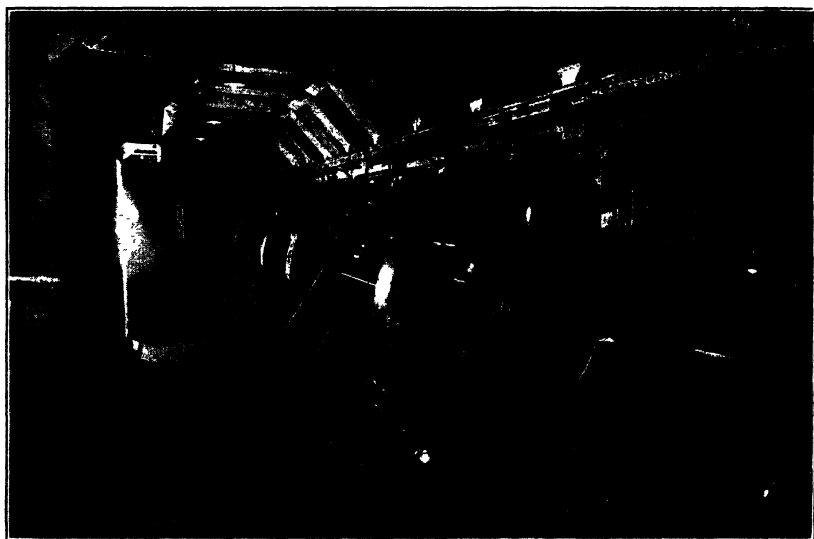
FIG. 7.—ELECTRICALLY DRIVEN PUMPS, 2800-FT. LEVEL OF HIGH ORE MINE.

ment transformers, and wiring connections; and is pushed into a tank-steel compartment containing the busbars and cable terminals, where provision is made for connecting all the apparatus into circuit. The live parts are enclosed and cannot be worked upon unless the truck is with-drawn from the compartment; this necessitates opening the oil switch before any circuits are exposed for inspection. In addition to being safe to operate in underground stations, this type of switch gear permits the installation of a moderate capacity oil switch in an enclosed cell without using concrete, which sometimes is undesirable because of ground move-ment. A switchboard panel is shown open for inspection in Fig. 9, while the others are connected into circuit. The cables enter the switchboards from below the floor at the back of the steel compartment, where there is a concrete runway in which the cables lie embedded in loose sand. This

method is also used on mine levels where the diamond-drill hole cables cross faults, except that the loose sand is covered with a 6-in. slab of con-
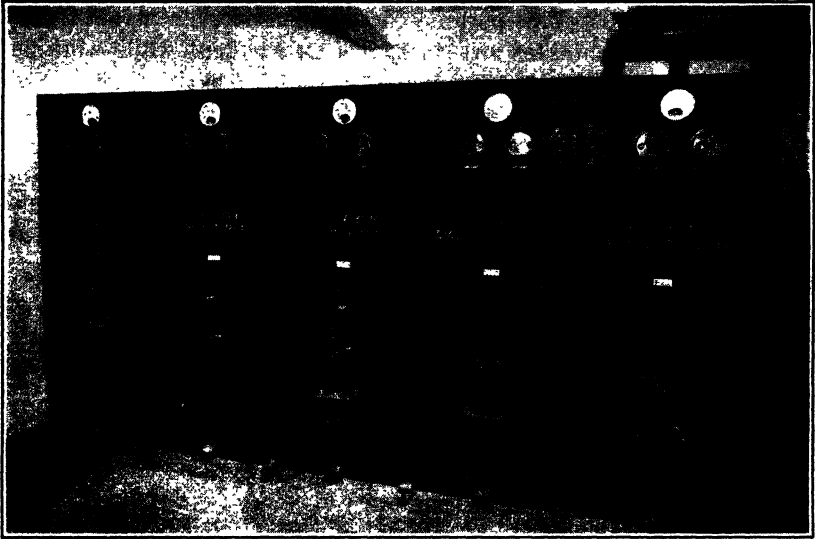
FIG. 8.—TRUCK-TYPE SWITCHBOARD ON 2200-FT. LEVEL OF HIGH ORE MINE.
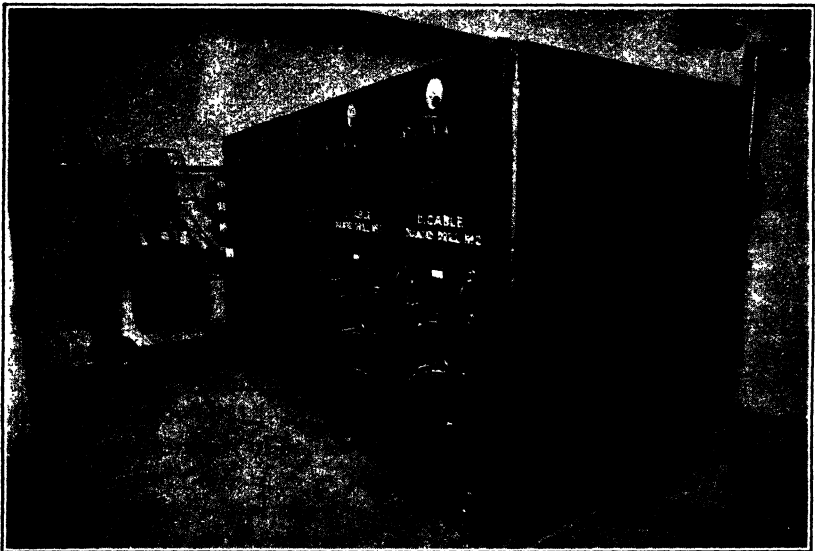
FIG. 9.—TRUCK-TYPE SWITCHBOARD WITH SWITCH GEAR WITHDRAWN FOR INSPECTION ON 2200-FT. LEVEL OF HIGH ORE MINE.

crete. This arrangement permits the ground to move without stressing the cable sheath, and in the event of a cable failure, none of the other cables are disturbed.

The control of each pump is accomplished by one 300-hp. compensator, which start the two motors simultaneously. Each compensator is connected to a busbar through disconnecting switches; the arrangement is shown in Fig. 10. This permits the compensator's removal from the live circuits and eliminates any danger of an accident while working on the electrical or pumping equipment. The compensators are furnished with ammeters, overload relays, and push buttons for stopping the motors. To expedite the finding of a defective pump on the pumping system, an ammeter is desirable in each motor circuit. This allows the pumpman to determine easily the faulty pump.
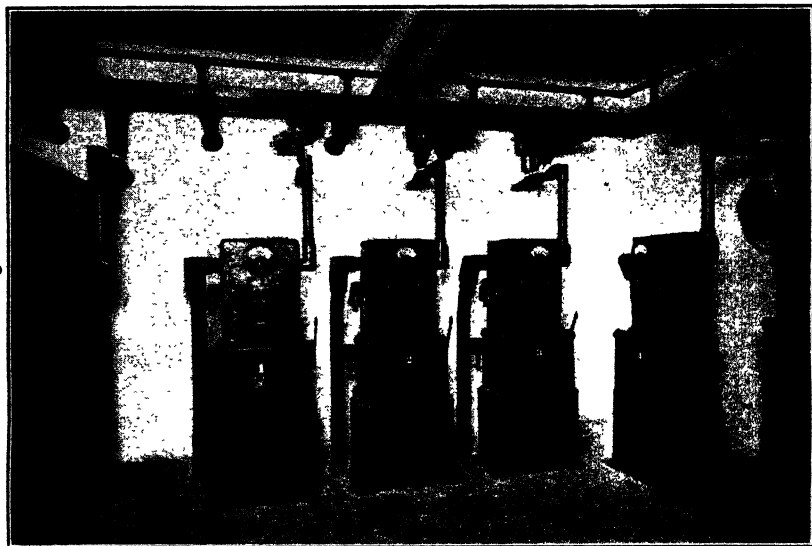
FIG. 10.—COMPENSATOR INSTALLATION FOR PUMP MOTOR ON 2800-FT. LEVEL OF HIGH ORE MINE.

Although the haulage system is operated over a ground return, the 275-volt direct current supplied to the trolleys throughout the mine is transmitted over a two-conductor cable. The haulage cable is insulated with varnished cambric (for a working pressure of 1200 volts), lead jacketed, and steel armored. This cable is installed in the gunited-shaft compartment beside the 2400-volt power cables. At each level where mine locomotives are used, the cable is brought out to the station and passed through junction boxes equipped with two terminals. One terminal is grounded for the rail connection, the other is arranged for a single-conductor lead-jacketed cable that connects, through a safety switch and magnetic blow-out circuit breaker, to the trolley wire. The safety switches, like the junction boxes, were designed for mine installations where there is considerable drip. The safety disconnecting switch is enclosed in a cast-

iron, weatherproof box, with a shed top. The handle is arranged so that it can be locked in the off-position.

To protect the cable from insulation stresses due to static or high-voltage disturbances, oxide film lightning arresters are connected to the busbars at each pumping station. This type of lightning arrester is better suited to underground mining installations than the electrolytic, because there is no electrolyte to evaporate in the warm underground pump rooms and charging operations are not necessary. For the 2400-volt circuit, twenty oil circuit breakers rated at 800 amp. and 15,000 volts were used. In addition 57,000 ft. of lead-jacketed steel-armored cable, together with the usual complement of wiring devices, were necessary. The basis for this electrical distribution system was eleven pumps, but provision was made in the initial installation for double this number.

Electric Signal Installations in Butte Mines

By C. D. Woodward,* Butte, Mont.

THE subject of electric signals for the despatching of mining cages through shafts has received considerable attention recently from various mining companies. The Anaconda Copper Mining Co. has found it necessary to develop a system that is reliable, safe and simple in operation, and easy to repair. The movement of all mine cages or skips is under the direction of a station tender, who rides on the cages when men or supplies are being lowered or raised; when ore is hoisted, he is stationed at a skip pocket for loading and despatching the skips.

For the station tender's use, two signal systems are installed in each hoisting compartment, and he is the only employee authorized to use them. One system is known as the "AC" or station bells, as it is operated from the 110-volt, 60-cycle, power service; the other system, called the "DC" or shaft bells, takes its energy from closed-circuit primary batteries. As a supplement to the two main signals, a buzzer system is installed. The general arrangement is similar to the main signal, except that there is a mine buzzer at each station. When a signal switch connected into the buzzer system is pulled down, all buzzers are operated, thus notifying the station tender that he is wanted at some particular level.

Placed on the station posts, and within easy reach of the men on the cages are the signal pull switches for the main signals. The station, or "AC," pull switches are furnished with a short rope. as shown in Fig. 1, while the shaft, or "DC," pull switches are equipped with ropes that extend down the shaft past the next station. To avoid all confusion in the use of the three signal systems, a standard arrangement has been adopted in all mines for the location of the pull switches. Fig. 2 illustrates the position of the pull switch at the various mine stations for the main signal systems, but it does not show the buzzer pull switch; this is fixed to the roof timbers about 25 ft. back from the shaft. If the skip pockets are a considerable distance below the station, a pull switch for each main shaft compartment is connected to the "AC" system and placed at this point. However, if the distance is short, a light cord is attached to the station pull switches and dropped down to the skip pocket.

* Chief Electrical Engineer, Anaconda Copper Mining Co.

To facilitate the sending of signals from any point in the shaft during the repairing or inspection period, the long rope hung from the "DC" pull switches at the stations is used.

On one side of the engineer's stand are three bells for the three sets of station pull switches; on the other side are three additional bells for the three sets of "DC" shaft pull switches. The bells have different tones so that the engineer can tell whether the signal is sent from either of the
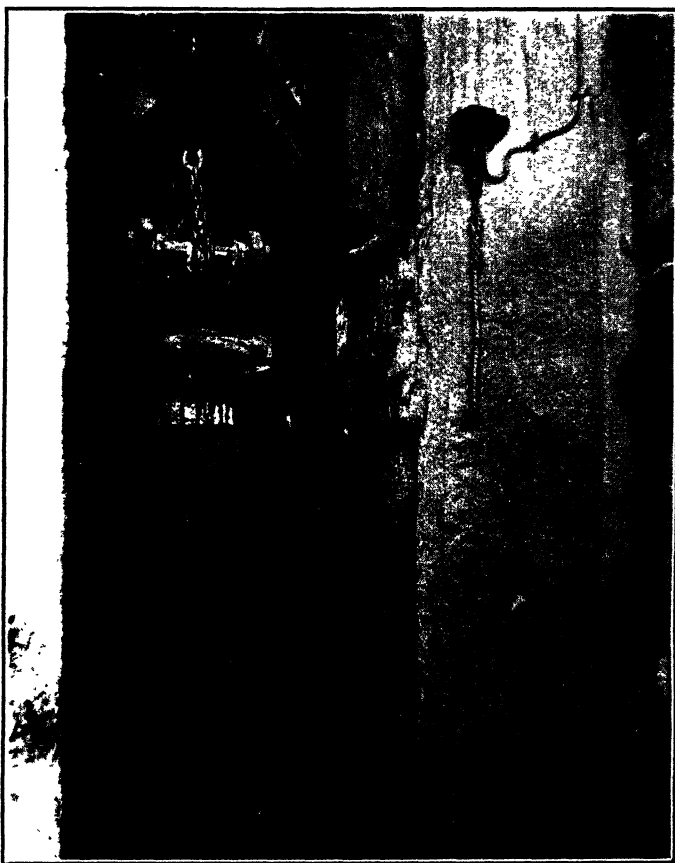
Fig. 1.—Typical signal-pole switch-station installation.

main shafts or the auxiliary shaft. In addition to these six bells, there is a buzzer operated by the buzzer signal system, Fig. 3.

The electrical connections for the "AC" signals and the buzzer system are shown diagrammatically in Fig. 4. Each bell on the engineer's platform is connected in parallel with a lamp, and in series with a double-pole switch. If one of the signal wires or station pull switches becomes short circuited or grounded, thus causing the single stroke bell to be held up, it can readily be disconnected from the circuit. The lamp, however,

will remain lighted and indicate when the trouble is removed. The buzzer system also has this arrangement, except that no lamp is used; but a push button is provided on the engineer's platform so that a return
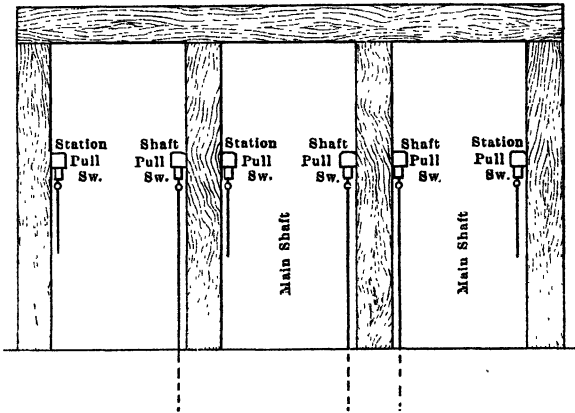
Note:-Pull Switches for Buzzer System are Set
25 Ft.Back from Main Shaft on each Station.

FIG. 2.—MINE SIGNAL SYSTEM OF ANACONDA COPPER MINING CO.

signal may be despatched to each station in the mine, which permits the engineer to call the station tender. Only four wires of each six conductor cable are in use for the main signals; the other two wires in the "AC"
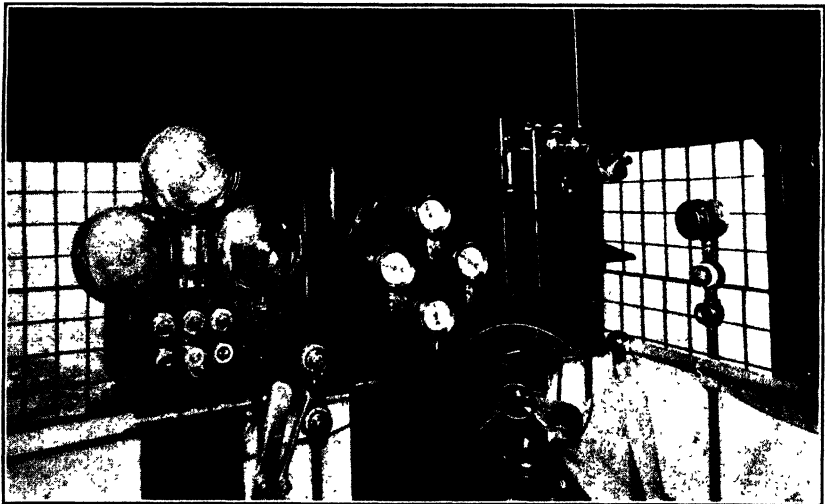
FIG. 3.—BELLS, TELEPHONE, AND BUZZER INSTALLATION ON HOISTING ENGINEER'S PLATFORM.

signal cable supply energy to the lamps in the junction boxes, while in the "DC" signal cable a 2500-ohm mine telephone is bridged across the spare circuit at each mine station. The connections for the "DC,"

or shaft signals, are similar to the "AC" signals shown, except that an Edison primary battery of approximately 40 volts supplants the transformer. At times, a special relay has been connected between the bells in the engine room and shaft signal circuits of the "DC" system, thus reducing the number of batteries. The relay, however, introduces complications and is not reliable; therefore its use has not been general.

Starting from the engineer's platform, where the signal bells, lights and buzzer are located, three cables extend into the mine and terminate in junction boxes at the first station. From this point the cables are

Wires 5 & 6 are used for
Phones or Lamps in Boxes
L-Lamp
P.B.-Push Button
B.-Junction Box
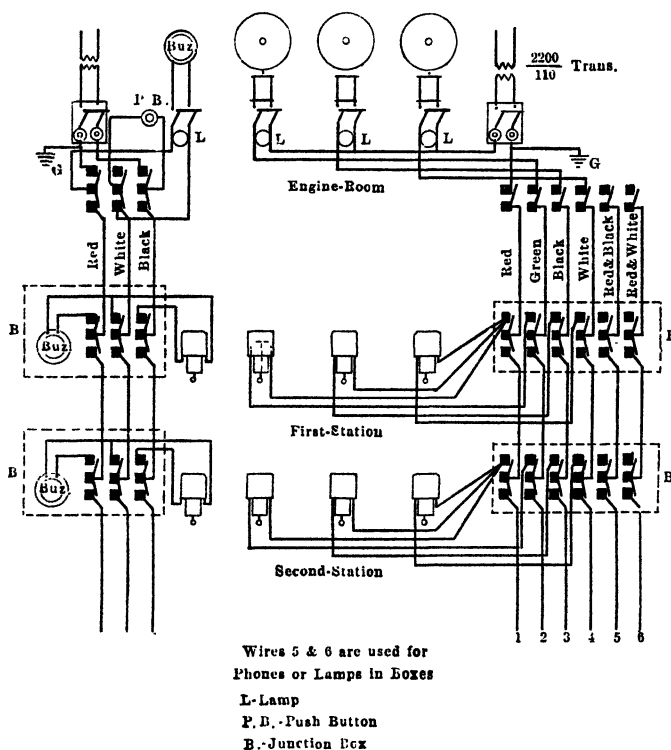
Fig. 4.—Standard arrangement of pull switches.

continued from station to station, passing through junction boxes, Fig. 5, wherein they are tapped for the pull switch connection. Each main signal cable contains six flexible No. 12, A.W.G. conductors, while the buzzer cable has three No. 8, A.W.G. flexible conductors. The individual cable conductors are insulated with rubber for 600 volts, and have different colored cotton braids for distinguishing purposes. The multiple conductor cables are protected by a $\frac{1}{16}$-in. lead sheath, a layer of soft jute run through hot tar, and two steel tapes applied in the reverse direction, over which is a layer of hard twisted jute saturated with asphalt coated

with soapstone.   The two wires extending from the junction boxes to the pull switches are solid No. 12, A.W.G. insulated with rubber having different colored cotton braids, laid parallel and covered with a $\frac{1}{16}$-in. (1.58 mm.) lead sheath.

The vertical cables in the shaft are in a separate compartment and supported by cast-iron cleats set on 30-ft. centers, except within 300 ft. of the collar of the shaft where ice may accumulate during the winter; here the cleats are placed 15 ft. (4.5 m.) apart.

The junction boxes are of cast iron, with a hinged door that is held closed against a tubular rubber gasket by wing nuts.   The cable entrances are also sealed, thus making the boxes watertight.   When the junction box is used for the main signals, it is equipped with an oiled slate panel
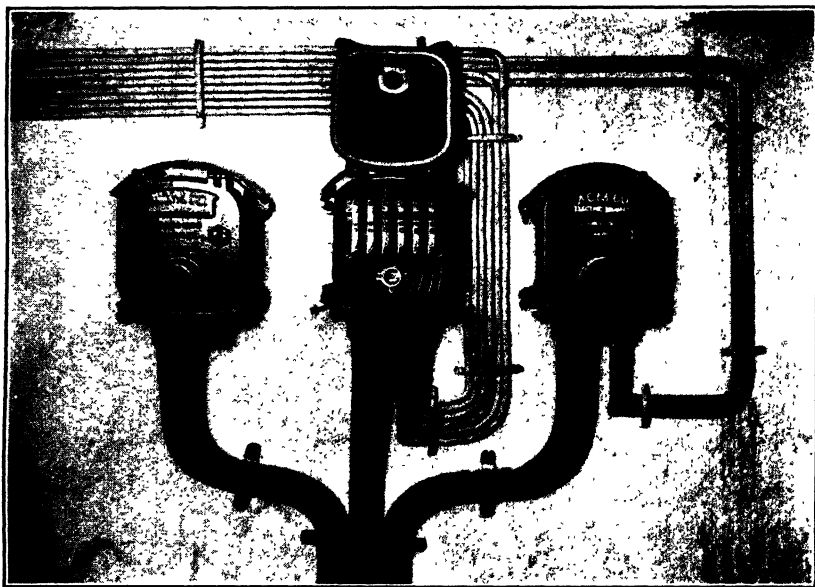
FIG. 5.—JUNCTION-BOX INSTALLATION ON MAIN AND BUZZER CABLES, SHOWING ONE BOX OPEN FOR INSPECTION.

on which are mounted brass terminals with bronze spring test links for each wire of the six conductor cable.   The boxes for the buzzer signals are similar, but the panels are arranged for a three-conductor cable and contain an alternating-current mine buzzer without the cover.   For the purpose of keeping the interior of the junction box dry and to indicate a power failure on the "AC" or buzzer system, a 10-watt lamp is placed in a standard receptacle set directly back of a boss on the door, fitted with a green glass.

The layout of the signal systems and the special apparatus described have been developed after considerable study to meet the service conditions in the Butte mines.

Electric Haulage Systems in Butte Mines

By C. D. Woodward,[*] Butte, Mont.

Prior to 1902, the tramming of ore from the stopes to the shafts, in the Butte mines, was done by man or animal power, but the demand for greater tonnage and the need for more improved methods of tramming the ore resulted in the installation of the compressed-air locomotive in a few mines. Some of the objections to animal haulage were thus eliminated, but this locomotive was not generally adopted, because of a number of limiting features. It was superseded by the electric trolley locomotive.

The early types of trolley locomotives were not satisfactory from the operators' viewpoint, but they had many economic advantages and after a number of years, in which improvements were constantly made in the structural details, they became dependable, so that the use of animals for main haulage in the mines has now been practically abandoned.

There are now operating in the Butte mines of the Anaconda Copper Mining Co., 185 trolley-type locomotives, each of which has replaced five animals. It cost approximately $50 per month to keep an animal, hence the economies that resulted fully justified the expenditure of electrifying the haulage systems.

The haulage conditions in this company's mines possess many advantageous features that are not always found in underground mine workings. The drifts and crosscuts are large, approximately 5 ft. (1.5 m.) wide by 7½ ft. (2.3 m.) high, and have curves of no less than 18 ft. (5.5 m.) radius; and the roadbeds of the drifts and crosscuts have a uniform grade of 0.5 per cent. toward the shaft, or in favor of the loaded trips. In all of the mines, except two, the track gage is 18 in. (45.7 cm.), and all tracks in the main haulageways are laid with 25-lb. (11.3 kg.) rails. The stations at the shaft are large, well-lighted, clean, and cool. The average temperature of the stations, however, is about 75° F. (24° C.), while the average temperature in some of the drifts and crosscuts is 92° F. (33.5° C.).

As the orebodies lie at varying distances from the hoisting shafts,

[*]Chief Electrical Engineer, Anaconda Copper Mining Co.

the length of hauls are not fixed, although the distance in all of the mines over which the ore is trammed is about the same. While the average haul is 1500 ft. (457 m.), a number of hauls are approximately 3000 ft. long.

Usually a train consists of ten loaded, roller-bearing cars; the number of cars per train is limited by the space available at the stations. Each car weighs, when empty, 900 lb. (408 kg.) and holds 14 cu. ft. (0.39 cu. m.) of ore, which weighs about 1800 lb.; thus the total weight of a train, without the locomotive, is 13½ tons. Normally, ten trips of ten cars of ore each are required per shift of 8 hr. on each mine level; this does not include, however, the tramming of waste or supplies. When waste is handled, the amount required on each level is variable; it also varies

Fig. 1.—3½-TON, 18-IN. GAGE, TROLLEY MINE LOCOMOTIVE.

from day to day. In general, the waste is hauled during the time the ore haulage is light, as the material is usually trammed from the shaft to the stopes for back-fill, which is against the grades. During a working day, the locomotives are in service 16 hr., hauling an average of 245 loaded cars, and returning the same number of empties.

Under normal conditions, a mine locomotive built for a drawbar pull of 750 lb. would perform the work that has been outlined. To obtain the mechanical strength for resisting the abuse to which a mine locomotive is always subjected, larger equipment has been installed. The trolley locomotives now in service have a drawbar-pull rating of 1450 lb. at a speed of 5.3 mi. per hr., and weigh 3½ tons. Most of them are the bar-steel frame style, Fig. 1, equipped with two 250-volt motors, that drive, through single reduction gears, four chilled cast-iron wheels.

The chilled cast-iron wheels have a greater life than steel-tired wheels, for the wheels come in contact with considerable acidiferous mine water. The current for the motors is obtained from a trolley wire placed in an inverted wooden trough. This trough, or guard, is placed at one side of the tracks, and is made of 2 by 8-in. strips for the back and 1 by -in. boards on each side.

In operation, the modern mine locomotive is reliable, for any delays in operation are generally attributable to broken trolley poles, wheel wear, or damaged wiring from the trolley wheel to the controller. The repairing of these parts is nearly the only cost of locomotive maintenance. In the mines, the trolley wire and rail bonds are not only a nuisance but
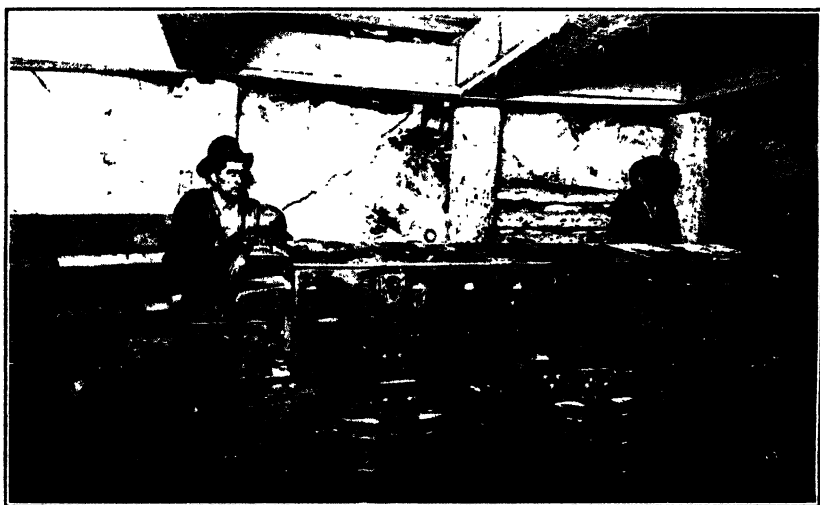
FIG. 2.—5½-TON, 18-IN. GAGE, STORAGE-BATTERY LOCOMOTIVE.

they reduce the safety factor, and contribute to delays and expensive repairs chargeable to the tramming systems.

As the trolley wire is a fire hazard and a potential danger to the workmen and as it limits the use of a trolley locomotive, particularly in the gathering operation, the storage-battery locomotive has been installed in a number of the Anaconda Copper Mining Co.'s mines. The first storage-battery locomotive was put into operation underground during the summer of 1919, and in a short time demonstrated that it was a marked success under all the haulage conditions. As a result, thirty additional machines were purchased, and most of them are in operation. The standard type of storage-battery locomotive adopted, Fig. 2, has a 3½-ton chassis, as this weight is consistent with good mechanical
        It is equipped with a compartment for 105 cells of Edison A5

batteries, held in 15 trays, making the total weight of the locomotive 5½ tons. It is rated at 1500-lb. drawbar pull at 3½ mi. per hr. on level track, with a maximum drawbar pull of 2500 lb. for starting and accelerating. The length of the locomotive is 139 in. over the bumpers. The width is 36 in., and the height over the battery compartment at the outer edge is 47 in. To discourage the carrying of drill steel and other supplies upon it, the top of the battery compartment has been provided with a curved cover, raised 3 in. along the center line parallel with the wheels. This feature increases the total height to 50 in. above the rails.

The locomotive's equipment consists of two 125-volt, ball-bearing motors, connected to the driving axles through single-reduction gears. The speed control is obtained by connecting the motors in various series-parallel combinations through a resistance by means of a drum-type controller provided with four series and three parallel positions. A reversing switch and a switch for the headlights mounted on each end of the locomotive are also contained in the controller case. Protection for the motor is provided by a circuit-breaker under the motorman's seat, while the batteries are protected by a fuse connected into each side of the circuit. For recording the amount of charge and discharge of the battery, a Sangamo ampere-hour meter is mounted on each locomotive. The meter is adjusted to run slow on charge, so as to compensate for battery losses and restore the proper amount of energy. In addition to the electrical equipment, the locomotives are provided with devices for sanding the rails when running in either direction; a standard automatic locking, screw-type, brake mechanism; and a 25-lb. locomotive bell.

Until recently, the switchboard used in connection with charging storage-battery locomotives underground was built by mounting standard knife switches, rheostats, and instruments on a slate panel. Consequently, this type of charging panel exposed the live electrical circuits at numerous points where they may come in accidental contact with the attendants. Furthermore, the open type of charging panel is a greater fire risk, and some skill is necessary to operate the switches and rheostats in proper sequence. To reduce to a minimum the faults of this style of charging panel, a self-contained dead-front panel unit has been designed for this purpose. Fig. 3 shows such a unit installed in one of the mines. The general design consists of a steel-plate front, equipped with push buttons, rheostat hand wheel, flush-type volt and ammeter. The rheostat and contactor for adjusting or opening and closing the charging circuit are mounted directly back of the panel on the supporting structure. All of this mechanism is enclosed on all sides under a drip-proof roof. The sides are screened and furnished with locks. Free access to the apparatus back of the panel is permitted to the repairmen, but tampering by unauthorized persons and accidental contact with live parts is prevented. The rheostat is electrically interlocked with the contactor so that it

cannot be connected into circuit until all resistance is in series with the battery.

When it is desired to charge a locomotive battery the sequence of events is as follows: After the battery is connected into circuit by a flexible cable and plug, the rheostat handle is turned in a counter-clockwise direction until it strikes a stop. The starting push button is then pressed and the rheostat handle rotated in a clockwise direction until the ammeter indicates the proper charging rate. The voltmeter is arranged

FIG. 3.—DEAD-FRONT SAFETY CHARGING PANEL.

to show either battery or line voltage. When it is desired to discontinue the charge, the "off" push button is pressed and the cable disconnected from the locomotive by withdrawing the plug from the receptacle. Provision has also been made for disconnecting the battery from the charging circuit automatically by means of a contact on the ampere-hour meter. This arrangement trips out the contactor on the panel after the predetermined number of ampere-hours charge.

The operation of the storage-battery locomotive in the mines has been found particularly advantageous indeed. The absence of the trolley

wire and track bonding has saved considerable money, not only because of the cost of maintenance, but because of the constant cost of advancing trolley wire in the drifts and crosscuts as mining progresses. The ability of the locomotive to operate over temporary tracks has resulted in a large saving of manual labor; this feature eliminates hand tramming from the face of the drift or crosscut to the main haulage system.

It has been found that the cost of a trolley wire and track-bonding installation on the long hauls is about equal to a set of storage batteries for a locomotive, hence there is little difference in cost between a trolley locomotive and a storage-battery locomotive installation. The cost of maintenance of the two types of installation has been in favor of the latter, because of the labor necessary to keep the trolley wire in alignment
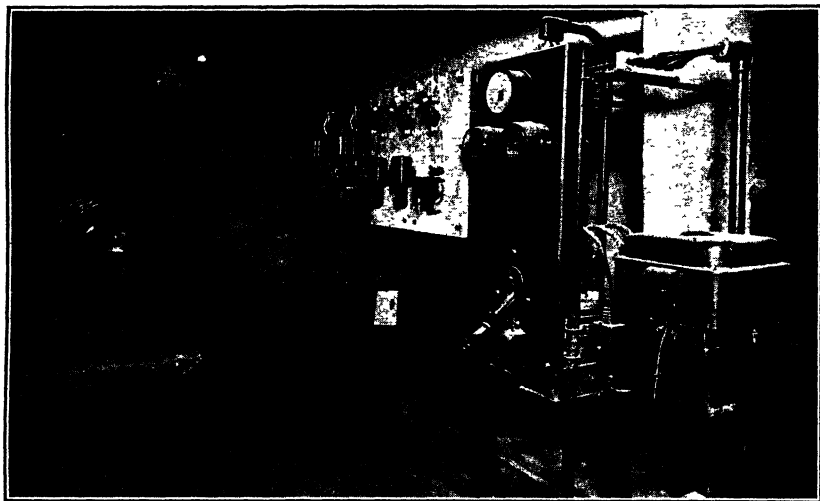
FIG. 4.—UNDERGROUND AUTOMATIC 250-VOLT 75-KW. SUBSTATION.

with the rails, due to the movement of ground in a number of the mines, and the constant repairing of the guard strip over the trolley wire. There are now operating in the mines 22 storage-battery locomotives, hauling an average of 245 loaded cars and returning the same number of empties in a 16-hr. day. This is the average duty of a locomotive.

As the storage batteries are not removed from the locomotive chassis for charging, this work is in charge of the motorman. The method of charging is as follows: At half past two in the morning, which is the end of a day's work, the locomotive is connected on the charging circuit at the charging station. The current is left on until the integrating ampere-hour meter indicates that the batteries are charged, when they are disconnected by the station tender. The time required for charging is approximately 3 hr. When the locomotive is taken from the station by

the first shift, sufficient energy is stored in the battery to operate the locomotive until noon. At that time it is given a ½-hr. boosting charge, sufficient for the afternoon shift, which quits work at 4 o'clock. The batteries are then given a charge of 2½ hr., and another ½-hr. boosting charge is given between 10 and 10.30 o'clock. This charging arrangement has proved satisfactory to the operating departments. Some water must be added to the battery electrolyte every other day; this is a simple matter, as each charging station is provided with a Universal battery filler. About 15 min. is required to replenish the water in each locomotive battery.

As a number of the mines are using both types of locomotive, the energy for charging purposes is supplied from the 275-volt service cables that distribute power to the mine levels for the trolley systems. The power for the haulage is furnished by motor-generator sets, which are rated at 75 kw. In a number of places the unit is installed in the mines. Fig. 4 shows an automatic underground substation which is set about 100 ft. back from the main shaft, and is enclosed on the side and front by expanded metal. The shift boss usually starts or stops the apparatus by a push button outside of the enclosure.

DISCUSSION

WILLIAM KELLY, Vulcan, Mich.—May I suggest that lead-covered wires have been used in mines in some cases because of a misunderstanding of the word "underground?" Ordinary lead-covered wires, as I understand it, are particularly suited for installations when the wires are laid in the ground, so that the lead covering is always grounded, but in the mines the wires are really an aerial installation. They may be below the general surface of the ground but they are not underground in the sense of being in the ground.

GRAHAM BRIGHT, East Pittsburgh, Pa.—One of the principal reasons for putting lead on the cable is to be absolutely sure that moisture is excluded. In many cases underground, the air is saturated. Without the lead covering, the cable, by alternate heating and cooling, may develop cracks and as soon as a crack starts moisture will enter. Some recent developments in the use of very high-grade rubber may produce a high-voltage cable for underground service without the use of lead.

The illustrations in this paper indicate that a high grade of workmanship was used in installing the equipment, to which fact a large part of the success in these operations is due.

HOWARD N. EAVENSON.—I had occasion, about three years ago, to investigate the installation in coal mines of cables carrying 2200 volts; in two cases lead-covered cables had been used but had proved unsatis-

factory. In each case the cable had been suspended. We were going to install a 6600-volt three-conductor cable, but after learning the conditions in the other mines and after consulting with some of the cable companies, we installed a cable that was covered with, I think, 30 per cent. rubber. That has been in use for about 2 years and I do not think it has given any trouble. The cable companies said that this cable would give much less trouble than the lead-covered cable.

C. E. ARNOLD, New York, N. Y.—In one mine where compressed-air locomotives were used and 17,000 or 18,000 tons of ore were hauled daily, the track grade was 0.4 per cent. in favor of the load; the average tramming distance was 0.48 mi.; and the power consumption was 0.37 kw.-hr. per ton, so long as the locomotives were kept in first-class condition. Before the adoption of compressed-air haulage at this mine, bids were received from makers of trolley equipment, who would not guarantee a much better power figure than this. My recollection is that their guarantees were about 0.33 kw.-hr. per ton of ore hauled.

However, power consumption was not the governing consideration in the adoption of compressed-air haulage; it was safety to life and limb. A mine nearby, where about 12,000 tons of ore were being trammed daily, was equipped with a well-maintained trolley system. But the tramming costs showed that the compensation for personal injuries was abnormally high on account of continued accidents caused through car loaders and timbermen coming in contact with the trolley wire at the various ore chutes. As a result of the experience at the first mine mentioned, those in charge of the latter mine scrapped their trolley system and installed a compressed-air haulage system.

GRAHAM BRIGHT.—I agree, that there are places in mines where mules should be used, and that there are some isolated cases where the compressed-air locomotive will prove more suitable than the trolley or storage-battery type, but in the large majority of cases the electric locomotive will show far greater economy than any compressed-air locomotive.

It is rather surprising that the small locomotives used in Butte have replaced five animals. The author states that it costs approximately $50 per month to keep an animal, and in addition to this saving there is a saving of four drivers.

About three years ago I checked up the maintenance cost of the trolley type of locomotives for the Butte mines and was surprised at how low it was. Besides indicating excellent care of the locomotive, it showed that the conditions of operation were not severe, which makes an ideal application for the storage-battery locomotive. The information given on page 102 illustrates that the service conditions are rather light for the trolley-type locomotives. These locomotives were, however, the smallest

that could be economically built, so that there is no criticism of the application.

The actual time of operation and the time of charging each locomotive are given, but it would be interesting if the actual number of kilowatt-hours taken out of the battery during each period and also the number of killowatt-hours restored to the battery were stated. This would give a good idea of how the battery is being worked and the life that could be expected.

The first storage batteries to be applied to mining conditions used 60-volt motors. The present standard is to use 80-volt motors. It has been felt for some time that 100 volts or 125 volts would be preferable and these higher voltages may some time be adopted. The selection of 105-cell Edison batteries indicates that the author is aware of the tendency of the times and is taking advantage of what may become a future standard.

It would be interesting to other mine operators if actual figures in regard to cost of operation and cost of upkeep of both types of locomotives could be given.

The safety charging panel illustrated is a great advance in the charging equipment for mine service; it is in marked contrast with the usual type found in coal-mining installations. The automatic substation has become firmly established in coal-mining practice and undoubtedly a number of automatic substation equipments will be installed in metal mines upon the active resumptions of metal mining.

GERARD B. ROSENBLATT, San Francisco, Calif. (written discussion).— This paper shows on how large a scale mechanical haulage has been introduced by a progressive mining company. It is encouraging to see the way in which storage-battery haulage has been received. Too many mine operators consider the storage-battery locomotive a delicate piece of mining machinery the use of which may be expensive. It is therefore pleasing to receive a definite statement from one in a position to know that the initial costs per ton-mile for storage-battery haulage, under the conditions that maintain in Butte, are not much different from the initial costs of trolley haulage.

The author states that the cost of maintenance of the two types of equipment is somewhat in favor of the storage-battery locomotive; is the basis of comparison the cost per locomotive or the cost per ton-mile of ore hauled? The only fair method of comparing haulage costs is on the basis of ton-mile of ore hauled and we would be glad to receive cost figures, expressed in cents per ton-mile, on both trolley and storage-battery haulage under Montana conditions.

In a paper presented before the San Francisco Section of the Institute, the writer showed the desirability of analyzing haulage problems on

the basis of a study of costs expressed in ton-miles of ore hauled and submitted a tabulation, or segregation, of costs that could be advantageously adopted in a majority of cases, and which would permit a fair comparison to be made between alternate methods of haulage in the same mine, or between haulage systems in different mines. Briefly, the segregation is as follows:

ANALYSIS OF HAULAGE COSTS, IN CENTS PER TON-MILE

Capital expense (interest and amortization on first cost)
Operating expense
    Operating labor and material
    Power
Maintenance expense
    Locomotive maintenance
    Car maintenance
    Trolley, track and station maintenance
General expense (supervision and proportionate overhead expense)

Several of the larger western companies have adopted this system and from the cost data accumulated some definite facts can be determined. It would be most interesting if the author could present similar figures on haulage in the Butte mines.

Several progressive ideas are mentioned. One is the type of charging panels used. Because battery voltages and battery charging circuits are usually of low voltage (that is, under 600 volts) it has been the practice not to consider them hazardous. As a matter of fact, in a mine, and frequently in surface industrial plants, 275 volts, or 250 volts, can be just as fatal as 440, 500, or even 2200 volts. It is courting danger to consider any exposed electrical circuit of 110 volts or over safe to handle without taking every precaution possible.

Another progressive idea is the automatic operation of the motor-generator substations supplying direct current for the haulage system. Such substations have been in vogue with electric interurban railways for some years, but it has been only lately that their economy and convenience have been recognized by mining companies. During the past year several score of such substations have been installed by mining companies, particularly in the coal fields; they have proved to be reliable, convenient, and economical. The automatic features are not complicated and are readily applicable to motor-generator sets and rotary converters. Many mining companies could reduce their haulage costs and simplify their operations by making their converting equipment automatic, or at least semiautomatic, in operation.

Another interesting point is the use of a high-voltage battery, so as to charge with maximum economy from ordinary trolley voltage. The usual trolley voltage in metal mining is 250–275, but the storage-battery locomotive is usually equipped with a battery that discharges at approxi-

mately 80 volts and requires 125–140 volts for charging; therefore, when a battery is charged from a 250-volt trolley circuit a definite proportion of the power drawn from the line must be wasted in resistance, or a special motor-generator set must be used. It is true that for a given kilowatt-hour capacity, a battery to charge from a nominal 250-volt circuit costs more than one suitable for charging from a 125-volt circuit, but an analysis in many cases will show that the power wasted and other costs involved in charging the lower voltage battery will soon pay for the increased cost of the higher voltage battery. It was not so long ago that standard storage-battery locomotives were designed for a battery discharge voltage of 60 volts; manufacturers and users will soon recognize the advantage of higher discharge voltages, possibly around 100 to 125 volts. The higher voltage has many advantages to offset the increased first cost of battery equipment involved therein.

K. A. PAULY, Schenectady, N. Y. (written discussion).—The automatic substation was first developed to meet the requirements of railway systems to supply outlying districts where the car service is infrequent. Perhaps because of the similarity between surface lines and underground haulage systems, mining electrical engineers became interested in the automatic substation soon after its application to railways and in the coal fields approximately thirty of these stations, averaging approximately 200 kw. each, have been purchased, many of which are in successful operation. The automatic features embodied in the control of these stations can be varied considerably to meet the special requirements of a particular installation.

Briefly stated, the advantages of automatic substations are as follows: Protection is provided at all times. Maintenance costs are reduced. Distribution voltage is improved. Feeder loss is decreased. Reliability is increased. Minimum operating personnel is required. Wear and tear on substation machines are reduced by elimination of unskillful handling.

Wire Rope and Safety in Hoisting at Butte Mines

BY W. N. TANNER* AND F. C. JACCARD,† BUTTE, MONT.

(New York Meeting, February, 1922)

THE wire-rope hoisting conditions at the mines of the Anaconda Copper Mining Co. in Butte, Mont., are very severe because of the conditions under which it is necessary to operate. A study was made, in the fall of 1919, for the purpose of changing these conditions in order to obtain longer rope and sheave life. There are at present in Butte hoisting engines equipped with wire rope as follows:

    9 main hoists using flat ropes.
    7 main hoists using 1½-in. rope.
    5 main hoists using 1¼-in. rope.
    6 shafts using 1⅛-in. rope.
    2 chippy hoists using 1¼-in. rope.
    9 chippy hoists using 1⅛-in. rope.
   15 shafts using 1-in. rope.
    7 shafts using ⅞-in. rope.
    1 shaft using ¾-in. rope.

In addition, there are a number of sinking engines using ⅝-in., ¾-in., ⅞-in., and 1-in. (15.9, 19, 22, and 25 mm.) rope.

The flat ropes run from ½ by 7½ in. (12.7 by 190.5 mm.) to ⅜ by 4½-in. (9.5 by 114 mm.) and are used on the older hoisting engines, which are set comparatively close to the shaft. Because of swelling and moving ground at some of the shafts, these ropes present certain difficulties in operation that are more pronounced than with round ropes; they are, therefore, being superseded by round ropes as newer engines are being installed. They are, however, much more easily inspected and repaired than round ropes.

In general, the practice in Butte is to use hoisting engines equipped with 12-ft. (3.7 m.) cast-iron drums, smooth faced, for 1½-in. rope and 6-ft. cast-iron drums, smooth faced, lagged with ⅜-in. steel plate for 1¼, 1⅛ and 1-in. (31.7, 28.6, 25 mm.) rope in main and chippy shafts. Cast-iron head sheaves, 12, 10, and 6 ft. in diameter, with grooves

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machined $\frac{1}{32}$ in. greater radius than the radius of rope, are used. Special "blue center" steel ropes, either regular or Lang lay, according to conditions, are used; they are designed to withstand excessive abrasion and are wound on drums in from two to four layers. On the smaller drums, approximately one half of the first layer remains on the drums at all times, in order to insure the rope reeving properly; this portion is tapped into place with a light hammer while the rope is being installed. The ropes are fastened to the cages by clamps. The ropes are kept well lubricated at all times, with a mixture of Whitmore's cable dressing and pine tar, using 1 gal. of pine tar to 1 lb. of cable dressing.

The ropes are inspected and tarred regularly, the interval depending on the amount of work the rope is doing and other conditions of operation. Whenever necessary, from 12 to 25 ft. are cut from the drum end to change the point of greatest wear, which occurs at the point where the rope climbs from the first to the second layer at the end of the drum. Some ropes are turned end for end; this serves to equalize the wear on the rope and to prolong the life From 10 to 12 ft. are cut from the cage end of the rope at intervals to compensate for the wear of the clamps. Ropes are changed whenever they begin to show signs of excessive wear, an extreme flattening of the outside wires, or the appearance of several broken outside wires in a strand. As far as possible, ropes are changed before they approach a dangerous condition.

The use of straight-faced drums has been general throughout the Butte district and is one of the conditions that tend to make rope service severe. Of late years, the cost of drum repairs has been high. All ropes are attached at the inside flange of the drums. On some hoists, the left rope is the overwind; on others, it is the underwind.

The head sheaves are so located that a line through the center line of the sheaves and parallel with the center line of hoist will strike t he drum from $\frac{1}{2}$ in. from the inner flange, as at the Pennsylvania hoist, to $15\frac{1}{2}$ in. from the inner flange, as at the Never Sweat hoist. The space between this line and the inner flange must be filled with rope at all times, for, if unwound in operation, this portion of the rope will not wind uniformly when hoisting the load. The sheaves are lined so that the centers of the grooves are in line with the center of the drums.

With 6-ft. drums, the general practice is to allow one-half of the first layer of rope to remain on the drum at all times, for because of the peculiarities of round wire rope, this portion of the first layer will not wind uniformly when hoisting the load. The first layer must wind uniformly on the drum as, with the straight-faced drum, the first layer of rope must form the grooves for the succeeding layer, and these grooves must be uniform te guide the remaining layers.

On shaft hoists using but two layers of rope, this condition might perhaps be overcome in part by using the lay rope required by the direction of rotation: but with four layers of rope on the drum the untwisting

action of the rope will occur on two of the layers.   Right lay ropes are used in practically all these installations, regular or Lang lay being used as the particular conditions may require.   For example, the Badger State shaft hoist has the left rope the overwind and the rope travels from right to left on the first layer.   When a new rope is placed in service, it has considerably more stretch than a rope that has been in service for a long time and in lowering the skip to the bottom level the rope will be elongated about 15 in. (38 cm.) by the weight of the rope hanging in the shaft.   After the skip is loaded, a further elongation of about 32 in. will result, making a total elongation of 47 in.   This elongation increases the length of rope lay, and as the rope is securely attached at drum and skip, the rope cannot rotate but has a pronounced tendency to do so in a direction to untwist the rope; therefore the rope has a tendency to travel to the left, causing the rope to wind non-uniformly until the fleet of rope on the drum counteracts this tendency, which occurs when the middle of the drum is reached; beyond this point the rope will wind uniformly.

On installations using small smooth-faced drums and where the fleet angle is severe, ropes have a tendency to interlock when winding on the drum.   This, together with the smaller diameter occasioned by the elongation of the rope under load, causes the groove made by the first layer of rope to be slightly less in pitch than the diameter of the rope, and causes the second and succeeding layers to be pinched together.   This condition causes pronounced abrasion of the rope in service.

On smooth-faced drums, when installing a new rope, it is necessary to start the rope on the drum winding a true helix.   To accomplish this, a starting piece must be introduced at the flange where the rope is attached. This was accomplished by using a manila rope starting piece, but it was found impossible to fit this manila rope on the drum to give a true helix. Now all drums are equipped with a steel starting piece machined to form a true helix.   This starting piece extends completely around the drum and gives very satisfactory results.

When installing new rope, the rope is fastened to the drum. through the inside flange, by suitable clamps and the idle rope is wound on the drum by tapping each wrap with a light hammer and block to make it bear against the adjacent wrap of rope.

As the unit radial pressure of the rope on the drum, in the case of the 1⅛ and 1¼-in. rope, winding four layers on the 6-ft. drums, is considerably over 2000 lb., grooves have been worn on the face of these drums and in some cases cracks have formed across the face.   In order to obtain longer life, the drums are being lagged on the face with ⅜-in. steel plate.   It will be noted that, in the case of the 1¼-in. rope, the drum diameter bears to the rope diameter a ratio of 57.6 to 1, which is under the generally accepted ratio of 60 to 1; this tends to make the conditions very severe on the rope.

Considerable thought has been given lately to the subject of equipping the engines with grooved drums because of the easier rope conditions to be obtained from these drums. With properly designed grooves, the rope has a continuous support, and the pressure of the rope on the drum is practically uniform, instead of being concentrated at the crown bearing points of the rope. Also there is no pinching and interlocking of the rope on the grooved drum. To determine the effect of the grooved drums under the severe conditions obtaining at Butte, the Badger State main hoist has just been equipped with 6-ft. grooved drums on which will be 1¼-in. wire rope, 6×19, Blue Center steel, special Seale construction, regular lay.

On the Butte installations using 12-ft. smooth-faced drums and 1½-in. wire rope, the unit radial pressure with three layers of rope is less than 2000 lb.; we have experienced no trouble with these drums, although some of the faces are beginning to show evidence of grooving after 15 yr. of service.

Because of the lack of ground around some of the shafts, and other local conditions, the fleet angles vary from 1° 6' to 1° 27', for hoists having 6-ft. drums and 1¼-in. rope, and from 1° 12' to 1° 53' for hoists having 12-ft. drums and 1½-in. rope. The two main hoists with 6-ft. drums and 1⅛-in. rope have fleet angles of 1°10' and 1° 24'.

The head sheaves are made of cast iron 6, 10, and 12 ft. in diameter, with the grooves machined to a radius 1/32 in. greater than the radius of the rope, and are carefully balanced to run perfectly true. Semisteel sheaves were tried but abandoned because it was impossible to get the material in the sheave uniform.

The radial pressure of the rope on the sheave is independent of the arc of contact and is determined by the rope tension and the sheave diameter. It is also directly proportional to the sheave diameter. Therefore the larger the sheave diameter for a given rope tension, the lower is the radial pressure of the rope on the sheave. The radial pressure divided by the rope diameter gives the unit radial pressure. The maximum unit radial pressure of a rope on a cast-iron sheave should not exceed 450 lb. with regular-lay rope, otherwise the sheave wear will be excessive. For Lang-lay rope, this pressure may be 10 per cent. more, or 495 lb.

On all main hoisting shafts, the unit radial pressures of the rope on the sheaves varies from 246 to 392-lb. values, which are sufficiently low to allow economical use of cast-iron sheaves. On the Steward and Original hoists, where Lang-lay ropes are used, the unit radial pressures are 467.5 and 475.8 lb., which are permissible values. On the Badger State hoist, where 6-ft. sheaves were used with 1¼-in. rope, the unit radial pressure was 612 lb., causing short life and giving continual sheave trouble. These sheaves have been replaced with 10-ft. sheaves, so that the unit radial pressure is well below 450 lb. and no further trouble is anticipated.

On the Chippy hoists, which use 1⅛ and 1¼-in. rope with 6-ft.

sheaves, the conditions are more severe, and some light cast-iron sheaves, bicycle type, with cast-steel liners, properly machined, in the bottom of the grooves are being tried.

It was the custom to install a sheave when necessary and keep that sheave in service until the metal at the bottom of the groove became worn to a thickness of about $1\frac{1}{2}$ in. Little or no attention was paid to the size of the worn groove nor to the wearing of the groove to one side or the other of the center line. The result was that a rope which was new at the time the new sheave was installed gave fairly long life but the life of each succeeding rope was considerably shorter. As the groove wears in the sheave, the diameter of the groove is worn to the size of the worn rope, which is less than that of a new rope, so that the new rope is squeezed into the narrow groove, causing excessive abrasion.

At present, a sheave is installed when a new rope is put on and, when the rope is worn out, the sheave is taken down, the groove machined to the correct size, and the sheave later replaced for a new rope; therefore, the new rope starts in service in a proper-sized groove. At the time this sheave is put in, the alignment of the sheave with the center line of the drum is checked and adjusted. This is expected to result in longer rope life. Due to swelling and moving ground around the different shafts, the head frames shift and settle from time to time, causing the sheave to fall out of line with the drum on the hoist and the rope to wear on the side of the groove. This causes excessive abrasion of the rope.

Monthly records are kept of the thickness of the metal in the groove of the different sheaves; also of the shape of the grooves in the different sheaves and we are working at present on the development of a templet gage for taking these measurements.

Considerable variation has been found in the life of the different ropes on the different shaft hoists, due to the varying conditions of operation and mechanical equipment; it is expected that considerably longer life will be obtained from the ropes under the improved mechanical conditions when the mines are again operating normally. The life of the ropes during the present period of depression cannot be taken in comparison with the life noted under operating conditions.

The cages and skips are built in our own shops, of the best materials, and are made heavy in comparison with the load they will handle. Before being put into service, they are painted one coat of red lead and one coat of Harrison's antoxide paint to protect them from corrosion The top deck of each string and the skip deck are equipped with heavy cast-iron or steel dogs actuated by heavy springs and held away from the guides by the weight of the string; in case of a broken rope or any slack rope they will release and grip the shaft guides, preventing the string from falling to the bottom. When the mines are operating, the cages

Data Pertaining to Main Hoisting Engines
Anaconda Copper Mining Company
Butte, Montana. Nov. 23, 1921.

Legend

Cages		Skips	
Top deck	= 1	Top deck	= IA
Middle deck	= 2	Skip	= S
Bottom deck on East		Load	= L
Colusa & Pilot Butte.		Fleet angle	= φ
Weight 1700 & 2000 lb. respectively.		Wt. of load per car	= LC

Main Hoists	A	B	B'	C	C'	D	E	F	G	H	X	Cages 1	Cages 2	LC	IA	S	L	Rope Dis. In.	Rope Lo Ft.	W Feet	Z Feet	φ °	φ '
NeverSweat	−10.8	134.7	129.7	102.4	113.2	5.4	6	10	56	33	15½				3,000	5,880	9,400	1¼	3,200	3.875	172.15	1	7
Mountain Con	22.7	142.8	137.3	63.8	41.1	5.0	6	10	56	33	12½				3,700	5,880	8,100	1¼	3,600	3.625	143.3	1	27
Original	6.1	132.1	127.1	115.0	108.9	5.0	12	10	63	50	5				3,300	6,500	12,000	1¼	3,900	4.833	167.37	1	39
Steward	14.8	123.6	118.6	101.0	86.2	5.2	12	10	63	40⅝	5				2,650	6,250	12,000	1½	4,300	4.833	146.6	1	53
Poulin	4.8	128.0	123.0	36.0	31.2	4.1	6	10	60	36								1	2,300	4.042	126.8	1	34
Tramway	7.9	156.6	150.6	92.4	84.5	4.9	12	12	58¾	52½	2¼	2,600	2,000	2,500				1½	3,300	4.708	172.68	1	49
Berkeley	24.5	186.7	181.7	77.6	53.1	4.1	6	10	56	33¾	12⅝				3,762	8,100	12,000	1¼	3,200	3.657	189.3	1	6
Mountain View	−21.1	219.7	213.7	80.1	101.2	4.1	12	12	68¼	52	3				3,000	5,880	8,100	1½	3,200	5.437	236.45	1	19
Pennsylvania	−34.8	173.4	167.4	98.6	63.8	4.9	12	12	66	51¼					3,500	6,500	12,000	1½	3,200	5.437	179.1	1	44
Leonard	−9.1	191.1	185.1	140.1	149.2	4.1	12	12	64	48½	½				3,500	6,500	12,000	1½	3,200	4.979	237.7	1	12
E. Colusa	10.11	169.1	165.1	85.0	74.8	4.1	6	10	56	34	12	2,800	1,700	2,500	3,762	8,225	12,000	1¼	3,200	3.666	180.38	1	10
Badger State	6.1	155.6	152.1	79.5	73.4	4.1	6	6	56	33⅜	12¼				3,000	5,880	8,100	1½	3,500	3.646	168.88	1	12
Rarus	22.2	192.5	188.5	99.0	76.8	4.6	12	12	67¾	48	3				3,500	6,500	14,000	1½	3,500	5.375	201.69	1	32
Pilot Butte	12.2	135.0	130.0	84.9	72.7	4.1	6	10	56	33¾	12¼	3,000	2,000	2,500				1⅜	3,500	3.646	148.9	1	24

and skips are inspected daily by the blacksmiths and boilermakers; all loose bolts are tightened and all small repairs are made. If any large repairs are necessary, the cages and skips are removed and new ones installed; this is done to keep the equipment safe and in working order and prevent as far as possible any accident due to failure of this equipment.

The principal hoisting engines are equipped with power-operated post brakes lined with wooden brake blocks and power-operated clutches. They are also equipped with the Lilly safety hoist controllers, to prevent overspeed and overwinding of the rope. Most of those in use in Butte are the old-type Lilly controllers, many improvements having been made in this device since their installation. This old type is positive in action and forms a very satisfactory safety device.

These hoist controllers are geared to each drum on a hoisting engine and are so arranged that, in case of overspeed, they will ring a bell to warn the engineer and then shut off the power, apply the brakes, and bring the engine to a stop. The brakes are applied gradually so that the load is brought to a stop without dangerous shock. This is necessary because the high momentum of the moving load must be overcome before it can be brought to a stop; otherwise the load will pull the drum through the brakes, burning the brake blocks and perhaps seriously damaging the engine and the rope. At the limit points of travel, the brake action is instantaneous. Stops are arranged at a proper distance from the top and bottom of the shaft so that in case of overspeed the load will be stopped at the landing. Stops are also arranged at the end of travel, so that the load is stopped regardless of the speed at these points. It is possible to run by the first safety stops at a low speed and then apply more power to the engine, so that the momentum of the load is sufficient to carry it into the sheaves even after the engine has been stopped at the end of travel. This is true only where the head frames are low and there is a very small distance between the dump and the sheave. In no case is the damage as great as it would be if no safety devices were employed. I believe that this has been overcome in the latest type of Lilly safety hoist controller, by causing a positive retardation in the load before this point is reached. When the mines are operating, the hoisting engines and appurtenances are inspected daily.

DISCUSSION

HORACE F. LUNT,* Denver, Colo.—Do they use detaching hooks on the cages and skips to prevent pulling the skip into the sheave?

W. B. DALY, Butte, Mont.—The Lilly safety controller prevents that. It is set at a given point so that if the engineer forgets where his cage or

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* Commissioner of Mines, Bureau of Mines.

skip is at any given time, the controller will take the lever from his hand and prevent that cage or skip from going into the dump. The device was invented by W. J. Lilly, formerly the chief master mechanic at the Anaconda Copper Mining Co.

C. L. COLBURN, Washington, D. C.—Have there been any cases in which the Lilly controller has failed?

W. B. DALY.—No; there have been times when the trip did not work, when the engineer had brought his skip slowly to the point of dumping and then forgot to reverse and pull the skip into the sheaves. The Lilly controller would stop that if there was plenty of head room.

C. L. COLBURN.—Do you think it would be worth while to put a detaching hook on a cage in addition to having the Lilly controller on the hoist?

W. B. DALY.—Detaching hooks are not necessary with the Lilly controller.

WILLIAM KELLY, Vulcan, Mich.—What is the speed of the cage?

W. B. DALY.—The law of Montana provides 800 ft. per min. when hoisting men and 2000 ft. when hoisting ore. But it makes no difference whether the speed is 2000 ft. or 5000 ft., the controller will stop the cage.

ROBERT PEELE,[*] New York, N. Y.—The Butte practice of using drums of such small diameter, relative to the depth of shaft, as to require the winding of several layers of rope on the drum, can hardly be considered common practice. In many other districts, where shafts are equally deep and the hoisting is heavy, the rule is to use drums that will avoid more than one layer of rope. Serious shock and wear occur at the point where the rope climbs from the first to the second layer at the end of the drum.

In hoisting, there are several other points where abnormal wear will always occur. Among them are the points of tangency of the hoisting rope on the head sheave and on the drum, when the cage is started from the bottom of the shaft; that is, the points where the rope bends in passing from tangent to curve. The starting stress may occur at that point repeatedly and therefore cause abnormal wear and internal stresses. Also, unless the rope is attached to the skip or cage by chains, in case slack rope is paid out, after the cage comes to rest upon the chairs, the rope will bend sharply where it enters the socket, and thus be subjected to internal stress.

It is not surprising that cast-iron drums are sometimes cracked by the excessive compressive strain produced by the coils of rope; a strain that is naturally greatest in drums having the greatest number of coils.   The authors of this paper have done well to call attention to the effect of elongation of the rope under load, thus reducing the pitch of the grooves made by the first layer of coils, and so causing abrasion by the pinching together of the rope coils of succeeding layers.

In other districts, sheaves more than 10 ft. in diameter are usually of the bicycle type, and for deep shafts, as in the Lake Superior copper district, the diameters are commonly 12 and 14 ft.   If there is any difference in the diameters of drum and sheave, the smaller of the two should be properly proportioned to the size of rope.   If the drum is larger, simply because more rope is required in a deep shaft, the sheave should be large enough to prevent excessive bending stresses in the rope. The best results are obtained when drum and sheave are of the same diameter, as the rope then winds constantly on a curve of the same radius. Manufacturers' tables usually specify the minimum allowable diameter of drum and sheave.   Some users incorrectly assume that the stated diameter is recommended for a given size of rope.

W. B. DALY.—The 6-ft. drums are being replaced by 10- or 12-ft. drums, which will relieve all of these conditions.   When these mines were first opened, the flat cable was used and, of course, the engine can be placed closer to the shaft with the flat cable than with the round rope. The new engines have been placed farther back to overcome that condition; but in one or two instances we did not have room.

B. F. TILLSON,* Franklin, N. J.—Reference is made to wear at points of overlap of the rope; a more serious result is a beating effect.   We think that we have a device to feed. the rope gradually into the second layer, but still there is a beating effect.   An example of this may be seen if a new Lang-lay rope is examined, where the wire throughout the greater length can be seen, and re-examine it later on.   Very soon after it has been put in service, it will be found that the circular section has a mushroom head where the metal has been beaten over.   Even more serious than the wear is the reduction in the diameter of rope from this cause. The metal is still there but the beating has fatigued it.

W. B. DALY.—The condition of the beat would be more than the pressure of the weight?

B. F. TILLSON.—If you beat a piece of metal with a light hammer the metal will flow and change its condition at that point.   That is the

* Mining Engineer, New Jersey Zinc Co.

point of maximum fatigue, and at that point the wires are likely to break; they show a crystalline fracture when they do.

W. B. DALY.—In determining the difference between the shock and the wear, is it possible to distinguish or determine the causes?

B. F. TILLSON.—At the point at which it occurs you can hear a certain slap.

W. B. DALY.—Have you been able to determine whether it was the shock or the wear that caused the break?

B. F. TILLSON.—It was not wear, because the metal is still there.

W. B. DALY.—The strain would flatten the rope as well as the shock.

B. F. TILLSON.—It would not change the section.

W. B. DALY—Nor would wear, necessarily, but the weight on the cable could cause the flattening effect.

B. F. TILLSON.—As you get the weight at other points, I would conclude, since it occurs at the point at which there is a slap and does not at points where there are still two layers, but without the slap, that the slap was the thing that did it and the slap was the blow.

GRAHAM BRIGHT,* East Pittsburgh, Pa. (written discussion).— Having visited many of the large hoisting installations in this country, and practically all of the hoists in Butte, the writer is particularly interested in this paper. For coal-mine hoists using conical drums or cylindroconical drums, the service is generally considered rather severe on the ropes, because of the deep grooving necessary to keep the rope in the proper position. The writer would like to know why the straight-face drum generally used throughout the Butte district is one of the conditions that tend to make rope service severe.

The general practice in the Butte district is to use two or more layers of cable on drums up to 12 ft. in diameter. In the copper country of northern Michigan, the general practice is to use drums 20 to 30 ft. in diameter and but one layer of rope. It would be interesting to know what conditions there are in northern Michigan that warrant the use of these large drums, which greatly increase the cost of the hoist installation.

If in using several layers of cable, the wear is first manifested at points where the cable climbs up the flange to start a new layer, would it not be possible to install a small auxiliary drum inside the main drum, upon which could be reeled a certain amount of cable that could be

let out from time to time, so as to present a new section of cable at the ends of the layer and thus increase the life of the cable?

It has been said that 12-ft. smooth-face drums using 1½-in. rope show evidence of grooving after 15 years' service. On many coal-mining hoists where the depth is not great and there is but one level, a single smooth-faced cylindrical drum is frequently used for balanced hoisting. Both ropes are wound on the same drum with a short distance between the ropes; in fact, one rope chases the other across the face of the drum. It is evident that a considerable portion of the drum on each side of the center receives rope wound in opposite directions, while the section of the drum near the flanges and next to the dead turns receives the rope wound in one direction only. The wear of the center portion of the drum is very rapid compared with the end portions, and has been so great in some instances that the hoist builders have furnished a renewable section in the center of the drum, which can be readily changed when worn.

Those who have traveled on any of the large hoists in Butte know that there is considerable pulsation during the accelerating period, because direct-connected steam or air hoists are used. With the electric hoist the pull applied to the cable is much more uniform, so that there are practically no pulsations during the accelerating period. Do the authors think that there would be any increase in the life of the rope if an electric motor, with its more uniform torque, were used?

CHARLES E. LOCKE,* Cambridge, Mass. (written discussion).—This paper indicates that flat ropes are being superseded by round ropes because of the difficulties of operation of the former from moving ground. It has seemed to me that the main advantage of round ropes was that the wear was less than that of flat ropes and that round-rope hoists therefore were more economical when costs were figured over a considerable period of years and account was taken of interest and depreciation. It would be interesting if the authors could supply figures showing the relative wear and cost of round and flat ropes.

The practice, in the Butte district, of winding more than one layer of rope on the drum is in contrast with the Lake Superior copper district, where the drums are made large enough for the entire rope to be wound in one layer only. One reason for the Butte practice is lack of room around the shafts. I would like to know whether the Anaconda engineers have considered the possible economy in the use of larger drums, to avoid some of the troubles mentioned in the paper which have led to some of the ingenious methods described for overcoming them.

Local wear of rope and of sheaves is met by all companies. When the length of hoist is constant, for example, in the mining of a flat seam

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of coal through a vertical shàft, it may mean considerable local wear of the rope because of the slipping of the rope on the shield during the period of acceleration. Where hoisting is done from various levels, this local wear is not so pronounced but, nevertheless, exists. Have the authors given any consideration to the use of friction head sheaves in which the wear is taken largely by a false lining instead of by the rope, similar to the device used by the Calumet and Hecla Co. on its Whiting hoists at the Red Jacket shaft? There the driving sheaves have false circumferences which save wear of both the rope and the sheaves.

Stope Cost Records and Mine Contracts of the Anaconda Copper Mining Co.

By C. L. Berrien,* Butte, Mont.

BEFORE the present company was formed, in 1916, each group of mines comprising the old organizations made its detailed daily and monthly mine cost records along the lines used before the consolidation; that is, there was no attempt made to standardize and compare the stope costs of all mines. The general cost sheets for all mines, however, were standardized, and cumulative labor cost sheets were compiled in the general superintendent's office. At most of the mines, a daily record or distribution sheet for each working place was kept, showing the number of men, hours of drilling, hours of timbering, hours of tramming, cars of ore, cars of waste, sticks of powder and feet of fuse. This information was kept by each shift boss and entered on the distribution sheet by him or, at some mines, by a tram boss. The superintendent solved his own problems under the direction of the general superintendent, but without coming in touch with other superintendents, either through records or general conferences.

Probably 95 per cent. of the stopes were square-set worked by day's pay; there were some back-filling and shrinkage stopes. Many of the development places, such as shaft sinking, station cutting, drifting and raises had always been run by contract. These contracts were given, for no definite period, at so much per linear foot and were measured monthly, or when completed, the men being paid monthly. In some cases, fairness to the employee was not maintained. On the whole, there was no standardization. There were no stope contracts, because of the objections of the Western Federation of Miners to general contracting in the mines.

CHANGES IN STOPING METHODS

Early in 1916, W. B. Daly, then general superintendent, and the author, then assistant general superintendent, visited the Southwest mining districts with the object of finding some improved mining

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* General Superintendent of Mines.

methods. On account of local conditions on the surface, as well as underground, the fact was kept in mind that no new method would be acceptable that would cause:

1. A settling of the surface or any underground workings or shafts then in use, or that would be used in the future.

2. A lowering of the grade of the ore.

3. A violation of the rules of the Safety First department.

The result of this trip was the adoption of the following methods:

1. The cut and fill, or rill system, in all veins where the ore and walls are fairly hard, whether the ore must be sorted or not.

2. The timbered rill, where the ore and walls are not hard, whether the ore must be sorted or not.

3. The use of chutes every fifth set in square-set stopes and the more general use of slides to chutes.

4. The keeping of daily cumulative detailed costs for each stope that the foreman may check the efficiency of his men.

5. The employment at each mine of men having a thorough knowledge of underground mining and accounting to maintain a system of detailed costs.

RECORD FORMS

In connection with these new methods, it was our desire to provide for obtaining, recording, and using the following information:

1. The comparative efficiency of different stoping methods.

2. The comparative cost of labor by different stoping methods.

3. The comparative cost of supplies by different stoping methods.

4. Total saving by new methods over old.

5. Efficiency, labor and supply costs of each stope in each mine and the separate details of these costs.

6. Periodical cumulative costs of all stopes.

7. A record of all mines stope costs for each foreman and superintendent.

Fig. 1 is the record sheet used at each mine. The first column contains the days of the month divided into groups of three days, the second column is for the stope number. Under tonnage are car tally, tons and tons per man; the product of a factor times the car tally gives the number of tons. Under labor are the subdivisions of breaking ground, timbering, shoveling and bulldozing, filling, and total. Under each subdivision is a column for days or shifts, and one for money. The information that goes under days is obtained by the shift boss, who enters it on the form shown in Fig. 2; the foreman's clerk completes the Fig. 1 record.

The same plan is followed in obtaining the supply cost. In this case the shift bosses use the form shown in Fig. 3, which, in turn, is handed to the clerk for figuring the money and entering on the stope cost record.

FIG. 1.

The recording of labor by the shift boss is by shifts of 8 hr. or quarters thereof. He also records this information for all other mine work. The system of obtaining the supply cost was made as simple as possible, because in a mine there is a limit to this work that is not found in a factory.

STOPE	MEN						
	Cars Ore	Drilling	Timbering	Shoveling	Filling	Hours Air	

MINE **LABOR** DATE

FIG. 2.

Air is recorded by hours, with 3 hr. for a full shift of drilling with a stoper drill, and 5 hr. for drifting drills. This is the proportion of these drills in air consumption; 15 cents was the cost of an hour of air.

STOPE	Posts	Caps	Girts	Stulls	Lagging	Special	Powder	Caps	Fuse

MINE **SUPPLIES** DATE

FIG. 3.

Timber placed is recorded only once in 24 hr. instead of by 8-hr. shifts. If recorded by each shift, duplication would result, as it is impossible for bosses to know whether all sets are placed and blocked. All

mines use the same cost per hour of air and for a post, a cap, a girt, a set of lagging, etc.

On the right hand side of Fig. 1 are columns that were added to this sheet when the practice of contract stoping was started in 1918.

Each mine, at the end of the month, has a completed stope cost record, which is sent to the office of the general superintendent, where the general stope cost record for all mines is made up; see Fig. 4. This sheet is almost a duplication of the mine stope cost record. It is made on tracing cloth, and blueprints of it are sent to each foreman and assistant superintendent. Powder, caps, and fuse are combined under explosives and net savings of the other methods over square-set stoping are shown. This system, and the methods of obtaining the information, have been used five years and have required no changes.

MINE CONTRACTS

The custom of contracting on sills, in shafts, raises, and stations still is carried on; repair work, stoping, tramming and all other work had always been by day's pay. Stope contracting, though, was not practiced until April, 1919. This system was gradually installed until, by July 1, 1919, it was an actual working success in all stopes at all mines. We had all had years of experience in contract price setting for level, raise and shaft work, so little difficulty was met in setting stope prices. Work in all stopes, raises, drifts, crosscuts, shafts and winzes is now done by contract on a cubic-foot basis, the price per cubic foot being regulated according to conditions, such as the hardness of the ore or waste, the width of the ore, the method of stoping, accessibility, temperature, and humidity.

The working places are measured each Wednesday by men from the mining engineering department, assisted by samplers and efficiency men, the work being checked by the foreman of the mine.

The contracts are made for one week, at a set price, but may be terminated after one day by either the foreman or the contractors. The men are notified of the price on Thursday morning, which is the beginning of each contract week. No change in the rate per foot is permitted during the life of the contract, but adjustments are made in special cases, which must be noted by both the engineer and the foreman. If any change in conditions arises during the week, making the price too favorable to the men and permitting them to earn too high a wage rate, the foreman terminates the contract and establishes a new rate. Likewise, if conditions unfavorable to the men arise, the rate is raised and a new contract started. These rules are enforced strictly for the purpose of preventing favoritism and seeing that the men receive a fair reward.

Fig. 5 is the form of report made out by the engineer. This record includes the contract number, working-place number, method of stoping,

STOPE COST RECORD

ALL MINES FOR

FEBRUARY 1920

TONNAGE — LABOR — SUPPLIES — NET SAVING

MINE | TONNAGE: Cars, Tons, Tons per Man | LABOR: COST PER TON (Breaking Ground, Timbering, Shoveling Ballasting, Filling), TOTAL (Days, Money), Cost of Labor per Ton | SUPPLIES: COST PER TON (Air, Timber, Timber Filling, Explosives), TOTAL MONEY, Supplies Cost per Ton, Total Cost per Ton | NET SAVING: Number of Stopes, Per Ton, Amount

SQUARE SET

Min. Con.
Bell
High Ore
Steward
Poulin
Treasury
Silver Bow
Berkeley
W. Gray Rock
Mtn. View
Pennsylvania
Leonard
West Colusa
East Colusa
Badger
Argyle
Nettie

Average cost per ton

RILL

Min. Con.
Bell
High Ore
Steward
Mountain
Mtn. View
Pennsylvania
Badger

Average cost per ton

TIMBERED RILL

High Ore
Badger

Average cost per ton

BACK FILLING

Min. Con.
Bell
Steward
W. Gray Rock

Average cost per ton

STULL

Steward
Treasury
Pennsylvania

Average cost per ton

FIG. 4.

FIG. 5.

amount of excavation or advance, size, timbered or not timbered, price, total money, shifts worked, rate per day, and remarks. One copy of this report goes through the foreman, timekeeper, and mines pay office to the general superintendent, and another copy through the chief engineer's office to the general superintendent. In the general superintendent's office, a contract record of every contract in each mine is made up, on the form shown in Fig. 6, by combining the records for all mines. The form explains itself; "T" is for timbered and "N.T." is for not timbered, there being this division of work during some weeks in some contracts.

On every second Tuesday, a joint meeting of all the foremen and assistant general superintendents is held, at which the assistant manager and general superintendent preside. The results of each contract at each mine are then read. If it appears that any set of contractors has earned too high, or too low, a rate per day, the foreman must explain whether it was on account of the conditions, too low or too high a price per foot, or the direct fault of the men. If the price was too low, it must be raised; if it was the fault of the men, they are discharged. Every detail is discussed to discover abuses and a close check is kept for the correction of any that may be found. At these meetings, matters of general interest are discussed and general operating instructions are given.

General Benefits Derived

The cost per ton of ore is less by contract. The men who are willing to work are given some individuality and the conservative men are inclined to break away from the I.W.W. theory of doing as little as possible in order to get by.

The stope cost record and contract system have given a more accurate check on the cost of underground operations than any method we have used.

By subtracting the ore tally from the engineers' contract measurements, a direct check on waste broken or tons of rock broken to obtain a ton of ore in each stope is obtained.

Measurement is allowed between the walls only, except where the wall is removed for timber. It is, therefore, to the best interests of the men not to break the walls, and the ore is kept cleaner and the average grade increased.

By taking the difference between the total labor cost of a stope on the stope cost record and the total contract money, the direct percentage of contracting in each stope or at each mine is obtained. Over 90 per cent. of the stope shifts are contract shifts. Any comparative cost data desired may be obtained directly from these records.

· The contract system tends to keep the foremen and all bosses in closer touch with the mine and creates better supervision. When contracts

Fig. 6.

are given, the list is posted at the mine showing price per foot and, when finished, full information of the results is posted showing: contract number, working place, price per foot, total shifts, total money, and rate earned. Every man at the mine is paid weekly, which accounts for a great part of the successful operation of contracts.

CONCLUSION

It is understood at all mines that any contractor is free to interview the foreman of the mine and, if a satisfactory adjustment of his complaint is not made, he may take his case to the company labor commissioner. The case is then brought to the general superintendent. During the period of general contracting from April, 1919, to date, over 60,000 contracts have been let, but only six have been brought to the general superintendent for adjustment. This record is sufficient evidence that the method is a proper one, and can only be attained by the most careful superintendence and absolute fairness to the employees.

Organization of Mine Sampling at Anaconda

By W. B. Daly* and F. A. Linforth,† Butte, Mont.

THE sampling and estimating of the orebodies in the mines of the Anaconda Copper Mining Co., at Butte, have been made a part of the work of the geological department. The wisdom of this assignment is apparent, considering the organization of that department, the nature of its records, and the scope of its regular work. In the first place, the district is divided into units of two or three mines each and a geologist is assigned to each group. His trips to all the working places are frequent and he follows the mining operations closely. His notes are graphic and detailed. In the office, he posts his notes on the standard maps and sections and on special maps for correlation purposes. With the approval of the chief or assistant chief geologist, he recommends development work underground and watches its progress. This gives the geological department a precise knowledge of the character of each orebody.

Sampling is done by one or two samplers at each mine working under a chief sampler stationed in the geological office. The chief sampler has access to all the office maps and is in frequent consultation with the geologists; his instructions to the samplers are issued with the approval of the chief or assistant chief geologist. Not only is it important that the geologic information be used to direct the sampling but the reports of the samplers and the records kept by their chief are useful to the geologist, who needs them in his work of estimating ore reserves and deciding on recommendations for development work. The sampler, moreover, is at the mine every day and makes a daily report, which keeps the geologist in close touch with the progress of development work.

The sampler is attached to the mine office and thus comes under the jurisdiction of the operating department. In fact, any special orders prepared by the chief sampler are submitted to the general superintendent, after approval by the geological department. The mine foreman may

* Assistant Manager of Mines, Anaconda Copper Mining Co.
† Assistant Chief Geologist, Anaconda Copper Mining Co.

also order certain special samples taken for him, if those required by the sampling department do not fully meet his requirements. The samplers are selected by the geological department with the approval of the operating department; they are generally young men with some technical training. The coöperation between the operating department and the geological office, which has long existed in Butte, makes this organization of the sampling branch practical and efficient.

The Orebodies

The ore deposits to be sampled are veins varying in width from a few inches to 50 or 60 ft. Steep dips predominate although dips at all angles are found. The veins belong to a complex system of fissuring of different ages and present a wide variety of vein filling. Some exposures present a hard mineralization of fairly uniform character from wall to wall, but many contain bands of altered granite, crushed in the process of faulting and carrying only scattered particles of metallic sulfides. A large part of the orebodies consist of veinlets of chalcocite or enargite traversing granite in such close succession as to make valuable ore of certain masses. Throughout the district, however, the structural features are fairly well defined and have served as a guide to the sampling.

Standardization

The first step under the present organization was to classify all the probable occurrences to be sampled into a few standard types. Four types were found to be enough to cover all exposures in the faces of drifts; they are: Uniform ore; part good and part low grade; part good, part low grade, and part waste; narrow streak good but balance doubtful.

The types are shown on drawings on a fairly large scale, together with some plans and sections of crosscut exposures and type sections of raises. The required sampling is indicated definitely on each sketch and a colored print of each is given to each sampler. This serves as a basis for all the sampling and, with certain written instructions accompanying each print, the results from all the mines are the same and the personal equation is eliminated. Figs. 1 and 2 show these standardization charts.

Underground Methods

The sampler is required to examine each place to be sampled and decide to which type it belongs. He then takes the required number of samples, at breast height, along a horizontal line, irrespective of the dip of the vein. He also measures the width horizontally. This measurement is correct because the longitudinal sections upon which the ore

blocks are laid out are projections on vertical planes. The ideal cut, or channel, across the face of the drift is impractical, from an operating point of view. Too much time would be lost if the miners had to wait
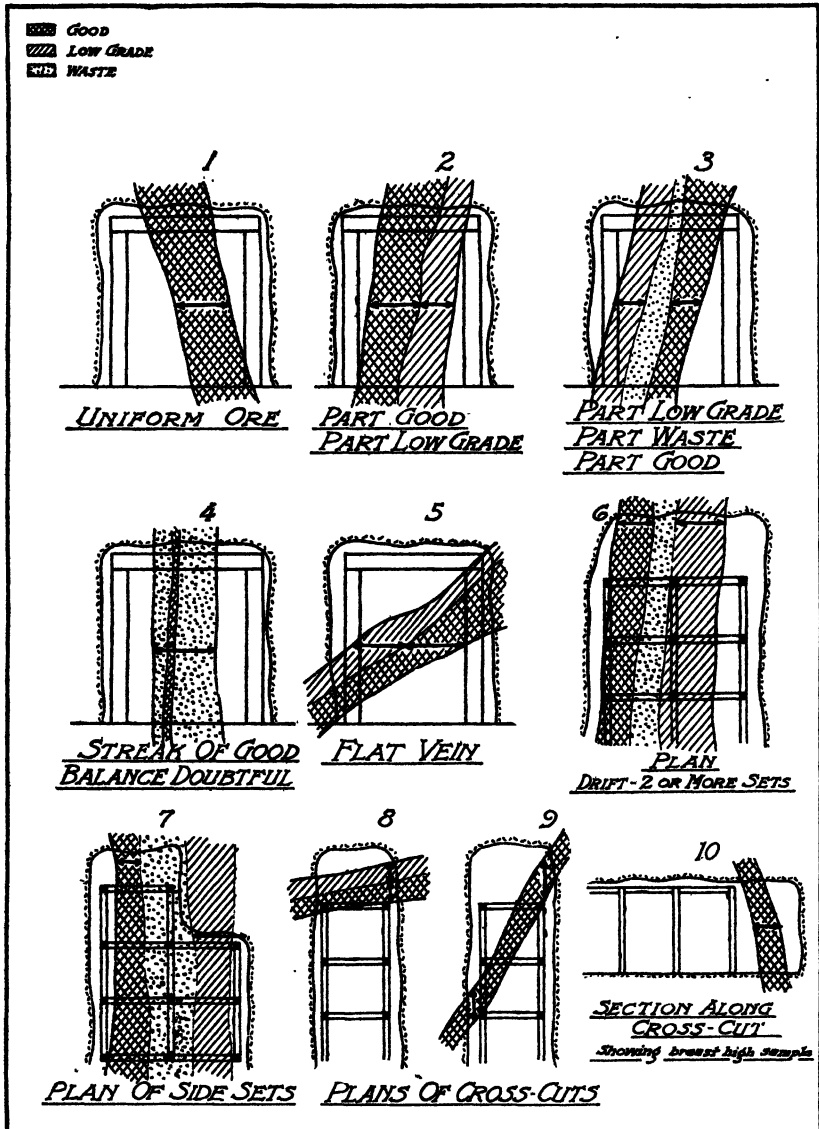
FIG. 1.

while the sampler laid out and chiseled a neat groove across the vein; indeed, the varying degree of hardness of parts of the vein would make such procedure difficult. The same results are obtained by the use

of the sampler's pick. The sampler takes equal amounts of material from points along a predetermined line and catches it in a sack held open by a wire ring. Tests show that the accuracy of such sampling
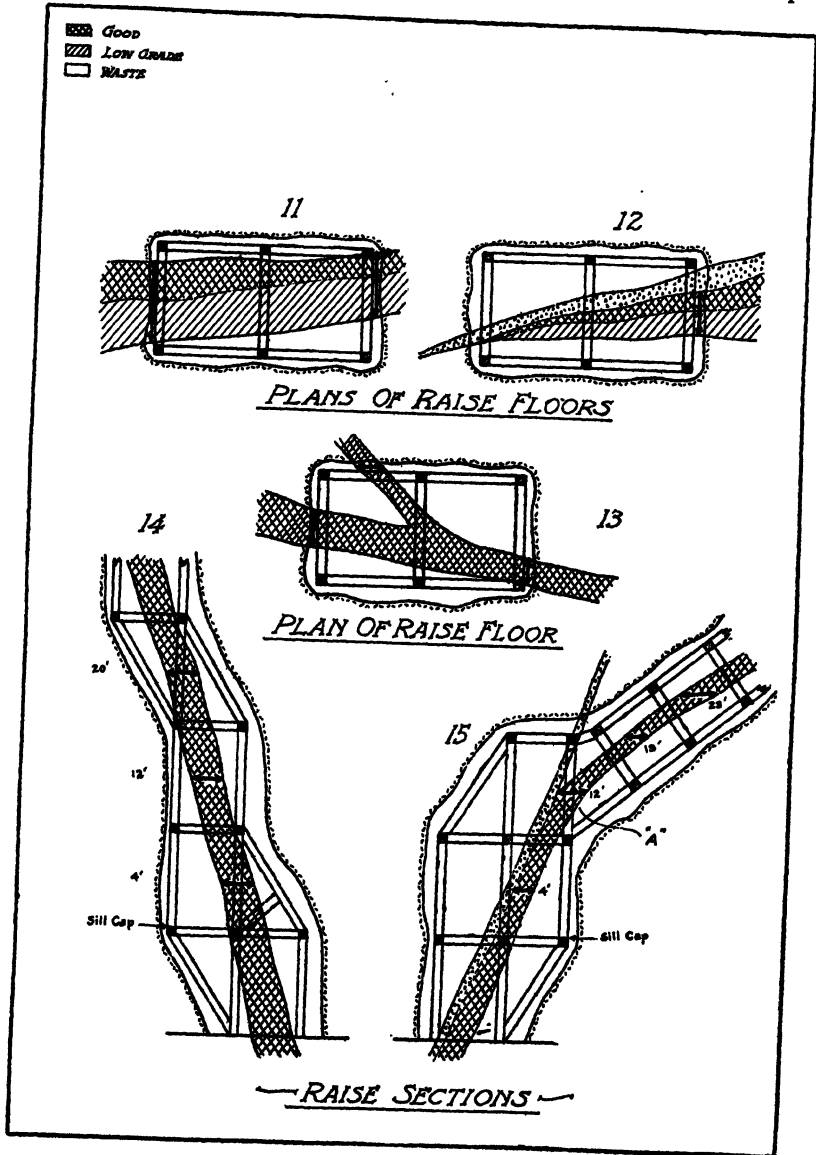
Fig. 2.

is sufficient for operating purposes. For instance, samplers were required to take duplicate samples at each face of four drifts during the progress of the work. One set of samples was called series A and the other series B.

The drifts varied in length from 93 to 199 ft. Two sets of averages were prepared, one from series A and one from series B; the results are as follows:

DRIFT No.	LENGTH, FEET	SERIES	COPPER, PER CENT.	SILVER, OUNCES	ZINC, PER CENT.
1	93	A	10.0	4.1	
		B	9.7	4.1	
2	97	A	2.4	3.8	11.4
		B	2.3	3.9	11.8
3	167	A	5.6	4.3	
		B	5.7	4.6	
4	199	A	5.0	4.0	
		B	5.0	4.3	

Other tests are made from time to time and the chief sampler makes frequent trips underground, accompanying the mine sampler, and takes duplicates of the day's samples. They do this independently, in fact without watching each other. A record is kept of these duplicates and each man's results in his mine are tabulated for comparison with the results obtained by the chief. Assuming that the chief's samples are correctly and accurately taken, the men are rated as high or low and are cautioned in the right direction. In order to be sure of the value of these tests, the assayer's results are checked from time to time by carefully dividing a sample and sending parts to the four assay laboratories.

From time to time a meeting of the samplers is held and questions are asked. Occasionally, at these meetings, demonstrations are made to show how errors may creep in and to assist in the standardization of the methods.

REPORTS

The samples taken are sent each day to the assay laboratory and the results returned to the sampler the next day. He keeps a record of the results in a book prepared for the purpose and makes a daily report to the geological office. This report is a simple form, calling attention to new places started and giving the results of the day's assays. These reports serve to check the calculation of the more formal weekly reports, which contain the average of the samples taken in each place for the week. The daily results are also copied into loose-leaf ledgers so that any averages required may be figured. They are then given to the geologists, who thereby keep in close touch with the progress of the development work. The weekly report ends with the samples taken and work done on Wednesday so that assay results are available; the averages are figured and the reports delivered to the general superintendent and the geological department on Friday. The method of figuring averages has been carefully standardized and the samplers are provided with type forms to show different cases. The weekly report gives the progress for the week, the total

to date, and other relative data besides the average assay value. Fig. 3 shows a typical daily report and Fig. 4 a weekly form.

Form No._____

O   O

SAMPLERS DAILY REPORT

Mine __ORIGINAL__   *Date* __Dec. 3, 1921__

	Number	REMARKS		Dist.	Width	% Cu.	Oz. Ag.	% Zn.
Dr. S. E.	1523			270	4	4.6	6.5	
Dr. N. W.	1876		N	71	3		6.5	3.1
			S	71	4		15.2	8.2
			Ave		7		11.4	6.0
Dr. E.	3020			64	2.5	3.6	1.4	
Raise	2385	East end		190	4.5	2.5	Tr.	
Raise	2385	West end		190	3.0	3.1	0.5	

FIG. 3.—SAMPLER'S DAILY REPORT.

Butte, Montana, __Dec. 15, 1921__

GEOLOGICAL DEPARTMENT

Following are the principal advancements made in different mine workings

during the week ending __Dec 14, 1921__ 19__   BADGER MINE _____Mine

LOCATION	FROM	REMARKS	Advance	Total	Width	% Cu.	Oz. Ag.	% Zn.
1337 D.W.	1308 D.W.	South Vein	18	717	5.2	3.0	3.5	0.5
1669 D.E.	1665 XCS	Split of Main Vein	17	133	2.6	4.7	5.6	Tr.
2055 XCN	2014 D.E.	Starts 263 E of 2012 XCN	25	25			Waste	
2232 D.W.	2202 XCN	North Vein	13	431	3.8	1.0	2.9	Tr.
2443 S.S.	2443 D.E.	North Vein Form 100' to 105' E. & 5' N			1.0	2.1	1.9	
		RAISES						
1347 Rs	1337 D.W.	South Vein	16	102	4.2	2.3	4.0	3.4
1841 Rs	1827 D.W.	Edith May Vein	8	125	5.0	1.5	4.3	14.2

FIG. 4.—WEEKLY SAMPLE REPORT.

OFFICE RECORDS

The principal office record consists of a set of cards cut from standard cross-section paper, ruled to tenths of an inch. Each drift and raise has a card and is identified by its number in the upper right-hand corner; the name of the mine is in the left-hand corner. The cards are kept in small boxes; they are arranged in alphabetical order for the mines and in numerical order for the cards in each mine. The horizontal scale on the cards is 1 in. = 50 ft., which is the same as the standard geologic map.

The vertical scale is 1 in. = 10 per cent. for copper, or 10 oz. for silver. A definite starting point is marked and a curve is plotted for each metal, using the weekly advance report to get the positions on the horizontal scale and the assay averages on the vertical. A black curve is used for copper and a red one for silver. The width of the sample is written under the base line in its proper place. These curves are useful in many ways. By placing one under the standard geologic map of the corresponding drift, it becomes an assay map. The curve shows the location of the better ore along the drift, it shows the proportion of silver to copper, and allows a fair estimate of the average assay value at a glance. If a curve descends precipitously to the zero line, the geologist visits the place to see if the vein has been faulted or if some change in the development work can be made. Curves for raises are plotted from vertical base
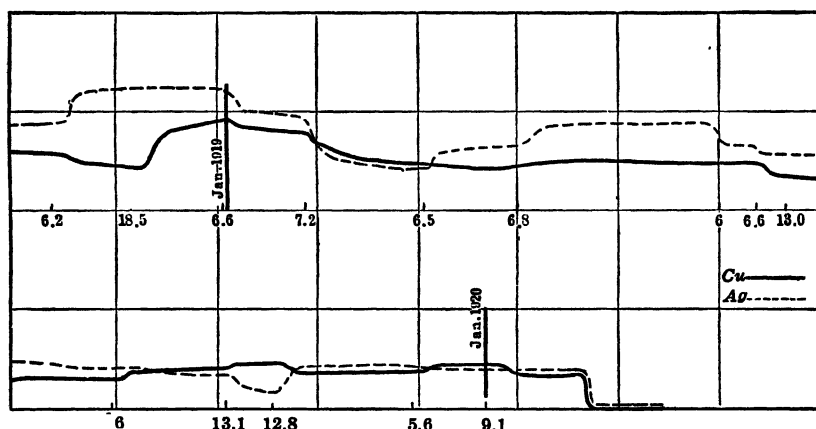
FIG. 5.—OFFICE SAMPLE RECORD.

lines and are equally useful. Fig. 5 shows a typical assay curve card For special averages or other information and for estimates connected with ore reserves, the ledgers containing copies of the daily reports are used but the curves are satisfactory for most purposes.

For the estimating of ore reserves, there is a longitudinal section of each vein made to a scale of 1 in. = 100 ft. and projected on a vertical plane parallel to the strike of the vein. A certain minimum width and assay value is assigned by the operating department as the lower limit for ore. Then the drifts and raises on the long section are colored red wherever the assay curve shows the required grade of copper ore and green for zinc. Waste drifts and raises are colored gray. This color scheme shows quite plainly what should be included in the ore reserve for each section. Blocks are, therefore, laid out upon the section and their volume and tonnage calculated in the usual way. For the grade of these blocks properly weighted, averages are calculated either from the curve records or from the more complete daily reports in the ledgers.

DISCUSSION

GEORGE A. PACKARD, Boston, Mass.—This paper gives valuable suggestions in determining the individual coefficients of men taking samples. If we could have on file the personal coefficients of every sampler in the country, as compared with accurate samples taken under different conditions, it would be most advantageous. It is remarkable that practically all samplers get too high results.

It has been stated here that a factor of safety is not necessary where the whole orebody is mined and there is no waste capping to be considered, because there is no dilution in mining; but in cases where dilution is possible, it is a question how largely the decrease in value of ore mined, as compared with sample averages, is a matter of dilution and how largely the factor of error in the individual sampling. If it is a factor of error, why is it always too high?

A few months ago the manager of a mine in California, which was mining gold ore that varied from $4 to $50 per ton, said that the mill results were between 80 and 85 per cent. of the average mine samples. Those mine samples were not always channel samples; some were car samples, a shovelful being taken from the top of each car, which, of course, is very inadequate sampling. At a copper mine where the ore occurs in beds, varying from 5 to 40 ft. in thickness, the ore from which is shipped to a smelter and according to the smelter analysis runs from 2.5 to 2.7 per cent. copper, there is a decrease of about 8 per cent. in the total value below the mine assays. That is not always the result of dilution. These are merely examples, but apparently always have the discrepancy in the same direction. Must we not continue to use a factor of safety in practically all sampling operations unless we can get the personal coefficient of all samplers employed?

W. B. DALY.—The average foreman has pride in his mine and is optimistic about its development, so there is a tendency on his part to exaggerate the value of a drift that he is driving. But it does not make any difference to the sampler whether a drift is or is not ore and as he is not going to benefit one way or the other by it, he has no grounds for undue optimism. For this reason, if the sampler operates independently of the foreman, we feel that our method comes closer to the actual value of the face than would be possible in any other way.

B. B. GOTTSBERGER, New York.—In taking the ore from the different parts of the mine, is the ore actually sent out sampled and is an effort made to get ore from different parts of the mine to check against a mill sample?

W. B. DALY.—That is done where development work is being carried on; that is in drifting, not stoping.

B. B. GOTTSBERGER.—I meant particularly ore coming out of stopes. Do you try to keep the exact grade of ore in each stope and combine that with the tonnage?

W. B. DALY.—We keep the stope and drift samples separate. Of course the ore varies from drift to stope and from one floor to the other. It varies materially from floor to floor but the average has usually worked out pretty nearly with the drift.

If the samplers and assayers are put under general jurisdiction and not under the mill or smelter foreman, the discrepancy will not occur. We found that where the sampler at the mine was under the jurisdiction of the foreman, that he had better samples than when he was placed under general jurisdiction.

Handling Ore in Mines of Butte District

BY H. R. TUNNELL, BUTTE, MONT.

EVERY one connected with a mine knows that it is hard to keep down the costs of moving ore from the place where it is broken to the shaft or portal. Considered broadly, the subject of handling would cover all work done in a mine, but here it has been limited to handling ore in stopes and drifts, through chutes, and on levels. Good ventilation and a strict enforcement of rules for the prevention of dust safeguard the health of the men and lower transportation costs.

HANDLING ORE IN STOPES AND DRIFTS

The object of all stoping is to mine ore of good grade at low cost. In Butte, most of the ore is mined, from veins dipping from 60° to the horizontal to those which are vertical, by means of square-set, rill, timbered-rill, and back-filling stopes. When blasted, a part of the rock is carried by gravity to the grizzlies over the chutes, the rest is shoveled.

Rock, when broken in a stope, may be deflected by inclined floors or slides toward the top of a chute leading to the level below. Ore broken clean enough to ship should be carried, by the force of its fall, into a chute or as nearly into that chute as the safety of the stope and the men permits. Ore that requires sorting, about 80 per cent. of all ore mined, is caught on a grizzly and the waste picked out. Waste rock broken in a stope is used to fill that stope and should be blasted into the gob with as little handling as possible.

About two-thirds of all the ore mined in Butte comes from square-set stopes; this method was used in nearly all cases until 1912, when the rill method of stoping was introduced in the Tramway mine, of the Anaconda Copper Mining Co. The shoveling and timbering costs in square-set stopes are high when compared with the costs in the rill stopes. All the ore cannot be carried to the chute, in falling from place, unless the chutes are very close together or the distance from the back of the stope to the top of the chute is sufficient to slide all the ore broken. The number of chutes is limited by the tendency of the ground to cave; if carried too close together the stope is weakened. Small stopes, or blocks,

as they are called, are used in working a large orebody, as they stand better and allow more of the ore to be handled on slides.

The rill stope is growing in favor throughout the district. These stopes are filled with waste from a chute at the upper end and floors are laid on this waste. The ore is drilled and blasted to the floor, which is from 40 to 45° to the horizontal, and runs to the grizzly at the lower end of the stope at a minimum cost for handling.

The timbered-rill is a rill stope with square-set timbering; it is used where the vein is too wide and the back is considered unsafe for the open-rill method. These stopes are filled and the ore is removed as in the open rills. The broken ore is drawn into the chute as cheaply, save for some interference by the stope timbers to its passage down the floors.[1]

The back-filling stope is worked without timber between chutes. The floors are level, as in the square-set stopes, yet rest on the waste filling as in the rills. Except where the veins are very flat, under 50°, the back-filling stope has been supplanted by the open rill.

Shoveling

Laying good floors and slides and careful sorting on the grizzlies are more important in a stope than using a shovel to put rock into the chute. Actual shoveling costs vary so much for the different mines, as well as for the individual stopes in any one mine, that an accurate statement of shoveling will not be given. The results given here are from the cost sheets for some of the larger mines over a 4-yr. period.

COMPARATIVE SHOVELING COSTS FOR DIFFERENT METHODS OF STOPING

Square-set stoping	100
Rill stoping	75
Timbered-rill stoping	85
Back-filling stoping	120

These results show that: The rill stope is the most economical; the timbered-rill stope makes the application of the rill principle more general. The back-filling stope is used to meet special conditions; the square-set stope is still used in heavy ground and where large orebodies are to be worked in blocks.

All the ore broken in drifts is shoveled into cars. The amount shoveled per man depends on car service, distance to be trammed by the shovelers, condition of cars and track, and ventilation, as well as the ability of the men. Slight delays in getting cars sometimes cause considerable losses in efficiency; for instance, two men who have averaged over fifty cars per shift (35 tons) shoveled, trammed, and dumped into a

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[1] H. L. Bicknell: Rill and Timbered-rill Stoping in Butte. *Compressed Air Magazine* (1921) 26.

chute may shovel but thirty cars when moved to another drift, because of bad tracks, stiff cars, poorly trained men on trains, etc. The well-trained shoveler of exceptional physique can overcome unfavorable working conditions, but good car service, ventilation, and equipment must be maintained if the average man is to do a satisfactory shift's work.

If a drift or crosscut is timbered to the breast, because of heavy ground, floors are laid on sills or spreaders; if untimbered, turn sheets of ¼-in. steel plate are placed against the breast to receive the broken ore. About 70 per cent. of the ore broken in drifts is shoveled from floors or turn sheets, the remainder from the rough bottom. From 12 to 18 tons per man-shift are shoveled in drifts and crosscuts.

A No. 2 shovel with a straight handle 42 in. long is used throughout the district; short shovels with a D-shaped handle were tried but were not liked by the miners. The tools used in shoveling are: No. 2 square-point or round-point shovel; No. 4 picks; 12-lb. rock hammers; No. 4 steel wheelbarrows; Anaconda-type mine cars. The grizzlies are timber or steel rails spaced 4 to 10 inches.

HANDLING ORE IN CHUTES

When an orebody is encountered, in developing a new level, square sets are used to timber the level along the vein, or the sill floor, as it is called. Enough work is done on the sill to determine the quality of the ore and the method of stoping that should be adopted. The location of the raises is then decided and one is started as soon as possible to improve ventilation.

If square-set stopes are to be used, a raise with one chute may be selected as it will stand better in heavy ground. As few veins are vertical, the raise timbers follow the foot wall and offset from time to time. This means that a floor standing vertically above the one below will be timbered with six posts; the offset floors, with nine or twelve posts. When the chutes are built, these offsets, by breaking the fall of the ore, reduce the wear on the chute lining. If a chute is vertical, false offsets or beats, Fig. 1, may be used. The ore mined in driving a raise of this type must be sorted at the chute mouth on the level.
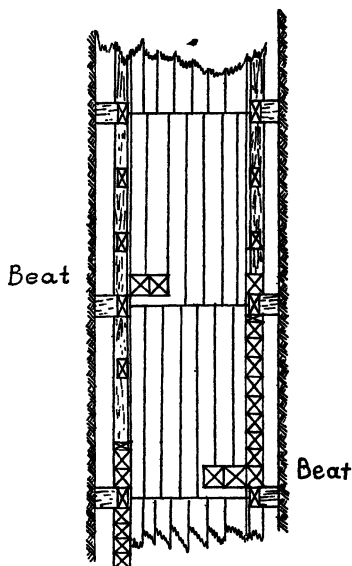
FIG. 1.

Sometimes a larger raise is used; two chutes may be carried and sorting landings built in the chute sets. Fig. 2 shows a raise of this type. Each chute is used alternately for ore and waste picked off the grizzly is thrown into the other chute. These large raises are usually driven in connection with rill stoping: their larger cross section insures better ventilation, the two chutes permit closer sorting of the comparatively large tonnage mined while the raise is being driven, and, later, waste is furnished the rills without building slides across the raise manway.

In laying out rill stopes, raises are driven at regular intervals along the vein as each raise will, when holed, furnish waste to two rills. If back-

Sorting
Chute

Grizzly
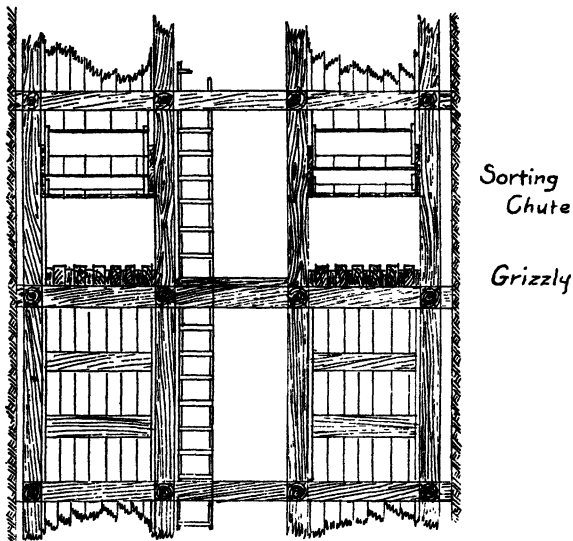
Fig. 2.

filling stopes are to be used, the raises are timbered with square sets but stope chutes are sometimes built of log-cabin cribbing. Stoping starts and stope chutes are built after the raises are holed to the level above. The lower floors of all chutes are solidly built and the square sets of both chute and manway are well braced. Transfer chutes are often driven without timber, in solid ground, away from the vein.

Stope chutes and transfer chutes, once a great source of dust, are kept wet. Water is piped to nearly every stope and as the dry stoping machines will soon be replaced by stopers using water, all stopes in the district will soon have water connections. After blasting, the rock pile is sprinkled, and if dust still forms, water is turned into the chute.

Fig. 3 shows three floors forming an offset in a chute. The timbers used are 8 to 10-in. round posts and caps, and 6 to 8-in. girts and

chute centers. The lowest floor is lined on three sides with 2-in. lagging; on the fourth side, with 10 by 10-in. bulkhead pieces laid skin to skin. The middle floor bulkhead is one-half the height of the posts. The top floor is lined with 2-in. lagging on all four sides.

Fig 4. shows three ways of lining a chute; the lowest floor is bricked with round and square blocks 20 in. long. The middle floor has an upright bulkhead (10 by 10-in.) instead of the usual flat bulkhead of Fig. 3. The top floor is bricked on four sides with 10 by 10-in. bulkhead pieces, with 2 by 6-in. spacing blocks at both ends.
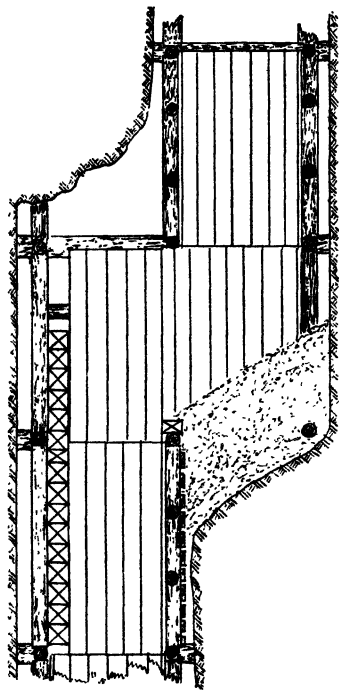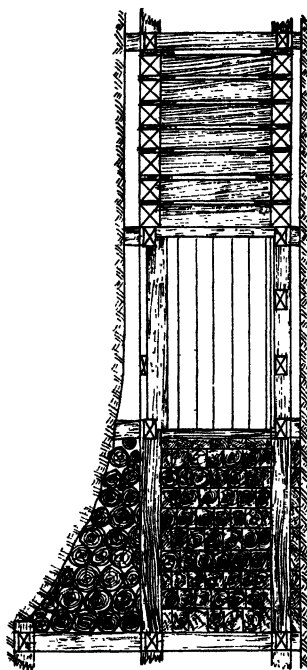
FIG. 3.   FIG. 4.

HANDLING ORE ON LEVELS

Underground haulage is divided into three classes; hand tramming, animal haulage and haulage by mechanical means. The amount of ore to be handled, length of haul, size of haulageway that can be readily maintained, and the grades to be overcome, determine which method is best adapted to any level.

Cars of the same type but of two sizes are used: The Anaconda No. 1, used in two mines, is 20-in. gage, holds 16 cu. ft., stands 3 ft. 10 in. above the rail, and is 2 ft. 2 in. wide. The Anaconda No. 2, used in twenty-eight mines, is 18-in. gage, holds 14 cu. ft., stands 3 ft. 10 in. above the rail, and is 2 ft. 0 in. wide. These cars are small for motor haulage but,

because of the heavy ground, caused by settlement of stopes and sills in the wide orebodies, it has not been considered advisable to change to a larger car. Of the 7000 cars in use 3000 have roller-bearing wheels, which will be the standard equipment. The doors are in front but the cars may be swung and dumped to either side; when dumping at the side, a device prevents the car tipping back and injuring the trammer. The dumping lever stands in a vertical position at the back of the car. When it is moved through 15° the car is free to swing, but the door is held by its latch until the lever has moved its full travel. Cars recently introduced at two of the mines have foot levers operating very ingenious dump locks and door latches instead of the usual hand levers.

For hand tramming, 12 to 16-lb. rails are used; for animal haulage, 12, 16, and 25-lb. rails; and for locomotive haulage, 25-lb. rails. The switches
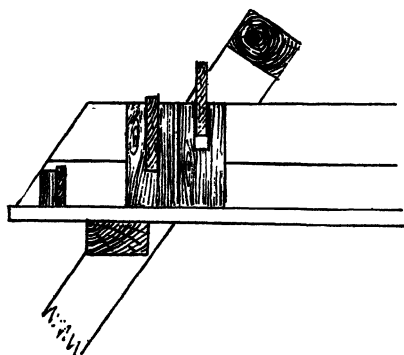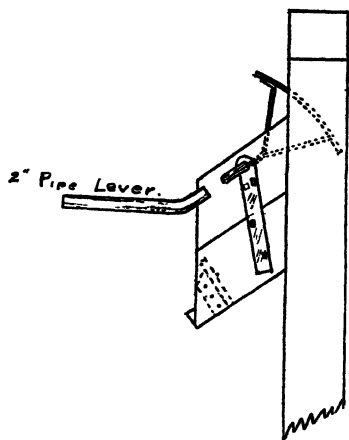
FIG. 5.   FIG. 6.

are forged from standard rail and in nearly all cases the frogs are riveted to ¼-in. plate. Long tongue and latch switches are used. The latch switches are made single and double throw, switch stands and spring latches of varying design are used. The track is laid on ties or runners. The ties are 3 by 10 by 30 in., 4 by 10 by 30 in and 5 by 7 by 48 in.; and are spaced from 20 to 42 in.; the runners are 4 by 10 by 64 in. In most of the mines, ditches are carried under the center of the track; in the others, to one side of the track. The average grade on new levels is about 0.50 per cent. in favor of loads. Heavy and adverse grades in some mines are the result of careless work done in opening levels and of settlement of the ground near large orebodies. In some of the older mines, while shafts and stations show little settlement, the upper levels have settled several feet over old stopes. With extended workings, the cost of regrading a level is often prohibitive.

There are wide variations in the size and design of chute mouths at the different mines; the sizes run from 22 in. wide by 24 in. high to 36 in. wide by 20 in. high. Some have iron doors, as in Fig. 6, with levers; some have iron stop boards, with or without levers, but most of them have simple wooden stop boards. The chute bottoms are inclined from 35 to 45° to the horizontal, and are protected from wear by ¼-in. steel plate. The height of the lip varies from 4 to 18 in. above the side of the car. Wooden stop boards are used in nearly all cases where the trammer loads his own car.

Hand and Animal Tramming

Ore is trammed by hand when opening new levels, on intermediate levels, and where the total amount to be handled is small or the distance not considerable. Less ore is transferred from chute to chute than in the past; nearly all ore is drawn directly from stope chutes by trains hauled by animals or locomotives.
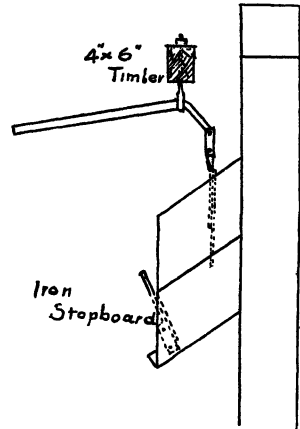
FIG. 7.

Horses and mules haul about 15 per cent. of the ore. A horse lives nearly as long in a mine as on the surface, many working there from 10 to 15 years; regular feeding, careful attention by stable bosses and veterinarian, and occasional periods of rest on the surface contribute to this result. A mine horse should be intelligent and kind; if it is not the time spent trying to train it will be wasted. The animal is usually driven single, but for long hauls two or more horses or mules driven tandem have been used. Ten cars form the standard trip for one animal, but because of heavy grades shorter trains are used in some places. A good mine horse is worth from $100 to $150; harness, $35; collar, $8. A horse should be shod once a month if it is worked every day. The cost of a horse per day is as follows:

Shoeing, $3 per month...	$0.10
Stable boss, caring for 12 horses.................................	0.50
Feed...	0.50
	$1.10

Care must be exercised in locating underground barns. The ideal place is just off a main air course, where the animals will be in good air but not in a draft and where the stable smells and vitiated air will go to the upcast shaft, without passing through places where men are at work. Provision must be made for drainage. The ground should be good, as

a considerable opening must be made to accommodate two or more horses. Stalls should not be less than 4 ft. 6 in. wide by 16 ft. from the track. A stall should be provided for each horse needed, and space for oat box, hay, and water tank. Barns have water piped in when possible and are white-washed at regular intervals to keep them light and sanitary.

Mechanical Haulage

Electric trolley locomotives have been used in the Butte district for the past 14 years. All tracks and switches are bonded with No. 4 soft-copper wire. The minimum radius on turns is 10 ft.; and on switches, 15 ft. Haulageways are timbered not less than 4 ft. 2 in. by 7 ft. 8 in. and the trolley box is carried at least 6 ft. 8 in. above the rail. No. 0 hard-drawn trolley wire is carried by hangers spaced not over 20 ft. apart in a box consisting of a back board (2 by 6 in.) nailed to drift caps or sprags with 1 by 10 in. side boards. Direct current of 250 volts is supplied to each mine, or group of mines, by motor-generator sets located on the surface. Protection is given the men loading at the chutes by cut-out switches, or by insulating the wire at the chute mouths. Lights are placed every 100 ft. along haulageways and over switches. Signal systems employing colored lamps are used where more than one locomotive is worked on a level.

Seventeen mines use trolley locomotives—17 Jeffreys, 24 General Electric, and 107 Westinghouse-Baldwins. These locomotives have the following dimensions:

BALDWIN No. 901 OR WESTINGHOUSE-BALDWIN		JEFFREY	
Weight	3½ tons	Weight	3½ tons
Gage	18 in.	Gage	18 in.
Wheel base	3 ft.	Wheel base	3 ft.
Length	11 ft.	Length	11 ft.
Speed	7 mi. per hr.	Speed	6 mi. per hr.
Drawbar pull	1200 lb.	Drawbar pull	1600 lb.
Under normal load motors develop	22 hp.	Under normal load motors develop	12 hp.

Storage-battery Locomotives

Eight years ago storage-battery locomotives were tried by the Butte & Superior Co., as the condition of the haulageways, because of very wide orebodies and heavy ground, made haulage by the trolley-type impracticable. These locomotives will now be used in all mines; they do away with the overhead trolley wire, which is always a source of danger to the men, and is expensive to install and maintain. In some cases, it is impossible to maintain a drift of sufficient height and width to make

haulage by the trolley type of locomotive safe. To put trolley locomotives on an old level, it is often necessary to retimber many of the old drifts before the trolley box can be built at a safe height. If 25-lb. rails are already laid, they must be bonded. The storage batteries require close attention by an electrician but, aside from this and the cost of replacing batteries, this locomotive is as easily maintained as the older type.

The cost per car of ore hauled by a Jeffrey storage-battery locomotive in the Black Rock mine, Butte & Superior Co.,[2] is as follows:

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Power per car hauled....................................  $0.0041
Repairs to locomotive, per car.........................   0.03205
Motorman and helper, per car...........................   0.06900
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                                                         $0.10515
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The measurements of the locomotive are as follows:

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Weight with battery....................................  3½ tons
Gage...................................................  18 in.
Wheel base.............................................  36 in.
Length.................................................  11 ft.
Maximum width..........................................  32 in.
Drawbar pull...........................................  1500 lb.
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During the year 1919,

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Number of locomotives operating........................       9
Cars of ore hauled.....................................  605,372
Cars of waste hauled...................................  132,415
Average haul of a car of ore...........................   800 ft.
Average haul of a car of waste.........................   747 ft.
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The Anaconda Copper Mining Co. is replacing its trolley locomotives with storage-battery locomotives of the following measurements:

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Weight.................................................  3½ tons
Gage...................................................  18 in.
Wheel base.............................................  36 in.
Length.................................................  11 ft.
Speed..................................................  3½ mi. per hr.
Drawbar pull...........................................  1500 lb.
Under normal load motor develops.......................  26 hp.
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[2] These figures are copied from a paper presented at the Convention of The National Safety Council: "Storage Battery Locomotives in Metal Mines," by E. V. Daveler, General Superintendent, and R. E. Renz, Chief Electrician, of the Butte & Superior Mining Company.

No other mechanical system of haulage is in use in the Butte district. Compressed-air locomotives were tried but their use was discontinued because of the danger and difficulty of maintaining high-pressure air lines underground.

STATIONS

The manner of handling ore trains on the stations is dependent on whether skips or cages are used to hoist the ore. If skips are used, the skip pockets are often large enough to store from 50 to 150 tons, although in some mines pockets holding one skipful are used and in others, on levels producing only a small tonnage, the cars are dumped directly into the skips. The cars from the trains are handled by station tenders (men working on cages) when the ore is hoisted by cage, dumped into the skip direct, and when pockets holding but one skip are used. With large skip pockets, the men on the train dump cars in the pockets; or if the tonnage is large enough, a man is kept on the station to dump the cars. The cars are dumped through holes covered by a grizzly with openings not less than 10 in.; in some cases as small an opening as 4 in. is used to sort out waste.

Storage-battery Locomotive as Applied to Mine Haulage

By Charles E. Stuart, E. E., New York, N. Y.

A paper on this subject can cover but a limited range. A thorough visualization of the subject would contemplate a comparative analysis of haulage machines and batteries of various types; the relationship of the storage-battery locomotive to other types of mine haulage should be considered and limiting factors in each case should be carefully developed. This paper will deal briefly with these factors.

So far as the writer has been able to determine, the first storage-battery locomotives for mine haulage were manufactured in 1897 or 1898 and were tried out in the Pocahontas coal field, but they were not sufficiently satisfactory to justify their continued use. Battery explosions were frequent and caused serious damage to the wiring, electrical equipment, and the clothing and bodies of the motormen. Sulfuric acid was used as the electrolyte.

Experiments were then conducted with locomotives bearing reels of flexible cable through which connection between the motors of the locomotive and the trolley wire was maintained. At first, these reels were turned by hand. Later, the reels were revolved by contact with the wheels of the locomotive, being raised out of contact with the wheels, by a lever, when necessary. About 1902, the plan of operating the reel, by chain and sprocket wheels, from one of the locomotive axles, as is still the practice, was devised. The popular floating motor-driven reel had its inception in the Pocahontas fields prior to 1906.

The maximum radius of operation of the reel-type locomotive is about 500 ft. (150 m.) from point of connection with trolley wire. This is ample for room work and certain entry work, but the need for a machine that would operate entirely independently of the trolley wire was so great that, with the development of the storage battery, renewed efforts were made to manufacture a successful storage-battery locomotive. About 1908, the gasoline locomotive was introduced. In spite of many objections to this type of machine for such work, the fact that hundreds of these machines were purchased was proof of the need for a machine operating independently of the trolley.

About 1913, several large manufacturers sought to produce a reliable storage-battery locomotive. For two or three years, although a satisfactory battery and thoroughly substantial mechanical parts and motors were available, the development was slow, because of the failure of the manufacturers thoroughly to understand mine requirements, including the abuse and rough handling that mine machinery must undergo and the difficulty of obtaining proper attention for such machinery. The locomotive was insufficiently motored and that manufacturer frequently got the business whose conscience or lack of knowledge permitted the introduction of the smallest battery or the smallest motor that would make a fair appearance in the quantitative sense.

The interest of the mine owner in haulage is concerned solely in the number of tons of coal or ore that he can safely haul in a day and the cost per ton of this work. As the selling price of coal is relatively small, the haulage system must be brought to the maximum efficiency. As late as 1916, a ton of coal was cut, loaded, hauled over a mile to the tipple, prepared as to size, dumped into the railroad car, and sold at a profit on cars at an average price of $1 per ton. As transportation is but a small part of the cost of a ton of coal, the haulage cost was very small.

In the coal mines having the lowest haulage costs there will be found: Entry trackage of not less than 40 lb. weight well supported and well maintained on ties; room rails of 20 lb. Cars are of the greatest cubical content permitted by the height of the seam (in low seams) or (in high seams) by ease of loading; in well-managed soft-coal mines, where the shovel is used, cars with a capacity of 235 cu. ft. are found. The track gage is not less than 36 in. and usually is over 40 in. The trips contain from 25 to 50 cars and the main haulage locomotives are from 10 to 30 tons. There is double track on the main headings, good voltage at the working centers, the gathering locomotives are from 4 to 6 tons, and the workings are concentrated.

ADVANTAGES OF THE STORAGE-BATTERY LOCOMOTIVE

Within the car limits indicated, with a concentration of work, general grades not exceeding 6 per cent., collecting the cars from a number of places or rooms and taking them to a siding where the trip is made up for the heavy locomotives is the work for the storage-battery locomotive. Under these conditions, the storage-battery locomotive containing motors and batteries of ample capacity for the actual cycle of duty has the following advantages:

1. It is unnecessary to string trolley wire and bond tracks away from the main haulageways.

2. In thin coal, it is unnecessary to brush; that is, to take up bottom in order to make height necessary for mules.

3. No time is lost by making cable connections with the trolley on entering rooms, nor by cutting the reel cable and splicing it where cut.

4. The chance of personal contact with the trolley wire is absent.

5. In a gassy mine, away from the well-ventilated headings, the absence of a spark at the wheel or the trolley may be a vital consideration. In those mines where safety lamps are required, it is only a matter of time before the use of the trolley and reel locomotive for gathering will be prohibited. The storage-battery locomotive can be, and is being, made gas-proof and will pass the tests of the U. S. Bureau of Mines.

6. The speed and haulage capacity of the machine, measured in pounds drawbar pull, will enable it to displace a number of mules and drivers and should materially lower the cost of gathering.

7. An invaluable consideration is the independence of the storage-battery locomotive with respect to either substation or power plant during idle days. At such times a storage-battery locomotive may be used for handling supplies or for other purposes.

EFFECT OF STORAGE-BATTERY LOCOMOTIVES ON POWER DEMAND OF MINES

A feature that has been given slight consideration but which is one of the most valuable characteristics of the storage-battery locomotive
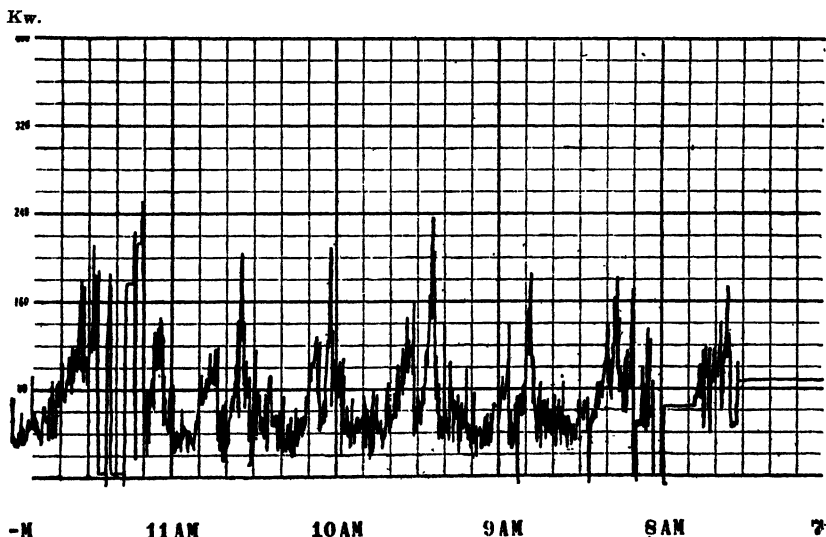
FIG. 1.—PERFORMANCE RECORD TAKEN ON GRAPHIC CHART.

is its effect on the power demand, as shown by Figs. 1 to 4. In Fig. 1 is shown a mine-load curve that is characteristic of most soft-coal mines, except those with heavy pumping and ventilation requirements. Fig. 2 is a characteristic curve of the demand of trolley locomotives. The

starting amperage of a heavy locomotive may exceed the heaviest amperage pulled in hauling. Fig. 3 shows that the relationship of kilowatts consumed to possible output at normal rating, in the case illustrated, is 19.35 per cent. It will be seen that the capacity of a
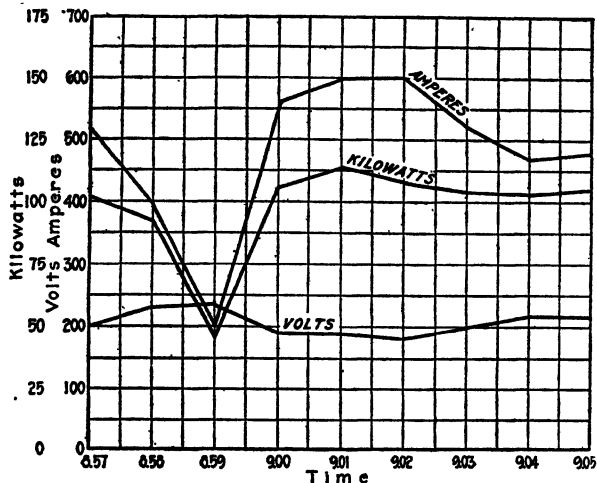
Fig. 2. —Characteristic curve of demand of trolley locomotives.

substation or power plant must be related to demands of short duration, which are invariably trolley or reel-type locomotive demands. By contrast, the demand while charging a 6-ton locomotive containing the heaviest permissible battery will not exceed approximately 15 kilowatts.
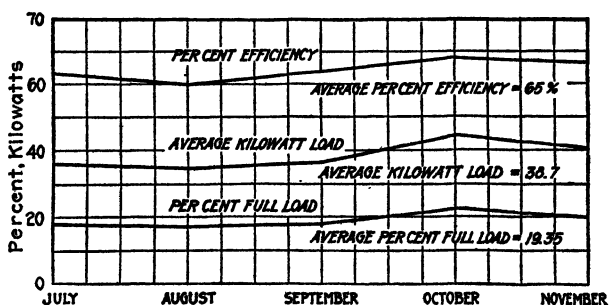
Fig. 3.—Ratio of average consumption to full-load capacity.

Storage-battery locomotives may be charged outside of the day shift, when the demand on the plant or substation is light. As a result, the demand of this locomotive need not conflict with the demand of the trolley-type locomotive or with other demands where the plant is well loaded or where the power contract penalizes demand.

A large portion of all power required for coal and metal mines is being purchased and converted from alternating current to direct current through substations. Figs. 3 and 4 show the average efficiency of a mine substation during 5 months. In this instance, 35 per cent. of power was lost in the process of conversion; this figure is above the average. Although this loss in conversion for the actual battery charge may not be lowered where a small motor-generator set is used to convert alternating current to direct current for battery charging, as soon as a battery is charged, the charging set automatically shuts down; also the heavy loss entailed for operating a substation for charging purposes is eliminated.

Storage-battery locomotives are especially suited to new or small mines, from the standpoint of both investment and operating costs. The writer is now working on plans for a mine in which alternating
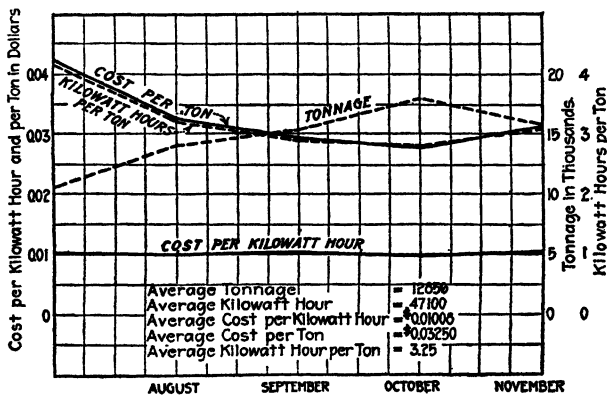
FIG. 4.—OUTPUT, CONSUMPTION, COST PER TON AND PER KILOWATT-HOUR OVER FIVE MONTHS.

current will be used for all stationary motor drives and for mining machines. For a short time, animal haulage will be the cheapest; but when the economic limit for this haulage is reached, it is proposed to purchase a charging set and storage-battery locomotives. In many cases, with a small mine, the storage-battery locomotive will be the most economical haulage unit for the life of the mine. With larger development, the third step would be the introduction of the heavy trolley type of locomotive and the diversion of the storage-battery locomotive to gathering.

The actual overall efficiency of a storage-battery locomotive is lower than the overall efficiency of the reel-type locomotive. However, the voltage loss between substations or power plant and the trolley type of locomotive will almost invariably discount this advantage. {The writer is inclined to doubt the advantage of either the trolley-type or the storage-battery locomotive over the other in the matter of efficiency.

FIELD OF DIFFERENT LOCOMOTIVES

From the foregoing, it may be assumed that the writer favors the storage-battery locomotive over the trolley or the trolley and reel type, within weight limitations. As a matter of fact, the writer considers that each type has its own fairly well-defined field. To define the limits of the fields would be exceedingly difficult; they must be defined for each mine.
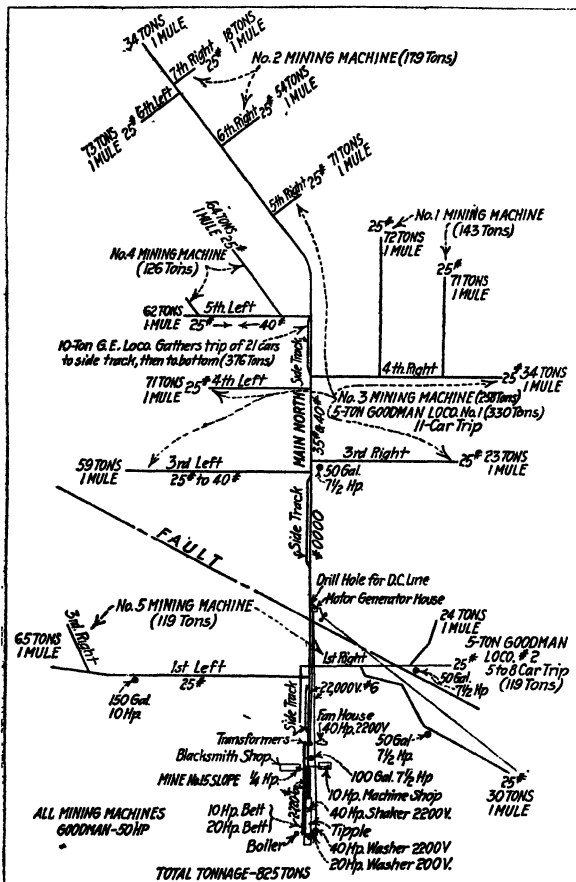
FIG. 5.—ELECTRICAL EQUIPMENT, MINE No. 15.

For instance, to define the trolley locomotive as a haulage machine and the storage-battery locomotive as a gathering machine, as the term gathering is used, a classification that is often made, is unfair to both machines.

The standard trolley type of locomotive is equipped with motors of a capacity equivalent to 10 hp. to the ton of weight; that is, the 6-ton locomotive will be equipped with motors of 60-hp. capacity. The speed of this locomotive at rated drawbar pull, will approximate 6 mi. per hr.

A storage-battery locomotive of the same weight will be equipped with a motor capacity approximating 4 hp. (maximum) to the ton and having a speed, at rated drawbar pull, of 3½ mi. per hr.

If, from other standpoints, a given condition favors the storage-battery locomotive and if the cycle of duty is within the range of that locomotive and its battery capacity, there is little object in using a locomotive with a greater capacity. However, it is impossible to apply a general rule and the writer is simply calling attention to salient points.

NOTE: Arrows indicate direction of Pillar Drawing in each Section

Barrier Pillar

Barrier Pillar

Barrier Pillar

Barrier Pillar

Rooms to be worked and Pillars pulled

Room and Pillar System to be continued in this Section to maintain output while developing Panel Territory

FAULT DOWN

Main Slope

Approximate Boundary Workable Coal

SLOPE MINE No. 15

0' 500' 1000' 1500'

FIG. 6.—PROPOSED PANEL SYSTEM, MINE No. 15, GALLOWAY COAL CO.

A plan of a mine worked on the room-and-pillar system is given in Fig. 5. Mules are used for gathering and the work per mule per day in the different sections is given. Were a locomotive to be applied to the first right and first left entries, it would be necessary to cover 21 mi. per day in order to handle 119 tons and to displace three mules. The same analysis applied to the entire mine will show that the number of

mules displaced by locomotives would be insufficient to justify an investment in locomotives; in fact, the locomotives would be operated at a loss as compared with mule gathering.

A proposed panel system applied to this mine is shown in Fig. 6. We have projected this into the unworked area, as a continuation from the present limits of the room-and-pillar system. This panel system produces a condition of maximum concentration, which is the condition most favorable to the use of storage-battery locomotives.

The Queens Coal and Mining Co., of Indiana, gives the following comparative results from using storage-battery locomotives with the panel system:

"We are operating No. 4 mine on the panel system and aim to keep the trolley partings within 800 ft. of the room necks, which means that the battery locomotives have 800 ft. of entry haulage each way or 1600 ft. to the round trip. The entries and rooms are fairly level throughout the mine. The locomotives take all loaded cars from the working faces and make up trips of six to sixteen cars, depending on the rooms working in an entry, and then pull the trip to the trolley parting. They return with the same number of empties as they brought out, in order to keep the turn even. The empty cars are left at the room necks and pushed to the face by the loaders.

"The number of cars handled per day by the average locomotive depends entirely on the mine conditions, such as the number of rooms working on an entry and the number of men working in those rooms. One day all of the loaders may work, while the next day from five to forty men may be out; this varies the tonnage and work done by the individual locomotives. When the first storage-battery locomotive was installed, a good motorman and trip rider, without difficulty, could gather from 140 to 160 cars in 8 hr.; as they became more familiar with the operation of the locomotive, the number of cars gathered gradually increased until as many as 264 cars have been gathered in an 8-hr. working shift by a locomotive operated by a good motorman and trip rider; gathering 200 to 210 cars is a common occurrence.

"The empty cars weigh approximately 1000 to 1200 lb. and hold from 2200 to 2600 lb. net of coal; all are of the plain-bearing type. Taking the weight of a loaded car as 3200 lb. and fifteen cars per trip, a train load weighs 24 net tons, plus 5 to 5½ tons for the locomotive. Thus, 220 tons net of coal are gathered and hauled to the parting per day by a locomotive gathering 200 cars. Unfortunately, due to our small cars our turnover is very large for the tonnage hoisted.

"Taking everything into consideration, five to six mules and as many drivers would be required to replace one battery locomotive where it has a chance to work at least 7 hr. a day continuously. In addition, we would be compelled to brush in order to make head room for the mules. The

locomotives are 33 in. in height from the rail to the top of the locomotive and weigh 5 tons.

Taking the cost of one locomotive, charging panel and necessary equipment to operate same as......................................	$5000.00
Interest at 6 per cent...	300.00
Depreciation on locomotive chassis at 5 per cent.....................	195.60
Repairs, such as brake shoes, lamps for headlights, springs, controller parts, etc., at $3 a month, which is ample.......................	36.00
Depreciation of battery, per year...................................	1056.00
Labor, 1 hr. a day at 94 cents an hour, to insert charging plugs and do any miscellaneous work necessary on the locomotive, as we cannot pay the men less than 1 hr. for overtime...........................	188.00
One motorman, at $8.13 a day......................................	1626.00
One trip rider, at $7.50 a day.......................................	1500.00
Current for recharging battery, average of 25 kw. per day, and covering all line losses $1.50..	300.00
Total cost of operating one battery locomotive for one year, working 200 days...	$5201.60

The cost of gathering with mules and drivers to replace one storage-battery locomotive, using six mules and six drivers for this work is as follows:

Cost of six mules, at $135 each....................................	$810.00
Interest at 6 per cent..	48.60
Depreciation, at 25 per cent., on mules in low coal....................	202.50
Keeping six mules 365 days at $1.25 each, which covers the cost of feed, harness, shoeing, insurance, stable care, barn boss, medicines, doctor, and other expenses in taking the mules in and out of the mines daily.	2,737 50
Six drivers, at $7.50 a day..	9,000.00
Total cost of operating six mules, 200 working days a year.............	$11,988.60

If it would require six mules and six drivers, at a cost of $11,988.60, to do the work of one storage-battery locomotive and the total cost of operating a locomotive is $5201.60, there is a net saving in favor of the locomotive of $6787.00; or more than sufficient money to pay for the complete cost of installation of a locomotive. Even though the conditions are liable to change in some parts of the mine, where four mules could do the work of one locomotive, the locomotives would still be the more economical.

The Bon Ayr Coal Co., of Indiana, says: "Our cars are of the roller-bearing type, weigh approximately 2000 to 2200 lb. empty, and carry 6000 lb. of coal net. We have two 6-ton storage-battery locomotives in operation and two others ordered. The longest run that a locomotive has to make in a given entry is 1700 ft., or 3400 ft. to the round trip, of which 200 ft. has a grade of 2 per cent. against the load and 200 ft. has a grade of 5 per cent. in favor of the load. The balance of the road is fairly level. One-half of the rooms are fairly level or slightly in favor of the load; the other half have a grade of 1½ to 2 per cent. against the load.

· "The storage-battery locomotives, as well as the reel-gathering locomotives, leave the bottom of the shaft with ten empty cars, which

they place at the necks of ten rooms. They then take from the working faces of ten rooms, the rooms being driven to a depth of 250 ft., ten loaded cars, making up a ten-car trip which is hauled to the shaft bottom. A 25-lb. rail is used in rooms and for cross entries.

Taking 2400 tons as the average hoist, with eight locomotives, each locomotive would have to handle 100 cars per shift, if the cars contained 3 tons each. But as it is impossible to make loaders average 3 tons net per car, it is often necessary for the locomotives to deliver 140 cars and take out 140 loaded cars in order to produce the tonnage of the shift.

"The locomotives are 39½ in. to 40 in. in height; the ties are 3 by 4 in. and the rail 3 in. high, which gives a total height of 46 in. from the bottom to the top of the locomotive. There is a 6-in. clearance between the top of the locomotive and the roof in our lowest coal. In many parts of our mine, the coal is much higher and naturally gives a greater clearance.

"The locomotives have gathered 16-car trips, but when working twenty rooms on an entry, with all men in, the 10-car trips work out satisfactorily, as that leaves ten rooms for the motormen to drop off ten empties and ten rooms from which to take out loaded cars. If there should be twenty-four rooms working on an individual run, the motorman would deliver twelve cars and take out twelve loaded ones.

"Mules would not be able to handle our work, as the cars, if loaded to capacity, would weigh over 4 tons, and the best the average mule would be able to handle would be one car at a time, so the mine would be full of mules. Also, 3400 ft. for a round trip with heavy cars would be beyond the mule capacity through a shift.

"The reel-type of gathering locomotives has given good service, but the storage-battery locomotives will gather more coal at less expense. Also, if the power is off temporarily and there is plenty of coal shot down they can continue to gather and haul the coal until the supply of empty cars has been exhausted. Besides, for miscellaneous-hauling on idle days or nights, it is not necessary to start up a generating unit of any kind in order to operate them.

"At the end of a day's work the locomotives are charged through a rheostat switchboard from a 250-volt direct-current circuit. As the switchboard is automatic, an attendant is unnecessary. The charge is started on the locomotives and as each battery is fully charged the charging circuit is automatically cut off. It is necessary to add water two or three times a week to each battery to replace the evaporation; this requires about 15 min. for each locomotive. Outside of general inspection, oiling, etc., no attention is required other than that which should be given any piece of mining equipment at the end of the day's work."

John B. Hicks, assistant superintendent of power and mining, Con-

solidation Coal Co., in a paper read before the Kentucky Mining Institute, June 4, 1920, said: "The following figures include renewals, daily care of battery, which includes the flushing, as well as all necessary repairs made to battery. Over a period of three years 3,003,361 tons were hauled at a cost of 12 cents per ton. These figures do not give the battery credit for the recovery of rails and pillar work or the hauling of supplies or bailing of water with these locomotives, but only the actual coal tonnage hauled."

In all of the above, only strictly the storage-battery locomotive, as distinguished from the combination storage-battery and trolley type locomotive, has been considered. The primary purpose of the latter locomotive has been the conservation of the battery charge. This locomotive, when operating under a trolley wire, will receive its power
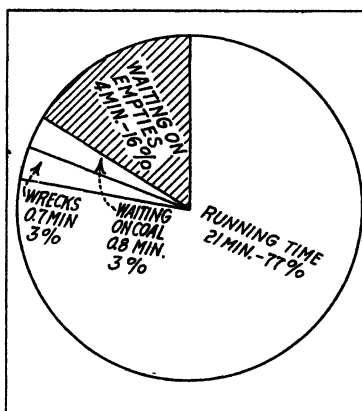
FIG. 7.—PERFORMANCE OF MOTORS, MINE No. 11. AVERAGE TRIP, 27.39 MINUTES.
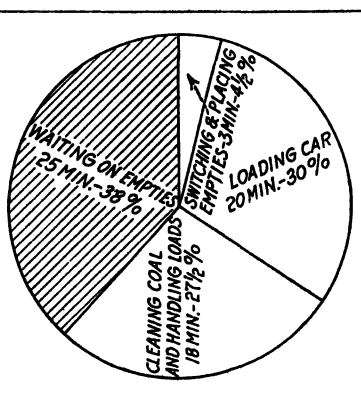
FIG. 8.—PERFORMANCE OF LOADERS, MINE No. 11. AVERAGE TRIP, 66 MINUTES.

directly from the trolley wire and at the same time the storage battery will receive a charge.

Table 1 was compiled after a careful study for a month at the Lynch mines of the United States Coal and Coke Co., at Lynch, Ky.

The performance of motors and loaders in a given mine, as obtained by a careful time study, is given in Figs. 7 and 8. Fig. 7 shows the time lost by a locomotive while waiting on empties. In this case the loss is very small, being but 16 per cent. of the total. Fig. 8, on the other hand, shows that the time spent by loaders waiting for empties is 38 per cent. of the total. It was the opinion of the mine operating department that additional locomotive equipment was necessary, but these charts clearly indicate that the curtailment of production was caused by an insufficient number of mine cars. The purchase of additional locomotives, instead of being helpful, would have been hurtful in this

Monthly Report, July, 1920.

Combination

At U. S. C. & C.

Date	Hours Worked	Driver's Wages 70 Cents an Hour	Coupler's Wages 65 Cents an Hour	Motor-house Rental 20 Per Cent. of Super. Cost	Int. on Inv., 6 Per Cent.	Depreciation on Locomotive, 10 Per Cent.	Depreciation of Battery 7 Yr. Life	Supervision Cost per Locomotive	Current Consumption, Kilo-watt-Hours	Current Costs $0.0175 per Kw-hr.	Repair Parts $150 per Loco-motive Year	Repair Labor, 1 Hr. per Day	Time for Repairs, Hours	Repair Shop Cost, 20 Per Cent. of Parts and Labor Costs	Distance Covered per Day, Feet	Number of Trips	Number of Loads Pulled	Number of Empties Spotted
1	11	$7.70	$7.15	$0.12	$1.32	$1.23	$1.62	$0.60	21	$0.37	$0.41	$0.80	1	$0.24	48,690	3	37	41
2	10	7.00	6.50	0.12	1.32	1.23	1.62	0.60	20	0.35	0.41	0.80	1	0.24	48,216	2	40	48
3	9	6.30	5.95	0.12	1.32	1.23	1.62	0.60	16	0.28	0.41	0.80	1	0.24	48,080	2	40	36
4				0.12	1.32	1.23	1.62	0.60			0.41	0.80	1	0.24				
5	9	6.30	5.95	0.12	1.32	1.23	1.62	0.60	24	0.42	0.41	0.80	1	0.24	45,350	2	37	36
6	10	7.00	6.50	0.12	1.32	1.23	1.62	0.60	26	0.46	0.41	0.80	1	0.24	55,906	2	51	55
7	11	7.70	7.15	0.12	1.32	1.23	1.62	0.60	27	0.48	0.41	0.80	1	0.24	81,228	3	67	65
8	11	7.70	7.15	0.12	1.32	1.23	1.62	0.60	24	0.42	0.41	0.80	1	0.24	89,896	3	74	84
9	10	7.00	6.50	0.12	1.32	1.23	1.62	0.60	13	0.23	0.41	0.80	1	0.24	50,381	2	34	49
10	10	7.00	6.50	0.12	1.32	1.23	1.62	0.60	27	0.48	0.41	0.80	1	0.24	93,666	3	78	77
11				0.12	1.32	1.23	1.62	0.60			0.41	0.80	1	0.24				
12	11	7.70	7.15	0.12	1.32	1.23	1.62	0.60	21	0.37	0.41	0.80	1	0.24	50,360	3	37	41
13	11	7.70	7.15	0.12	1.32	1.23	1.62	0.60	28	0.49	0.41	0.80	1	0.24	92,022	3	77	67
14	10	7.00	6.50	0.12	1.32	1.23	1.62	0.60	27	0.48	0.41	0.80	1	0.24	93,004	5	63	61
15	11	7.70	7.15	0.12	1.32	1.23	1.62	0.60	26	0.46	0.41	0.80	1	0.24	97,335	3	61	61
16	11	7.70	7.15	0.12	1.32	1.23	1.62	0.60	28	0.49	0.41	0.80	1	0.24	106,526	4	77	80
17	11	7.70	7.15	0.12	1.32	1.23	1.62	0.60	29	0.51	0.41	0.80	1	0.24	103,393	4	69	74
18				0.12	1.32	1.23	1.62	0.60			0.41	0.80	1	0.24				
19	12	8.40	7.80	0.12	1.32	1.23	1.62	0.60	28	0.49	0.41	0.80	1	0.24	98,824	4	63	60
20	10	7.00	6.50	0.12	1.32	1.23	1.62	0.60	30	0.53	0.41	0.80	1	0.24	47,946	2	50	50
21	12	8.40	7.80	0.12	1.32	1.23	1.62	0.60	29	0.51	0.41	0.80	1	0.24	99,065	3	60	62
22	11	7.70	7.15	0.12	1.32	1.23	1.62	0.60	26	0.46	0.41	0.80	1	0.24	86,621	4	64	62
23	12	8.40	7.80	0.12	1.32	1.23	1.62	0.60	27	0.48	0.41	0.80	1	0.24	81,630	3	55	53
24	12	8.40	7.80	0.12	1.32	1.23	1.62	0.60	26	0.46	0.41	0.80	1	0.24	117,748	4	73	79
25				0.12	1.32	1.23	1.62	0.60			0.41	0.80	1	0.24				
26	12	8.40	7.80	0.12	1.32	1.23	1.62	0.60	27	0.48	0.41	0.80	1	0.24	99,660	5	63	61
27	12	8.40	7.80	0.12	1.32	1.23	1.62	0.60	29	0.51	0.41	0.80	1	0.24	80,140	4	58	58
28	10	7.00	6.50	0.12	1.32	1.23	1.62	0.60	22	0.39	0.41	0.80	1	0.24	91,004	4	59	59
29	10	7.00	6.50	0.12	1.32	1.23	1.62	0.60	23	0.41	0.41	0.80	1	0.24	117,570	4	75	72
30	11	7.70	7.15	0.12	1.32	1.23	1.62	0.60	22	0.39	0.41	0.80	1	0.24	69,748	3	53	52
31	10	7.00	6.50	0.12	1.32	1.23	1.62	0.60	24	0.42	0.41	0.80	1	0.24	106,609	4	60	60
		$203.00	$188.70	$3.72	$40.92	$38.13	$50.22	$16.20	kw-hr. 670	$11.71	$12.71	$24.80	31	$7.44	2200,622 416.7mi. 15.4mi. per day	Avg. 3.2 per	Avg. 58.3 day	Avg. 59.

SUMMARY

Tons hauled in month.. 4098.4
Total cost per month.. $597.55
Average cost per ton.. $0.145
Mules replaced.. 8
Mule drivers and crab men replaced.. 8
Cost of operating per mile.. $0.135

REMARKS

Previous equipment used in same section, 8 mules, 8 men, and 2 crab units.
Feed @ $20.00 per month each... $160.00
Depreciation at 25 per cent. at an average cost of $175.................. 29.16
Interest on investment @ 6 per cent..................................... 7.00

Locomotive
Co.'s Mine, Lynch, Ky.

Locomotive No. 908 Type "D.T."

Operative Costs | **Total Costs**

Average Distance Covered per Trip, Feet	Average Distance Traveled per Car, Feet	Average Level Distance per Trip of 7.7 of Total 43 Per Cent., Feet	Average Distance per Trip of Total 35 Per Cent., Average of 7.7 Per Cent. Against Loads, Feet	Average Distance per Trip of Total 22 Per Cent., Average of 4.1 Per Cent. Favor of Loads, Feet	Time Hauling Loads Hr.	Min.	Time Spotting Empties Hr.	Min.	Time Lost Waiting for Empties Hr.	Min.	Time Lost Waiting for Loads Hr.	Min.	Time Lost, Other Reasons Hr.	Min.	Average Time per Car, Full Trip, Minutes	Load Weight in Cars, Coal and Locomotive, Hauled per Trip, Tons	Load Weight in Cars and Locomotive, Hauled per Trip, Tons	Net Tons Coal Hauled per Day	Mules Replaced	Men Replaced	Fixed Costs	Maintenance Costs
16,230	623	6,978.9	5,680.5	3,570.6	2	50	3	51				30	3	53	4.7	66.0	35.9	96.2	8	8	$19.74	$1.42
24,108	547	10,366.4	8,437.8	5,305.7	1	57	2	09			2	06	2	58	2.8	106.0	61.2	104.0	8	8	18.39	1.50
24,040	632	10,337.2	8,414.0	5,288.8	2	30	1	58			2	04	2	08	3.5	106.0	39.4	104.0	8	8	17.14	1.73
																					4.29	1.45
22,675	621	9,750.2	7,936.2	4,988.5	2	02	2	29			1	09	3	20	3.7	93.5	39.4	96.2	8	8	17.14	1.87
27,953	534	12,019.7	9,783.5	6,149.6	1	27	1	47		35	2	42	2	57	2.2	118.5	53.1	106.6	8	8	18.39	1.91
27,076	691	11,642.6	9,476.6	5,956.7	2	47	2	58	1	40	1	10	2	20	2.2	117.5	55.6	172.2	8	8	19.74	1.93
29,965	607	12,884.9	10,487.7	6,592.3	3	06	3			47		48	2	49	2.4	126.0	61.2	192.4	8	8	19.74	1.87
25,190	607	10,831.7	8,816.5	5,541.8		45	1	20	6	35			1	20	2.0	91.0	61.2	88.4	8	8	18.39	1.68
31,222	650	13,425.4	10,927.7	6,868.8	2	35	2	25			1	20	3		3.8	136.0	65.8	202.8	8	8	18.39	1.93
																					4.29	1.45
16,786	645	7,217.9	5,875.1	3,692.8	2	36	3	21	1	35			2	58	4.5	66.0	35.9	96.2	8	8	19.24	1.82
30,674	639	13,189.8	10,735.9	6,748.2	3	19	2	59		30		10	3	42	5.2	136.0	56.6	200.2	8	8	19.74	1.94
18,600	750	7,998.0	6,510.0	4,092.0	2	42	3	34		50		34	2	25	3.0	66.0	33.6	163.9	8	8	18.39	1.93
32,448	798	13,952.6	11,356.8	7,138.5	3	03	3	56	1	08			2	33	3.4	106.0	52.0	158.6	8	8	19.74	1.91
26,657	678	11,462.5	9,329.9	5,864.5	4	05	3	35				25	2	55	2.9	101.0	52.0	200.2	8	8	19.74	1.94
25,843	738	11,112.4	9,045.0	5,685.4	3	10	4	20					2	40	3.1	91.0	48.5	179.4	8	8	19.74	1.96
																					4.29	1.45
24,706	803	10,623.5	8,647.0	5,435.3	3	48	4	45					2	42	4.0	89.5	34.5	172.2	8	8	21.09	1.94
23,973	479	10,308.3	8,390.5	5,274.0	2	30	4	05					3	05	3.9	131.0	63.5	130.0	8	8	18.39	1.98
33,021	812	14,199.0	11,557.3	7,264.6	3	10	3	10					4	55	3.1	106.0	53.3	156.0	8	8	21.09	1.96
21,655	687	9,311.6	7,579.2	4,764.1	2	45	2	55	1	30		33	2	47	2.7	86.0	41.6	166.4	8	8	19.74	1.80
27,210	755	11,700.3	9,523.5	5,986.2	3	46	4	19				14	3	26	4.4	97.5	36.4	165.0	8	8	21.09	1.93
29,437	774	12,657.9	10,302.9	6,476.1	3	24	4	26	2	20			2	05	3.0	97.0	51.3	189.8	8	8	21.09	1.91
																					4.29	1.45
19,832	803	8,527.7	6,941.2	4,363.0	3	38	3	48	2	40		10	1	29	3.5	69.0	33.6	163.8	8	8	21.09	1.93
20,035	690	8,615.0	7,012.2	4,407.7	3		4	29	1	40		15	1	39	3.8	76.5	39.3	150.8	8	8	21.09	1.96
22,751	771	9,782.9	7,962.8	5,005.2	2	40	3	07			2	15	1	38	2.9	79.5	39.8	153.4	8	8	18.39	1.84
29,392	799	12,638.5	10,287.2	6,466.2	3	22	3	55				15	2	03	2.9	99.5	39.4	195.0	8	8	18.39	1.96
23,249	664	9,997.0	8,137.1	5,114.7	2	10	2	18	2	40		41	2	45	2.5	94.0	45.7	137.8	8	8	19.74	1.84
26,652	838	11,460.3	9,328.2	5,863.4	2	42	2	55			1	17	2	46	2.8	81.0	40.5	156.0	8	8	18.39	1.87
Avg. 25,236	Avg. 690	Avg. 10,851.5	Avg. 8,832.7	Avg. 5,551.9	75	49	87	10	24	30	18	38	73	18	Avg. 3.3	2,635.0	1,270.3	4,098.4	8	8	$540.89	$56.66

REMARKS.—*Continued*

Shoeing, stabling, and harness...................................	$ 16.00
Drivers and crab men at actual time locomotive worked...	1,624.00
Monthly costs...	$1,836.16
Locomotive monthly costs..	597.55
Saving monthly..	$1,238.61
Saving yearly...	$14,863.32

COMPARATIVE RESULTS

Per ton cost hauled by mules......	$0.448
Per ton cost hauled by locomotive.	0.145
Per ton saving...................	0.303

case, both from an operating point of view and from the standpoint of cost.

This fact is further shown by Fig. 9, which is the summary of a series of analyses made at several mines. The performance shown is more characteristic of delays encountered or permitted than is indicated in Figs. 7 and 8. To sum up, a locomotive should not be blamed for those conditions that prevent the locomotive from working to the capacity for which it is built. Many locomotives are blamed for delays of one kind and another that reduce the tonnage handled by it, but this is more true of the storage-battery locomotive than of any other type.
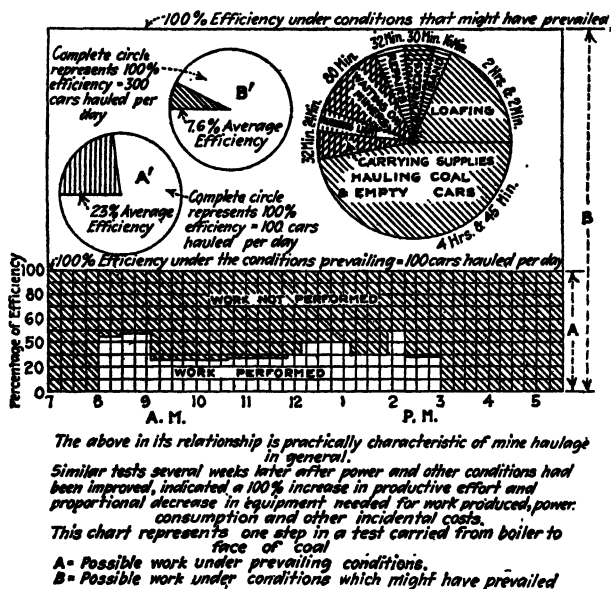
FIG. 9.—GRAPHIC ANALYSIS OF DAILY PERFORMANCE OF LOCOMOTIVES.

One important manufacturer has developed a combination locomotive. It is equipped with two motors, one driven by the storage-battery charge and the second by a current from the trolley wire. This second motor is built up to 35 hp. for the 6-ton machine and the locomotive is designed, when driven by this motor, to produce its rated drawbar pull at 6 mi. per hr. The successful application of this principle will represent a valuable step forward in the development of locomotives for mine haulage. There are many conditions in which, in addition to the actual work of gathering, the locomotive should be capable of strictly haulage duty, in the sense of speed and sustained drawbar pull.

On account of space limitation, the motor receiving the full voltage from the wire has a lower rating than the present haulage locomotive. though equal to that applied to the haulage locomotive of a few years ago,

At the same time, when this locomotive is being driven by the storage battery, for gathering, the motor receiving primary voltage will have an opportunity to cool. However, in any event, its application should be determined by a study of conditions.

As was stated in the beginning, the writer has not touched upon the mechanical construction, motors, or battery equipment. There are on the market several locomotive designs and several types of batteries. To some extent, the difference with respect to battery is of a character to make selection of battery rest largely on the investment that a purchaser is prepared to make in the first instance; this is a subject that should be treated in a special paper.

CONCLUSIONS

To sum up, the storage-battery locomotive is proving a complete success and a means for reducing costs as well as risk of accident, where properly applied. Its failure at any mine will be accounted for by one of the following:

1. A duty cycle in excess of battery capacity; this has been the most frequent cause for failure.

2. Misapplication is a common error; that is, the effort to use the storage-battery locomotive where mule power is cheaper or where man power in conjunction with the strictly haulage type of locomotive would be cheaper.

3. Failure to select a locomotive that will stand the abuse of mine practice.

4. Failure to provide a reasonable amount of care and attention.

DISCUSSION

HOWARD N. EAVENSON, Pittsburgh, Pa.—The conditions under which the storage-battery locomotives operated at Lynch were exceptionally favorable to the locomotive and exceptionally unfavorable to the mule. The top was low and the grades heavy, so that mules could not be used economically.

I am not opposed to the storage-battery or any other type of locomotive, where it can be used, but the United States Coal and Coke Co. has found that for grades up to 8 or 9 per cent. and workings over 5 ft. in height, where the cars must be delivered one at a time at the face or taken from there one at a time, that no type of mechanical haulage can compete with the mule. The average cost is considerably under half of the savings made in this particular instance. That condition, of course, may not hold everywhere, but it does in the mines of this company in Lynch and in the Pocahontas field.

CHARLES E. STUART.—Of course in the case of the high coal, the conditions are less favorable to the storage-battery or the reel-type of

locomotive. Can Mr. Eavenson tell us what a mule can do under a thoroughly concentrated mining system, also what the storage-battery locomotive and what the gathering type of trolley locomotive will do, because there is the point on which the question hinges. The mule is cheaper in first cost and is not difficult to maintain; so it is really a question of how many mules can be displaced by the use of the locomotive under the most concentrated system of mining.

HOWARD N. EAVENSON.—Under the best conditions, we have had mules that would gather (that is, one driver with two mules ordinarily) 75 to 100 cars a day; but the average was from 30 to 35 cars. Of course, this is taking the cars from the face of the working place one at a time and putting back the empties. We have never had any type of gathering motor that would do more than twice that much. Inquiries at a number of mines where gathering locomotives were used showed that from 75 to 150 cars per day were placed, but either more than one car was put at the face at a time or else the men took the cars from the face and put them at the room deck.

I do not know of a case where, putting the cars in one at a time could be done cheaper by mechanical means than by mule, for the mule can go through a breakthrough and must travel only one-half the distance that any kind of a mechanical apparatus has to go. In some cases where double-track rooms are used, the mule puts an empty on one track, goes across 20 or 30 ft. and takes down a load. As a general proposition, under our conditions, the output of any kind of a gathering motor was about double that of a driver with two mules.

WILLIAM KELLY, Vulcan, Mich.—In the iron ore mines, one mule hauls four 3-ton cars, making a load of 12 tons of ore; in some cases a mule has hauled six cars, but the regular practice is to take four cars.

ARTHUR THACHER, St. Louis, Mo.—On what grades can the storage-battery locomotive be used?

W. H. PATTON, Philadelphia, Pa.—In one case they were used on a grade of 17½ per cent.; we sanded the tracks, backed down into the room, and pulled out two cars weighing 9600 lb. each, with a 6-ton locomotive, without the wheels spinning.

CHARLES E. STUART.—In the case of the two Indiana companies mentioned in the paper, the locomotives were used in high coal, comparatively speaking. Does Mr. Eavenson criticize the operating figures given, which show a distinct saving in favor of the storage-battery locomotive?

HOWARD N. EAVENSON.—I have not had the time to analyze the figures and it was not my intention to criticize the storage-battery locomotive. I simply gave the experience at the mines with which I have

been connected, which has also been the experience of the largest operating company in the Connellsville region. I do not know the conditions in Indiana, but I question if only one car is put at the face at one time; we have never been able to do it. I wish we might have, because our policy has always been to use mechanical arrangements if possible.

CHARLES E. STUART.—I do not hold any brief for the storage-battery locomotive; is it not possible that those mines to which Mr. Eavenson has referred presented the conditions indicated in Figs. 5 and 6? In other words, here is a case where a locomotive would have to cover about 75 cars; no type of locomotive would be applicable under that condition. The point that I have tried to bring out is that the economical use of a locomotive is entirely a question of a given condition.

The Queens Coal and Mining Co., under the conditions shown, claims to gather 200 to 210 cars in a day and to have displaced six mules and six drivers. The Bon Ayr Coal Co. has displaced not only mules but also the reel-type locomotive, and claims to have gotten results that, in dollars and cents, are favorable to the storage-battery locomotive. However, the whole question is what the machine will do and what the mules will do under a given condition.

GRAHAM BRIGHT, East Pittsburgh, Pa.—The author has given a new viewpoint on the application of storage-battery locomotives in mines, and while he has made some recommendations in regard to scheduling more closely the locomotive operation in a mine, I do not believe that he thinks that this could be accomplished as well as in an industrial plant. The constantly changing conditions and the difficulty of supervision in coal mines, make it difficult to schedule operations closely, but much can be done toward improving these conditions.

I do not agree with the statement that the small operation is a good place for a storage-battery locomotive. Some of the larger operators have found that storage-battery locomotives give good service if there are sufficient of them to be cared for by an expert; but as it does not pay to keep a high-grade man to care for only one or two, in many cases they are not satisfactory. Therefore, unless the mine is big enough to require enough locomotives to keep an expert it should use the gathering trolley-type locomotive.

The author says that in some of the earlier locomotives the motors were of insufficient capacity; my experience has been that, in the earlier locomotives, the battery was discharged before the motor had a chance to heat up. The battery capacity has been the real limiting feature. If you were to instal on present locomotives motors that were applied seven or eight years ago with some of the large batteries that are being used at present, the motors might start to give trouble.

The paper mentions a combination type of locomotive in which one motor is used on the trolley and another on the battery. I question the advisability of this scheme, for one-half of the motor equipment is idle most of the time. When the locomotive is on the trolley, it is necessary to accelerate the battery armature and retard it every time a start is made and vice versa. A better scheme would be to use the two motors all the time, using them in parallel on the battery and in series on the trolley. In the long run a scheme of that kind will prove more economical than one which puts the entire capacity in each motor and uses them alternately.

Safety Practice for Hoisting Ropes

By R. M. RAYMOND,* NEW YORK, N. Y.

THE Mining Section of The National Safety Council recently sent out a questionnaire to operators, regarding the class of rope used, specifications required or obtained, factors of safety observed, methods of fastening ropes, lubrication, inspection, and points observed when rope is to be discarded. The answers received from this questionnaire, largely from metal mines, give interesting information, from which recommendations for safety practice on hoisting ropes are drawn.

DEPTH OF MINE

The depth in the majority of cases noted is from 1000 ft. to 1500 ft. (304 to 457 m.) and records come from a number hoisting from depths of 2000 to 3000 ft. Hoisting is generally done in balance, with a few companies using counterweights.

Seldom, more than two shifts are occupied in hoisting. A number of large operators limit the shift for engineers to 6 hr. A few of the larger operators have separate hoists for men and supplies, with a higher class rope for men. The range of loads varies from 7 to 17 tons and greater in a few cases, with a balanced weight of 4 to 7 tons.

SIZE OF DRUMS AND SHEAVES

The size of drums and size of sheaves are generally in accordance with recommendations of rope manufacturers. Only one-half of the operators use grooved drums; one, The Sulphide Corpn., New South Wales, uses wood-lagged drums. Very few operators have drums capable of holding all the rope in a single wrap. Cone drums or cone and cylinder drums, in the mines heard from, are very rare.

A few mines of shallow depth use a small drum and large-sized sheave. While this may be allowed, and in some cases is all right, the size of drum and sheave should be fully equal to that recommended by the rope manufacturers and should be at least 60 to 100 times the diameter of the

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rope used, or one thousand times the diameter of the largest wire in any of the strands.

Operations show two wraps wherever possible, and not more than three wraps on the drum; the choice of several would be not more than one wrap if possible, and not more than two as good practice. A few operators, though too few, give importance to the distance at which the engine is set from the shaft and the fleet angle between sheave and drum. This, in most cases, can readily be arranged to reduce the fleet angle to 1° 30', and provide at the same time a drum that will hold all the rope in one wrap or in not over two wraps. Hans Behr recommends, in specifications for hoisting engines, that the last groove shall rise gradually above the other grooves in order to start the second layer without shock. If, beside, grooved drums are used, these points will reduce the abrasion on the rope to a minimum.

The matter of overwind and underwind of ropes receives perhaps too little attention. The experience of operators in South Africa, according to Vaughan, of the Mines Department, indicates that the rope which underwinds wears more rapidly than the overwind rope by 20 per cent., and that the average life of rope is prolonged by alternating the ropes on the drums.

In a majority of cases, sheaves are cast iron, turned out true, $\frac{1}{16}$ or $\frac{1}{8}$ in. larger than the size of rope used. Others use special lining of cast iron, which is removable; the best type of this is the Nordberg sheave. Several companies use hardwood blocks, preferably maple. A few use hemp packing at first or to fill the groove when the sheave becomes worn.

More attention should be given to the material and weight of sheaves. These should be made as light as possible, the bicycle type preferred; of larger diameter generally than recommended by rope manufacturers; set very firm in bearings, well babbitted; aligned carefully with center of drum, observing the lead or fleet angle mentioned previously. Wooden linings have generally proved satisfactory where used, though such linings add greatly to the weight and require much care in renewal. The best wood lining is maple soaked in linseed oil. Cast-iron sheaves or cast-iron liners prove most satisfactory. Grooves must be kept true and machined to the same diameter of drums and to the proper width of groove when new rope is put on. Top of head frames and sheaves should be made more easily accessible for frequent inspection.

Size of Rope

Ropes vary in size from 1 to $1\frac{1}{2}$ in.; generally they are 6 × 19 with a hemp center. Most operators use the higher class rope, plow steel or special brands, improved; a few use a flattened strand, 6 × 25. Many

prefer the Lang lay, which is the favorite with the larger operators hoisting from the greater depths. Lang-lay ropes are used in nine cases out of ten in South Africa; they are generally used in Australia, and their use is becoming more general in this country. The Sulphide Corpn. finds that with 6 × 7 Lang-lay rope, improved plow-steel wires, both longer service and more satisfactory inspection for wear and corrosion can be obtained. The experience of the New Jersey Zinc Co. is much the same; particularly in its incline shaft, it finds plow-steel wire gives the best service. Care is required, in using the Lang-lay rope, in the early operations, to see that no bird-caging or kinks develop. When chairs are used, no more rope should be unwound from the drum than is required to reach the chairs. A solid attachment for Lang lay should always be made, not a swivel attachment. Possible twists should be taken out, with much care, once or twice when new rope is put into service.

ROPE SPECIFICATIONS

Few companies stipulate specifications for rope; reliance is placed on the tensile strength given by the manufacturers. Preference is shown for plow steel by the larger operators, with the understanding that the tensile strength of the wires is, or approaches, 200,000 lb. per sq. in. The International Nickel Co. gives a full description of working conditions and asks manufacturers to recommend a rope, with complete description. It requires a factor of safety of seven for hoisting ore and a factor of ten for the man hoist. The United Verde mine stipulates low-phosphorus steel, high carbon and manganese, with a tensile strength of the wires of 200,000 lb. per sq. in. The Copper Range Co. stipulates 1¼-in. plow steel, 6 × 19, regular lay; 58 tons tensile strength, and on test has found the rope to have a strength of 75 tons. The Broken Hill mine uses 1¼-in. 6 × 19, Lang lay, and specifies a breaking stress of the rope of 60 tons and best flexible plow steel or its equivalent. The Anaconda company does not specify particularly, but uses improved plow-steel in all cases.

While the manufacturers' recommendations seem to be generally satisfactory, it is most important that purchasers give clearly their working conditions, nature of shaft lining, attention given to timbers and guides, the factor of safety desired, whether mine water is acid or not, and whether harmful gases are encountered in the lower part or any portion of the mine.

A more satisfactory statement from the manufacturers would be the tensile strength of steel from which the wire is drawn, as well as the breaking strength of the rope; the aggregate strength of all the wires is 10 to 20 per cent. less than that of the wire bar. When new rope is purchased, a certificate should be required from the manufacturer,

the rope should be stored under cover, in a cool place, properly coiled and not left on the ground.

The Canadian mining law requires a certificate from the manufacturer showing date of manufacture, size, weight, number and size of wires, class of core, breaking stress of steel from which wire is made, and estimated breaking load of rope. This certificate is recorded in the "Rope Record Book" and additional data kept of operation of rope when in use at the mine.

FACTOR OF SAFETY OF ROPES

The factor of safety stipulated or estimated varies considerably; generally that estimated by the manufacturers in supplying rope is considered satisfactory; this is five to six for new rope. Factors of safety for worn rope vary from three and one-half to five, and these variations for discarding rope correspond somewhat with speed and depth.

None of the state laws in this country indicate any factor of safety. The Canadian mining law stipulates a factor of safety of six in shafts less than 2000 ft., deep; and in shafts 2000 to 3000 ft., not less than five. The International Nickel Co. uses a factor of seven and specifies such a rope for its conditions. The commission appointed in South Africa to inquire into the use of hoisting ropes considered this matter carefully and recommended a minimum factor of safety of six, but believes that for deep levels it is impossible to start with a factor of safety of more than seven. Such a rope, when reduced to a factor of safety of six, may be used for six months longer but no more, and even then will have an enormous reserve of strength. It was proved also that shocks due to changes in velocity were dangerous only to short lengths; in lengths of 1500 ft. and over shocks would remain constant. The New South Wales mining law requires that a rope be tested before use and that it have a factor of safety of eight for ore and of ten for men. The Sulphide Corpn. uses a factor of safety of nine for new rope on ore and of ten for rope on man hoists. The regulations of the new mining law proposed for this country are as follows:

The factor of safety of ropes when newly installed in shafts less than 3000 ft. deep shall in no case be less than six.

It shall be unlawful to use any rope or cable for the raising or lowering of men when its factor of safety, based on its existing strength and dead load, shall have fallen below 4.5.

It shall be unlawful to use any rope or cable of the so-called 6 × 19 standard construction for the raising or lowering of men, either when the number of broken wires in one lay of said rope exceeds six, or when the wires on the crown of the strands are worn down to less than 65 per cent. of their original diameter, or when the superficial inspection provided for in this section shows marked signs of corrosion; provided however, that when such broken wires are reduced by wear more than 30 per cent. in cross-section, the number of breaks in any lay of the rope shall not exceed three.

The committee framing the law discussed the matter of safety with manufacturers and a variable safety factor was recommended, dependent on the depth of shafts. This is given in Table 1.

TABLE 1.—*Factors of Safety Recommended for Hoisting Ropes*

Length of Rope, in Feet	Minimum Safety Factor for New Rope	Minimum Safety Factor when Rope must be Discarded	Percentage Reduction
500 or less............................	8	6.4	20.0
500 to 1000...........................	7	5.8	17.0
1000 to 2000..........................	6	5.0	16.5
2000 to 3000..........................	5	4.3	14.0
3000 and over.........................	4	3.6	10.0

It would seem that the variable factor of safety suggested for the new mining law is a good recommendation, but a maximum factor of safety of not less than six, and a minimum of not less than four and one-half would be better for depths below 2000 ft. Where man hoists and ore hoists are in use in the same shaft and the same size of rope used, it would be well to change the ropes from the man hoists to the ore hoists after a period of six months or a year.

HOISTING SPEED

Speed in the shallow shafts submitted is from 600 to 1000 ft. per min. for ore, and in deeper shafts 1600 to 3000 ft. per min. Speed for hoisting men in almost all cases is reduced to 400 to 600 ft., though in a few cases it runs as high as 1000 ft. per min. Acceleration varies considerably, depending on depth. Table 2 shows acceleration practiced by several of the important operators.

TABLE 2.—*Acceleration in Various Mines*

	Shaft	Average Length of Haul, Feet	Maximum Hoisting Speed, Ft. per Min.	Acceleration, Ft. per Sec. per Sec.	To Reach Max. Speed, Feet	Time to Reach Max. Speed, Seconds
N. J. Zinc Co., Franklin, N. J........	Inclined	915	3000	3.33	375	15
Inter. Nickel Co., Creighton, Ont.....	Inclined	1600	1100	2.33	260	15
Witherbee-Sherman, Mineville, N. Y..	Inclined	1000	1200	1.33	150	15
Copper Range Co., Painesdale, Mich...	Inclined	1500	2360	2.48	130	10
Republic Iron and Steel Co., Birmingham, Ala.........................	Inclined	5400	2200	1.47	459	25
Sulphide Corpn., N. S. W., Australia...	Vertical	825	3650	5.06	300	15
Calumet and Arizona, Warren, Ariz....	Vertical	1500	1600	5.33	70	5
United Verde Copper Co., Clarkdale, Ariz..............................	Vertical	850	2000	4.7	115	10
North Butte Co., Butte, Mont........	Vertical	3600	2700	2.53	400	17
Great Boulder Prop., Boulder, W. A...	Vertical	2000	2000	5.85	95	6
Old Ben Coal Corpn., Frankfort, Ill....	Vertical	475	3600	12.00	150	5

Compared with the draft of the new mining law, the speeds for hoisting and accelerating practiced by the different companies do not show any excess over the speeds recommended. In fact, the acceleration speeds practiced are considerably lower than the maximum allowed acceleration of the proposed law. An accelerating speed of 3 to 4 ft. per sec. for depths of 2000 ft. is preferable to the maximum acceleration suggested. Speed for hoisting men varies considerably and practice varies in different countries. Speed on such occasions should not be over 500 to 600 ft. per min. For great depths, higher speed may be permitted, with the period of acceleration and of retardation extended. The Mines Inspection Act of New South Wales requires a cage speed no greater than 300 ft. per min. within 50 ft. of the surface or bottom, when hoisting men. Great care should be taken in starting and stopping, especially with new rope. It is most important that no slack be permitted when the hoist is resting on chairs; with a foot of slack in the rope's length, a jerk may cause a stress of four times the weight lifted.

INSPECTION OF ROPES

Method and frequency of inspection of ropes vary from daily inspection of terminals, including wraps remaining on drums, to weekly inspection. Fully 70 per cent. of the operators make inspection daily, and one makes inspection three times a day when hoisting men. Daily inspection is in some cases left to hoistmen, who in a few cases are depended on to watch and report broken wires. Daily inspection, generally, is made by the mine foreman, and minute inspection made weekly, twice per month or monthly, by the foreman and assistant to master mechanic, or other competent persons. In a few cases the entire length of rope is inspected carefully once a week, running the rope through the inspector's hands. One company inspects new ropes every two weeks with a magnifying glass. When rope shows signs of wear, inspections are made more frequently. Terminals are cut off every two or three months, in lengths of 3 to 10 ft.

Generally speaking, rope inspection is quite thorough; and from the experience of the greater number of the operators giving replies, a standard conclusion may be made as follows:

Daily inspection of terminals should be made, including wraps remaining on the drum when the cage or skip is down; daily inspection of both the terminals above cage or skip, and of the rope at point of contact with sheaves. On new ropes the latter spot is important; and while it varies slightly due to the stretch, a short cut every month for the first three months is a point worth considering   Thorough inspection of the whole length of rope should be made every week, and twice a week when the rope is new. Once a month portions of the rope should be carefully inspected in sections that are likely to show wear and the rope should

be cleaned for this purpose. Examination of rope fastenings should be made before hoisting men. The Mining Act of Canada stipulates that rope shall be examined once a month to ascertain deterioration. For this purpose the rope must be thoroughly cleaned at points selected by inspectors, who shall note reduction in circumference and the proportion of wear. Results of inspection should be made in Rope Record Book and a copy of weekly and monthly inspections sent to mine superintendent and mine manager.

FASTENING OF ROPE TO DRUM AND TO CAGE

Fastening on drums in all cases is made by three wraps, at least, remaining on drum when the cage is at the bottom. The rope is then passed through a hole in the drum at a very acute angle, with one or two turns around the shaft, and fastened back on itself with two clamps or clips, or fastened around a spoke with clamps; or better, fastened by a
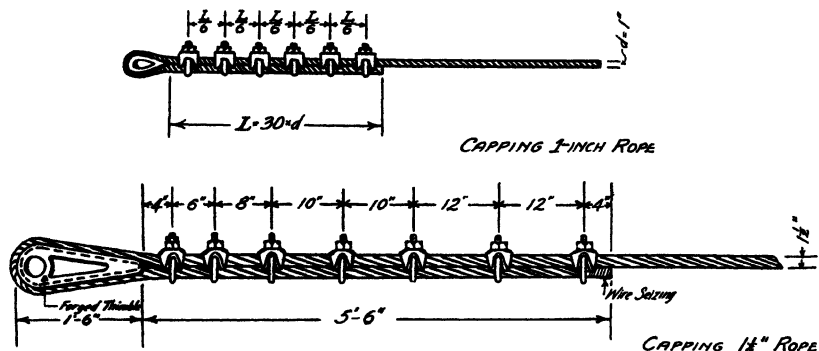

FIG. 1.—CORRECT METHOD OF CAPPING WIRE ROPE.

clamp cast in the spindle of the drum. At least two full wraps, and better three, should remain on the drum when the cage is at the bottom of the shaft. The terminal inside the drum should be fastened securely and inspected frequently.

Fastening at the cage or skip is, in most cases, done by means of clips or clamps. Reports show that these are placed at varying intervals, the U-bolt of the clamp usually being placed on the loose end of the rope and the cast portion of the clamp on the solid portion of the rope. The portion of the rope covered by clamps, or a length of 6 ft. to 10 ft. or 20 ft. is cut off every three or every six months, or whenever inspection shows that it is necessary. In a few cases, half the clamps are renewed every six months.

Fastening to the cage by clamps seems to give the best opportunity for inspection, and while sockets are considered and reported generally as satisfactory, the majority of operators prefer the clamps. These should

be fastened as shown in Fig. 1, the rope being passed around a large thimble, and further secured by means of locknuts. Careful inspection should be made of the rope when clips are attached to see that it is free from any broken wires and that the rope is thoroughly clean when the clips are attached. Nuts and locknuts should be examined every day on the regular inspection of the ropes. Lengths of at least 10 to 20 ft.
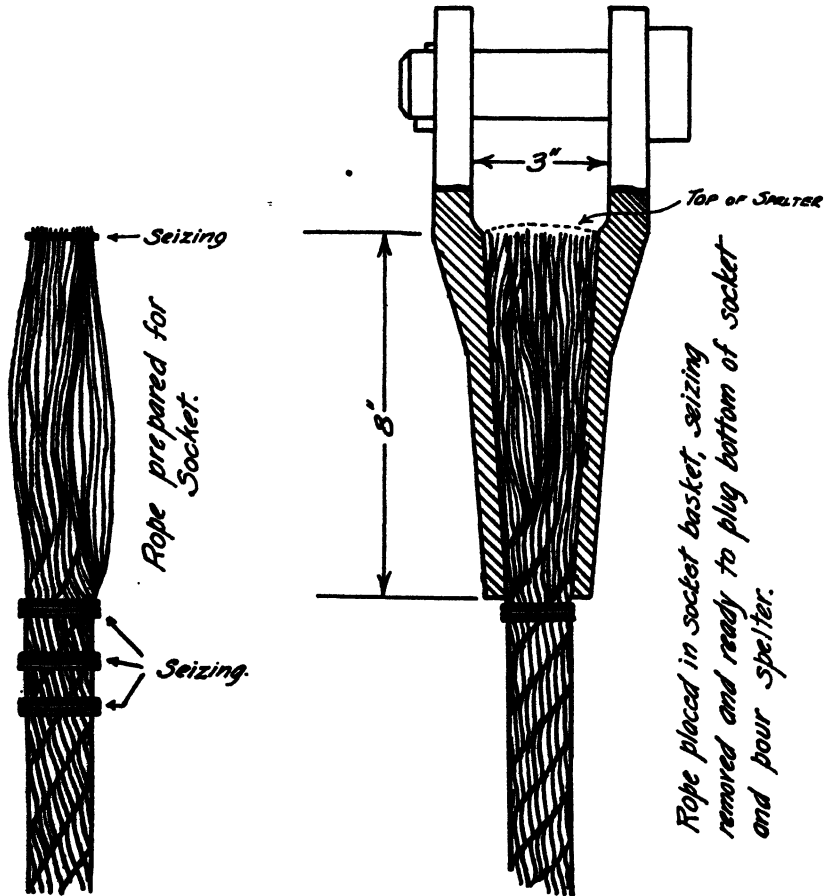

FIG. 2.—CORRECT METHOD OF SOCKETING WIRE ROPE, GREAT BOULDER PROPRIETARY GOLD MINES.

should be cut off every three months or six months, depending on hoisting conditions and the amount of ore hoisted. Strains will show on the portion of the rope attached to the thimble and the portion of the rope in advance of the clips. When rope is cut off, the portion above the clips should be periodically tested to find the tensile strength of the rope at that point; the size and diameter should be noted, also the condition of the

wires, and this diameter compared with the original diameter of the rope. Also, an examination of the interior of the rope and the condition of the hemp core should be made, as well as an examination for corrosion and brittleness of the wires, which is likely to be most severe in the lengths near the cage. The International Nickel Co. regularly sends a specimen of the rope cut off to a testing laboratory.

While two-thirds of the operators use clips, a number use sockets, and very careful attention is given to the socketing of the ropes. In most cases in which they are used, when the wires are properly secured, sockets are found quite satisfactory. The Sulphide Corpn. of New South Wales found that sockets did not give sufficient opportunity for inspection and that corrosion occurred although at other points the rope showed no indication of such a thing. The Hudson Coal Co. and the Great Boulder mine use a socket longer than that generally used, see Fig. 2. This permits the rope to extend into the socket a short distance in unaltered form, about 2 in. in the case of the Great Boulder mine and 5 in. in the practice of the Hudson Coal Co. This gives greater strength at the point of contact of rope and socket and less breaking action on the individual wires. The method of these companies is to separate the wires, cut out the core to the point where the rope remains intact, clean the wires with gasoline and diluted muriatic acid, wash them in a dilute solution of soda, insert them in the socket to the upper seizings on the rope, taking care to get them in an absolutely straight line. The Hudson Coal Co. and most of the operators simply loosen and broom out the wires. The Great Boulder finds that turning the wires nearly at right angles does not cause the wires to break, and has found this practice satisfactory in severe work for over twenty years. The socket is warmed to a temperature of 50° to 60° F. and pure spelter is poured in carefully and continuously until it fills the socket and covers the wires. Ropes newly capped, either by socket or by clips, should be run several trips with a light load before heavy loads or men are handled.

Where sockets are used for fastening, the greatest care should be taken in cleaning the wires and in setting them before the metal is poured into the socket, so that uniform stress will come on each wire. Sockets should be cut out every three or six months. Lengths of rope 6 to 10 ft. should also be cut off every three months or six months, depending on hoisting conditions, load, stress on the rope, as well as water and atmospheric conditions. Careful examination and tests of these pieces should be made, as mentioned under inspection of clips.

LIFE OF ROPE

· On this point considerable diversity of practice prevails. Several operators discard rope when the wires on the outside of the strands are

worn to less than 60 per cent. of the original area, and when inspection shows signs of corrosion. Others take more particular note of the number of broken wires. One company discards rope when the number of broken wires in any consecutive 10 ft. exceeds 10 per cent. of the total wires in the rope; another very careful company condemns the rope when twelve wires per foot are found, or if the wires be pitted and can be pried out of place easily, even if no broken wires are found; another discards rope when 60 to 70 wires are broken in one strand. One company finds that a decrease of $\frac{1}{8}$-in. (3.17 mm.) in diameter renders the rope unserviceable and cuts grooves in the sheaves; another company abandons the rope when a decrease of $\frac{3}{16}$ in. in diameter is shown or when seven wires are broken in any one strand. The Anaconda Copper Co. discards when not more than 20 per cent. of the strength of the rope is used up by abrasion and broken wires. The Sulphide Corpn. generally discards its 6 X 7 rope when one wire in any lay shows fractures; even if apparently in good condition, rope is discarded after two years' service. A few companies put a time limit on the use of the rope, even when characteristics of the rope are good. One iron company turns the rope end for end after six months and discards it after twelve months' service. The International Nickel Co., which takes special care of its ropes, abandons them after two years' use, even if a rope is apparently in good condition.

After the tensile strength of ropes of regular lay has been reduced 25 per cent. by alteration and wear, the limit of safety has usually been reached. With Lang-lay rope this reduction may be extended to 30 or 35 per cent. of the tensile strength. When wires of the ordinary lay on the outer side of any strand are worn to 60 per cent. of the original area, or when outer wires of the Lang lay are reduced to 50 per cent. of their original area, it is safe practice to abandon the rope. The number of broken wires permissible in any one lay should be very few. Some companies discard the rope when the number of breaks in any consecutive 10 ft. exceeds 10 per cent. of the total number of wires composing the rope, which seems as many as should be allowed. The proposed mining law concludes that if six broken wires occur in any lay, the rope should be condemned. With these differences in point of view and of practice, must be considered the reduction of safety factor ascertained by tests, the corrosion, if any, effect of the mine atmosphere, the work done by the rope, speed of hoisting, etc. A practice that will increase the life of the rope would be to change ropes from one drum to the other every six months, so that the rope having the underwind on the drum will be alternated with the rope having the overwind. Ropes should be turned end for end every six months or every year, depending on the life that is found possible and permissible. Where a rope is in constant active service and in heavy duty, attention to all details gives fully two years life, with a possibility of a longer time. Light or irregular duty may

permit use for a longer time. The wearing parts, the working conditions, the broken wires due to wear must, with careful inspection, determine the safe period for use.

LUBRICATION OF ROPE

Lubrication of rope varies from the use of any of the compounds manufactured by the oil companies for hoisting rope, to a combination of these compounds with oil, in the colder countries, and, in some cases crude oil, or alternate applications of oil and rope compound combined. A few companies lubricate with heavy black oil or crude oil, arranged to drip on at the sheaves, with special dressing of compound at intervals. The greater number prefer the lubricating compounds made especially for ropes. One mine that contains acid water applies thin compound, followed by heavier compound once a month. The International Nickel Co. uses the same lubricant applied every two weeks. In winter this is followed by black oil; the rope is passed through a hollow box and the heated lubricant is poured in. A number of companies prefer a special compound of the Texas oils for wet shafts, applied every two weeks; another company uses a lubricant every 60 days, with an additional drip above the sheaves, using waste oil from the compressors.

The Great Boulder mine uses castor oil, applied through a tray or cone bolted around the rope, with a gland or valve at the lower part of the cone packed with rubber. As salt water is encountered in the shaft, the rope must be carefully lubricated every three weeks. The use of oil in connection with lubrication, as practiced by several of the companies, is a wise method. Lubrication must not only be for the outer part of the rope but for the lubrication of the individual wires inside the strands. Heating the lubricant increases its power of penetration. In experiments made by The International Nickel Co., cold oil applied to a core did not penetrate to the hemp in the core at all, but a considerable amount of hot oil did reach the hemp.

Old lubricant on the rope should be cleaned off every three or six months by running the rope through steam in a trough and swabbing off the rope. Hot oil should then be applied and a good lubricant, free from acid, should be applied every two or three weeks. One of the main objects to keep in mind is to apply, first, heated oil which will penetrate to the inside of the strands and then the heavy outer lubricant, and both should be well rubbed in.

ROLLERS IN INCLINED SHAFTS

Practice is about evenly divided between the use of wooden and steel rollers. One large company operating with heavy loads in a shaft, with an inclination of 60°, uses lap-welded boiler tubing 12 in. long, 6 in.

in diameter, with spindles turning on bronze bearings in cast-steel frames. This gives entire satisfaction. The Great Boulder mine finds steel rollers 8 in. in diameter very satisfactory; also that disks of old belting 8 in. in diameter, clamped together, make very good rollers. The Ray mine, Arizona, Cranberry Creek coal mine, Witherbee-Sherman mine all use wooden rollers. The Copper Range Co. uses rollers made up of wrought-iron disks 32 in. in diameter, filled with maple blocks wedged in on the spindle, which is set in brass-bushed bearings. These work very satisfactorily. The Wharton Steel Co. uses steel rollers on roller bearings, which have worked very well. General South African practice favors steel rollers, which are preferred to any other, and these, with care, have given good service.

Acknowledgments are herewith made to the friends and officials who replied to the questionnaire; also to Hans Behr and to B. F. Tillson, for suggestions for good practice.

DISCUSSION

B. F. Tillson,* Franklin, N. J.—A roller that we think is superior to those mentioned in this paper is made of soft rubber similar to that used for the treads of automobile tires. Soft rubber will outwear hardwood blocks and give much less wear of the rope.

We once built too nice a bearing; the ball bearings were too free from friction. After the skip left, there was not sufficient friction in the bearing to bring the idler to a state of rest before the skip returned; therefore the tendency was for the idler to scour the rope before it ceased rotating in one direction and started to rotate in the other. We designed a bearing with greater friction which comes to rest an instant before the rope returns and does not have the inertia to overcome in the idler.

A question is always raised as to the factor of safety; of course we all realize that what we call a factor of safety is a factor of ignorance. Is it not our duty to define that factor of ignorance more closely? We buy a rope, specify a certain installation, and assume that it must have an initial factor of safety of a certain amount; when it gets down to another factor of safety, which may be five times the breaking strength of the rope for the load being carried, we take it off. There is no reason for that except that it is a factor of ignorance. If the rope is strong enough, it does not have to be five times as strong as is necessary. The point is, have we made the research necessary to limit that factor of ignorance? Should we not analyze our conditions and bring them into as narrow a limit as possible, and then determine the maximum percentage variation for each condition? ·

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Maximum hoisting speed is prescribed in many mining laws, with which I take issue, because they prescribe for something of which there is no definite knowledge. The speed at which we may operate in a shaft depends on the conditions of the haulageway and the equipment. I see no reason why the production of an operator who has put his equipment in proper shape should be limited by a law prescribing maximum hoisting speeds. More information should be obtained on the subject and the speeds should be prescribed by those who enforce the laws. Why penalize the people who have good equipment? What is the use of having first-class equipment if you cannot take advantage of it?

Tests on Samples of Plow-steel 1⅛-in. Hoisting Ropes of Six Strands Each, of Nineteen Wires, Seale Patent Lay*

	Right-hand Regular-lay Rope Drum End, American Steel & Wire Co.	Right-hand Lang-lay Roebling Rope, Load End
In service on ore skips from....	April 27, 1910, to Feb. 8, 1912, or 21⅓ months.	Feb. 8, 1912, to April 1, 1913, and still in service, or 13¾ months.
Total short tons hoisted, per rope....................	315,759	266,460
Foot-tons of work done on ore, per rope....................	292,110,560	259,333,284
Gross work done, in foot-tons, with load of 6.10 tons, skip of 2.75 tons and including work done on ropes lowering the empty skips.	Average length of 1,273 ft. of rope at 1.27 tons, 676,500,000.	Average length of 1,340 ft. of rope at 1.34 tons, 607,000,000.
Measured diameter of rope tested, inches...............	1⅞₄	1⅝₄
1. Full sample, no wires cut....	Three strands broke at tension of 109,300 lb.	Six strands broke at tension of 121,000 lb.
2. Full sample, no wires cut....	Six strands broke at tension of 111,200 lb.; average 110,250 lb.	Three and one-half strands broke at tension of 110,700 lb. Average 115,850 lb. Second sample broke where rope was bent about thimble in service.
3. One strand severed..........	Two strands broke at tension of 91,300 lb., so showed 82.8 per cent. of initial strength.	One strand and ten wires broke at tension of 89,300 lb., so showed 77.2 per cent. of initial strength.
4. Two strands severed........	Two strands broke at tension of 71,700 lb., so showed 65 per cent. of initial strength.	One strand and five wires broke at tension of 77,200 lb., so showed 66.7 per cent. of initial strength.
5. Twelve wires severed in one strand within a distance of 1 ft. and some additional wires nicked in the process.	Seventeen wires failed first and then three strands broke at tension of 103,000 lb., so showed 93.3 per cent. of initial strength.	Seven wires failed first and then six strands broke at tension of 104,500 lb., so showed 90.3 per cent. of initial strength.
6. Twenty wires severed within 1 ft., ten in line on successive strands on one side of rope and ten in line on opposite side with some additional wires nicked.	Nineteen wires failed first and then two and one-half strands broke at tension of 70,200 lb., so showed 63.7 per cent. of initial strength.	Five wires failed first and then six strands broke at tension of 93,900 lb., so showed 81.1 per cent. of initial strength.

* Test pieces 3 ft. long in the clear, made at plant of J. A. Roebling's Sons on April 11, 1913.

My impression is that much of our information in regard to the testing of ropes is based on the tests of new ropes. The results obtained by testing a rope that has been put into service (that has possibly had its tensile strength increased by the fact that work has been done on it, whose hemp center is compacted and whose wires are gathered into a more permanent location in respect to each other) will greatly influence our viewpoints as to when a rope is in a dangerous condition. The accompanying table gives a few figures of tests made for me some years ago.

Another factor is the torsion in the rope produced by the movement of a rope over a head sheave. There is bound to be a tendency for slight wear from pressure on the head sheave. The inertia of the sheave as the rope starts to pass through it naturally tends to twist the rope, just as if a screw is forced into a nut. For this reason it is essential that the rope should be properly attached to the cage, especially on ordinary-lay rope, because the torsion will increase as the load approaches the head sheave.

GRAHAM BRIGHT,* East Pittsburgh, Pa.—Are the factors of safety used in determining the size of cables used for hoisting based on the dead load, or is acceleration taken into account?

One of the tables shows an accelerating rate of 13 ft. per sec. per sec. As this particular hoist is, no doubt, a steam hoist, the pulsating effect of the engine will probably produce a maximum accelerating rate of about 15 ft. per sec. per sec. This is about one-half of the effect due to gravity, so that with this high rate of acceleration, 50 per cent. of the dead weight must be added to obtain the stress on the cable during the accelerating period. Where such high rates of acceleration are used, it is most important to take this into account in applying factors of safety. Such a high rate of acceleration is too great for safe operation of a mine hoist.

B. F. TILLSON.—It is common not to consider acceleration when basing the installation of ropes on factors of safety. That is one of the items that should be eliminated.

RUDOLPH KUDLICH,† Washington, D. C.—The acceleration stresses should be considered when figuring the factor of safety, but the bending stresses are much more important. The different mining regions have quite different customs regarding the size of hoist ropes. The Pennsylvania anthracite and bituminous, the West Virginia, and the Tennessee districts, as nearly as we have been able to find out, follow quite well the rope manufacturers' recommendations as to the relation of the diameter of the rope to the sheave. In Illinois, they use a large rope in proportion to the sheave diameter. The result is that the stress due to curvature of

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the larger ropes around the smaller sheaves is twice, or three times the weight of the net load and reduces the static dead load factor of safety from five to an actual factor of safety of three or four.

J. F. Howe,* Worcester, Mass.—The factors of bending are sometimes confused; one factor is the bending stress, another is loss of strength due to bending. It is possible to bend a rope over a very small sheave and lose only 10 per cent. of the strength; that would not be a good operating condition as would be speedily shown by the breakage of the wires, but the actual loss of strength in the rope is not as great as the bending stress would indicate. The bending stress indicates the stress in the outer fiber that would be produced by a load equivalent to it. In other words, a bending stress of 5 tons on a load would be the same stress in the outer fiber as would be caused by a direct load of 5 tons; that does not reduce the strength of the rope by 5 tons, but puts a stress in the outer fiber equivalent to that.

Robert Peele,† New York, N. Y.—A stress sometimes lost sight of is the starting stress. In almost all cases where chairs are used on starting the load, there will be a few inches of slack rope; dynamometer tests have shown that the additional load on the rope is very great. It is good practice to use a larger factor of safety for a shallow shaft than for a deep shaft, because in a shallow shaft the rope has less total resilience. In a deep shaft, with a long rope, there may be sufficent elasticity to take care of the starting stress. The allowance for starting stress may be reduced therefore as the shaft deepens, and it might disappear entirely in a shaft several thousand feet deep. Once, in a Quincy shaft, when I was on a man car, just starting up from the 54th level, a signal was suddenly given to stop the car at a level 200 or 300 ft. above the starting point. The engineer instantly shut off steam and applied the brakes, perhaps imagining that an accident was impending. The car, with four or five men on it, acted in much the same way as a bird cage on the end of a spring; it had a maximum up and down motion of perhaps 20 or 30 ft. before it gradually came to rest. This illustrates the amount of resilience in long rope. Table 1 is well founded, because the minimum factor of safety may properly decrease with the depth of the shaft.

The result of bending back the wires in making hoisting-rope sockets, depends partly on the kind of steel in the rope. Sharp bending of the wires is more injurious when the steel is of high tensile strength (high temper), than when the rope is made of mild steel. Also, heating the wires, preparatory to bending them, must be carefully controlled to avoid drawing their temper.

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† Professor of Mining Engineering, Columbia University.

At high hoisting speeds, with large drums and short periods of acceleration, the inertia of a heavy head sheave may cause considerable frictional wear on the rope. On suddenly checking such an engine, preparatory to landing the cage, a 14-ft. sheave, weighing say 3 tons and running at high velocity, may chafe the rope seriously before the sheave comes to rest.

B. F. TILLSON.—The following outline has been proposed for any investigation of steel hoisting ropes for mines:

1. Relative advantages of Lang-lay and ordinary-lay ropes for various services:
    (a) Proper methods of attachment to load, swivelled or unswivelled connection.
    (b) Rope sockets or clamps and methods of applying, number required and spacing to break rope before failure of attachment.
2. Corresponding reduction in "Factors of Safety" due to:
    (a) Severance of wires in one or several strands and distribution per running foot of rope.
    (b) Reduction of area of metallic cross section of rope because of: (1) wear, (2) corrosion.
    (c) Changes in initial structure of metal from: (1) Bending stresses—static, kinetic and reversals, (2) repeated shocks—either tensile or beating from overlap on hoisting drums, (3) overloading and stretch, (4) heat treatment involved in cleaning ropes by steam, (5) electrolysis due to acid waters, alkaline waters, and stray electric currents, (6) hydrogen embrittlement, (7) torsion due to fixed attachment of ends and resistance of sheave to rotation.
    (d) Life of hoisting ropes as measured by: (1) Foot-tons of work performed, (2) appearance of ropes externally and internally, (3) stretch of the rope (permanent elongation), (4) length of time in service and number of trips hoisted.
3. Wire-rope specifications and methods of testing.
4. Proper factors of safety as influenced by:
    (a) Hoisting of men or ore.
    (b) Maximum acceleration.
    (c) Size of head sheaves, hoist drums, and angle of lead.
    (d) Shaft equipment.
5. Limiting hoisting speeds for men and ore, respectively.
6. Care of hoisting ropes:
    (a) Inspection.
    (b) Cleaning and lubrication.

RUDOLF KUDLICH (written discussion).—Professor Raymond's discussion of the replies received in answer to this questionnaire on the use of hoisting ropes brings out many points which, if generally followed by mining companies, will lead to greater safety in mine hoisting; but these replies were received chiefly from metal-mining companies, and refer to comparatively deep shafts. A survey of hoisting practice in coal mines, made by the Bureau of Mines, showed that in various districts quite different conditions existed, and different practices were followed, in part due to natural physical conditions and in part to choice of equipment.

In the northern anthracite district of Pennsylvania, where the depth of shafts vary from a few hundred to almost 2000 ft., it has become the custom to use rather heavy engines with large drums and sheaves. Shafts are usually comparatively dry, so that the ropes suffer little from corrosion if protected by a suitable lubricant. Cars are loaded by hand, are rather low, and need not be of heavy construction, allowing a light cage construction and comparatively large factors of safety on ropes. The maximum speed of hoisting at many of these shafts is high, ranging up to almost 4000 ft. per min., though the rate of acceleration is low on account of the heavy engines and drums. Ropes are used on shafts until they show slight wear, when they are used on slopes or rope haulageways until worn out.

In the southern anthracite district, the depths of shafts are comparable with those of the northern districts. Coal is loaded from chutes, necessitating heavier car construction and allowing larger cars. Sizes of drums and sheaves for coal hoists are about the same as in the northern region, and the same size ropes are used, working at a lower factor of safety. Shafts are often wet and the water quite acid, making it difficult, if not impossible, to prevent internal corrosion which occasionally results in ropes breaking in service. The average life of ropes in this region is about two years.

The large-tonnage mines of the Illinois and Indiana bituminous field present quite different conditions. Here the shafts rarely exceed 500 ft., and are dry. Engines with small drums and sheaves are used to handle comparatively heavy loads that require ropes too large for the sheave diameter. The small light drums allow rapid acceleration but cause additional stresses which, with the excessive bending stresses, reduce the apparent factor of safety of 5 to 6, as calculated on static loads, to an actual factor of safety of 3 to 4. Though the maximum speed is less than that attained in the northern anthracite mines, rates of acceleration and retardation at the beginning and end of the hoist are so much greater that the average speed of hoisting is often greater. This all tends toward rapid deterioration of the ropes, which average about nine months of use, and often wear out in three or four months. In practically all shafts six-strand, nineteen-wire rope is used in preference to six-strand seven-wire rope, though at some mines flattened strand rope is preferred. Crucible steel is used in most cases, plow steel, as a rule, being used when the load has been increased and it is not possible to increase the size of the rope on account of drums, sheaves, etc. Lang-lay rope is generally preferred, and is sometimes used even where the rope is to be used on a slope after being taken from the shaft.

It is the almost universal practice to have only one layer of rope on the drum. In most of the older hoists wood-lagged cylindrical drums are used; on the newer ones, straight, coned, or profiled cast-iron drums.

Drum diameters vary from 5 to 14 ft. and sheaves from 6 to 14 ft.   Sheaves usually are unlined; some have the throat chilled, others are of soft cast iron.   Because of the shallow depths, it is possible to keep the angle of fleet low, in most cases less than 1½ per cent.

There is the same lack of uniformity in rules for determining the end of the life of a rope in coal and metal mining.   In the northern anthracite district, where the ropes discarded from shafts can be worn out on slopes, condemning a rope for shaft use does not mean a total loss of the rope; hence ropes are condemned for very slight signs of wear, say from one to half a dozen broken wires in a thousand feet of rope; or because the rope has stopped stretching.   Several of the companies take the ropes off shafts after two years' service, even though they show none of these signs of wear.   On slopes, ropes are condemned for signs of external wear, internal wear, as shown by looseness of the wires, reduction of diameter of rope, or broken wires.   No hard and fast rules are set.   The inspector follows his judgment.

In the other two fields mentioned, ropes are worn out on the shaft, and more definite indications of the life of the rope would seem necessary. Where corrosion is the deciding factor, the general appearance of the rope is indicative of external corrosion, while looseness of wires, reduction of diameter and examination of the portion cut off when resocketing, indicate the amount of internal corrosion.   Seldom, if ever, are ropes opened for internal examination.   Where the ropes fail from bending strains and wear, the number of broken wires in a given length of the rope are considered.   Definitions of dangerous conditions vary; for instance, 10 wires in 10 ft., two adjoining wires of one strand, six broken wires in one length of lay, or a rapid increase in the number of broken wires in the entire rope.

Use of Wire Rope in Mining Operations

BY JAMES F. HOWE,* M. E., WORCESTER, MASS.

EVERY engineer and user of wire rope is desirous of information that will enable him to determine whether the performance of any particular piece of rope is satisfactory, and what conditions can be changed so as to increase the life of a rope. In most cases, it is far from easy to analyze the conditions of service and tell which is the greatest factor in determining the life of wire rope. If we start on a layout for an entirely new rope system, the task of making conditions satisfactory is simplified so far as the engineering details are concerned; but if the plant requiring wire rope is in operation, any change is likely to involve the expenditure of considerable money to replace and change over machinery that must be kept in constant operation, because a large monetary loss will result from a shutdown for even a few days. Often wire rope is kept operating under bad conditions, as the user thinks that it is cheaper and quicker to replace the rope than to make alterations in the hoisting machinery.

A typical case is a metal mine operating in what is now almost the center of a large mining city, where it owns a few acres of mineral rights. It was expected that the shaft would go to a depth of 1000 ft. (305 m.), so a moderate-sized rope, about 1¼ in. (31.7 mm.), was used on a 6-ft. drum with a 7-ft. head-frame sheave and rather short lead from drum to head sheave. This mine has gone deeper each year until it is now 2500 ft. deep and the rope speed has been increased from 1500 ft. per min. to over 3000 ft. The original rope just filled the drum in one layer, now it overwinds three layers. The original ropes were probably 6 × 19 crucible steel and lasted from 12 to 18 mo.; now, the highest grade plow steel is used, yet the average service is only about 6 mo. The cause is not hard to find. If the engineers had known at the outset what would be done with this hoist, the design would have been very different. Conditions at this mine, from a wire-rope standpoint, have gradually become worse and it is difficult to see how rope service on this hoist can be compared with the service of previous years with any fairness to the wire-

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rope maker.   Excessive speed, overwinding, and heavy loads combine to
wear out this wire rope.

SAFETY

The first matter that should receive consideration in any rope problem
is the question of safety.   The slogan "safety first" has become a house-
hold phrase, but we use these words so commonly that their original
significance has been lost.   Most rope installations are readily made
"safe first" by taking into account the known forces acting upon them
and then applying a suitable safety factor.   From the moment a wire
rope is installed, however, most of the forces that act upon it tend to
produce a gradual deterioration,which ultimately brings the rope to the
point where it must be taken off and a new one substituted.   What these
various causes are and what effect they have on rope service will be
discussed later.

REGULAR-LAY AND LANG-LAY ROPE

Both of these forms of rope construction are old and both have their
advocates.   In Great Britain, Continental Europe, and South Africa,
Lang-lay ropes constitute probably 80 per cent. of all used in mining.
In the United States, probably 90 per cent. of all ropes in use are of
regular lay.   To find the reason for this difference in practice, it will be
necessary to study the question very carefully.   Lang lay was one of the
earliest forms of rope manufacture practiced abroad.   As a result, the
wire-rope users became familiar with the characteristics of this lay and
learned early how to handle it successfully.

In a Lang-lay rope, the wires in the strands and the strands in the rope
are twisted in the same direction, giving the rope the characteristic
appearance of being loosely twisted.   Each individual wire appears on
the surface of the strand for a longer distance than in the case of regular-
lay rope.   The worn spots on any wire are longer and such ropes appear
to wear down farther than on a regular-lay rope.   In a regular-lay rope,
the wires in the strands have a left-hand lay and the strands in the rope
have a right-hand lay; or in other words, the lay of the strands opposes
the lay of the rope.   The rope has the appearance of being more closely
twisted or laid together.   A worn regular-lay rope exposes more worn
crowns per foot but the worn spots are shorter.

If a Lang-lay rope is twisted in a direction to unlay the rope, the wires
in the strands will untwist at about one-third the rate of the untwisting
that occurs in the rope proper, assuming the same angle of lay for each.
As a result of such untwisting, the outside wires of the strands are
loosened, the amount depending on the amount of untwisting.   Excessive
stretching of a Lang-lay rope will produce about the same effect.   The
only opposition, therefore, to the continued untwisting of a Lang-lay
rope is the stiffness of the individual wires to the torsional stress tending

to untwist them.   If a Lang-lay rope is twisted in a direction to shorten
the lay of the rope, there is at the same time a shortening of the lay of
the wires in the strands equal to about one-third of the shortening of rope
lay.   The result of this twisting of the lay tighter is to produce a double
shortening of the rope; consequently, a powerful force is exerted against a
shortening of the lay compared to a weak force opposing the untwisting
action.   For this reason, a swivel on a Lang-lay rope will often cause
undue untwisting.

A force tending to untwist the regular-lay rope meets violent opposi-
tion because the wires in the strands tend to twist together by an amount
equal to about one-third of the twisting of the rope, instead of untwisting
as in the case of the Lang lay.   When the operation is reversed and a
regular-lay rope is twisted together, the wires in the strands tend to
loosen up.   In all of these illustrations, the forces are assumed to be
acting on finished ropes in service.

In as much as most wire ropes in service tend to lengthen in lay, or
stretch, it is obvious that it is the behavior of Lang lay and regular lay
ropes to the untwisting forces that is of most importance to rope users,
as this is the action that occurs most frequently in actual service.   The
question, therefore, as to the adaptability of Lang lay or regular lay
to any rope installation must be settled first on whether both ends of the
rope are fast at all times.   If they are, no harmful untwisting should
occur that would deform the Lang-lay rope.

The next point to be considered is the question of splicing.   In the
case of a shaft rope, splicing must never be used, for safety reasons; but
on mine haulages, splicing occurs frequently as worn sections are replaced
by new ones.   The lack of familiarity with the splicing of Lang-lay rope
is one of the drawbacks to its more extensive use in the United States.
Regular-lay ropes are more easily spliced by the men employed at the
mines to do this work; and more often than for any other reason, rope
users go back to regular-lay rope after trying one or two Lang-lay ropes
because they cannot make a satisfactory splice.   The principal trouble
is that the tucks pull out when put in by the ordinary method.   In every
case, a longer tuck is necessary than with a regular-lay rope.   The longer
the lay of the rope, the more difficult it is to make the splice hold.   It is
always advisable to wrap the ends of the strands to be tucked with tarred
marlin to increase the size enough so that the rope strands will press
firmly against the tucked strand; sometimes the strands may be crossed
over to good advantage.   Here and there we find a man who can suc-
cessfully splice Lang-lay rope, so that it is possible for all splicers to do
equally well by the exercise of a little ingenuity.

The next point to be decided is whether the wear on the rope will·be
improved with Lang-lay, as compared with regular-lay rope.   This is not
always susceptible to analysis before actual trial, but certain conditions

are more favorable to Lang-lay rope than others.   For example, if a wire rope drags or rubs, as frequently occurs on an incline or haulage, then in general Lang-lay rope will give the best results.   Also, in cases of severe overwinding on hoisting drums, the Lang-lay rope will frequently give the best service.   One fact, however, must not be overlooked: foreign rope practice on Lang-lay rope application is to use the largest possible diameter of sheaves and drums, which of course has a decided effect on the whole problem.   The best mining practice in the United States will compare favorably with any used abroad, but there are cases where our practice is not as good.   It should be remembered that Lang-lay rope, to give the best results, must have the best attention.

A new Lang-lay rope, composed of the same number, size, and strength of wires, is from 4 to 8 per cent. stronger on an actual breaking test than a regular-lay rope.   This extra strength is no doubt one reason for the large use of Lang-lay rope abroad, where laws regarding safety factors are usually under direct government supervision and actual proof test is required, at regular intervals, to determine whether a rope should be replaced or continue in service.

STRENGTH OF ROPE

The approximate strengths of ropes of different constructions and strengths of steel are standardized in the catalogs of the various American rope manufacturers.   There has been a tendency, in some cases, to take these strength figures as a minimum value, which is not the intent of the manufacturers.   Successive tests of contiguous pieces of wire rope cut from the same piece will show a variation from maximum to minimum. This difference on large ropes ($1\frac{1}{8}$ in. or over) might run from 3 to 5 per cent. and on smaller ropes, $\frac{1}{4}$ in., it might reach 8 per cent.; the average is about half way between.

A definition of the word approximate would seem to be necessary; the best engineering practice would indicate that approximate could well be defined as covering a variation up to but not exceeding 5 per cent. under the strength value so given.   In support of this definition, it need only be noted that when testing a newly made rope, the full value of its strength is never obtained.   After a few days' service under normal load, the strands embed themselves into the hemp center in the manner in which they are to work, and a small amount of stretch, about 1 per cent. in the first 30 days, will usually occur in a mine rope or other rope under similar load conditions of operation.   A test of such a rope after, say, 30 days of operation will invariably show a higher value than the original test, which may amount to nearly 10 per cent.   Some ropes that have been condemned have shown, on actual test, values in excess of their original tests, even with several broken wires in the section under test, substantiating the fact that the strength of a rope increases somewhat after the rope

is put into service. A similar condition would doubtless occur if a specimen of rope were put into a testing machine and allowed to stay for several days under, say, 30-per cent. load until the stretch had been taken out of it. For refined calculations of rope strengths, a strength 5 per cent. less than catalog approximate strengths is conservative and makes little difference in the load that a rope will safely carry.

ATTACHMENT OF ROPE TO DRUM

The most common method of attaching a wire rope to a drum is to pass the end of the rope through a hole in the drum and either clamp it to the drum with U-bolts or clamps, or to pass the rope around the drum shaft inside the drum and fasten the end back on itself. At least three complete wraps are usually left on the drum when the cage is down to the lowest level of the lift. This method seldom gives trouble, but in a few cases there has been a slight creeping of the rope at the point where the rope goes through the drum as the load comes off with the cage resting on the bottom, causing wear at this point of the drum.

A better method, and one requiring less rope, would be to cast a zinc cone on the end of the rope after it has passed through the drum and have the drum recessed so as to receive the end of the rope thus treated; then no slippage can occur. Or a special wire-rope open socket can be attached to the rope after it has been pulled through the hole in the drum and fastened to a lug on the inside of the drum by means of the socket pin.

ATTACHMENT OF ROPE TO CAGE OR SKIP

For fastening the end of the wire rope to the cage or skip, different methods are used; and in as much as the cage end is most liable to give way in case of accident, it is most important that this end have the most efficient method of fastening possible. In the case of the drum end, the friction of three wraps around the drum assists in making the drum-end safe even if clamps are used. It is on the skip or cage end where the greatest improvement can be made in attachments. The ideal cage fastening should develop the full strength of the rope. Of what value is a high-grade rope designed to give a safety factor of 6 if the end fastening or attachment will give only 66 per cent. efficiency? But how can this point be ascertained, some one may ask. Actual testing is the only way by which any method may be checked up, and the man making the fastening should also be checked. Unless this is done, it is impossible to tell whether the work of making the fastening has been properly done or not. In making many of the fastenings, a mine operator is dependent entirely on the skill or judgment of the man doing this work; and while he may be perfectly conscientious in his work, he may overlook important factors that may render the fastening weak or inefficient. In all probability, the managers and engineers in charge of mines would be

surprised, and in some cases alarmed, if they knew the actual efficiency of their rope fastenings compared with what it is possible to secure by improved methods.

The various methods of making end fastenings on cages and skips may be classified as follows: Wire-rope clamps for forming small loop with thimble, wire rope clips for forming small loop with thimble, eye spliced in rope for forming small loop with thimble, and socket attachment.

Wire-rope Clamps for End Fastenings

The first method involves the use of one or more clamps, usually made in halves with a series of bolts either in the center or in the center and on the edges of the clamp. The halves of the clamp are usually castings, preferably of steel with recesses or grooves spirally along the gripping surfaces, and when drawn up to the tightest position, there is from $\frac{1}{8}$ to $\frac{1}{4}$ in. (3.17 to 6.35 mm.) between them. A wire-rope thimble is usually inserted in the small loop formed below the clamp to protect the rope from abrasion from the pin on the cage or skip. The bolts of the clamp are drawn up tight with no load upon the rope, using as large a wrench as is practicable for the bolt being tightened. This clamp does not appreciably distort the rope being clamped. The power of such a clamp to hold is dependent on the following factors:

l = length of clamp, in inches;
D = diameter of rope (rope goes through clamp twice), in inches;
n = number of bolts;
d = diameter of bolts, in inches;
f = coefficient of friction of surfaces in contact;
t = tension on each bolt when tightened (half this is exerted on each rope in clamp), in pounds;
S = strength of rope or required strength of clamps, in pounds;
N = number of clamps required, if more than one is used;
$\dfrac{nt}{2Dl}$ = pressure per square inch of clamping surface (projected area)

Then
$$S = \frac{N}{2}\left(2Dl \times \frac{nt}{2Dl}\right)f = \frac{N}{2}\,(ntf)$$

As the strength S of a rope varies as the square of the diameter, a 2-in. rope will require a clamping surface four times as large as a 1-in. rope, assuming that the coefficient of friction is the same on a large rope as on a small one. One factor that must be considered is the maintaining of the tension on the bolts of the clamp. A wire rope will yield a little after it has been compressed for a time and will cause a slackening of the bolt tension. Furthermore, under tension, there is a reduction in the diameter of the rope, which reduction is increased as the load increases. It is therefore necessary to maintain tension on the bolts. It is doubtful

if large bolts set up with very high tension would be safe unless tightened at frequent intervals during the first few days of operation. As high pressures per square inch are not advisable, it would mean making clamps about three times the size given in the formula to insure reasonable safety. Clamps give an efficiency of 80 to 85 per cent. under careful application, based on actual tests.

Wire-rope Clips

These consist of a forged-steel base with four prongs and a U-bolt or loop, with nuts on each end, for clamping down a wire rope. A steel thimble must be inserted in the loop to protect the rope from abrasion. Clips depend for their holding power on the distortion by the U-bolt of one of the two parts of rope being gripped. The proper method of attaching a clip is to have the loop bite on the short side of the rope forming the loop, so that the flat base of the clip will rest against the tension side of the rope. The number of clips required depends on the size of rope to be fastened. The number for various sizes of ropes is given in Table 1.

TABLE 1.—*Number of Wire-rope Clips for Various Ropes*

Diameter of Rope, Inches	Number of Clips Recommended	Size of U-bolt, Inches	Size of Wrench for Tightening, Inches	Minimum Spacing Between Clips, Inches
1/4	2	3/8	8	4
5/16	2	3/8	8	4
3/8	2	3/8	8	4
7/16	2	1/2	8	4
1/2	2	1/2	12	5
5/8	3	9/16	12	5
3/4	3	5/8	12	5
7/8	4	3/4	12	6
1	4	3/4	16	6
1 1/8	5	3/4	16	6
1 1/4	5	7/8	16	8
1 3/8	6	7/8	16	8
1 1/2	6	7/8	16	8
1 5/8	6	7/8	16	10
1 3/4	6	1	20	10
1 7/8	6	1	20	10
2	6	1	20	10
2 1/4	6	1 1/8	24	12
2 1/2	6	1 1/8	24	12
2 3/4	6	1 1/4	24	12
3	6	1 1/4	24	12

The larger a rope, the greater is the number of clips required, notwithstanding the increased size and power of the clip. If a single clip could be tightened sufficiently to hold, it would probably bite off the

rope on the short side, so that several clips should be used. The efficiency of clip fastenings is approximately 85 per cent.

Eye Spliced into Rope

Splicing an eye in the rope consists in tucking each strand of the short end of the rope for five times underneath the proper strand of the tension side of rope. This fastening is not dependent on clamps and holds equally well under all loads, but it is only about 90 per cent. efficient. On tests for ultimate strength, the strands nip each other and break in the splice. An expert splicer is required to make this fastening.

A modified form of eye splice, which the writer recently patented,[1] is capable of developing practically 100 per cent. of the strength of any rope so spliced. The end of the rope is split into halves, and these halves are crossed over and laid back on each other so that each side of the loop contains one-half the strands without any tucks. The laying of the strands back around each other is such that friction does most of the holding, only a light fastening being necessary to hold the strands after making the eye. Actual tests show this to be the most efficient form of eye splice developed.

Socket Attachment

For the sake of clearness, this will be considered under two subheads, the old method and the new.

Old Method.—Several modifications of this method have been used but, in general, it consists in bringing a wire rope through the small end of the basket of a socket. The rope is then seized a short distance from the end, the hemp center cut out, and the wires bent outward, one at a time, or inward toward the center. When all the wires have been bent over, the rope is drawn down into the basket of the socket and babbitt or lead poured around to fill the voids. Sometimes one or more tapered pins are inserted before pouring in the babbitt or lead. In a few cases, zinc is used, but in most cases the wires are not cleaned except that they are wiped with waste. If the wires are large and stiff, they are sometimes heated red hot with a torch to facilitate the bending. If the man doing the socketing is in a hurry to finish the work, instead of bending over single wires, a whole strand of nineteen wires is heated red hot and bent at one time. The idea of bending over the wires is to make them hold and the babbitt serves to fill up the spaces and conceal what is underneath.

When bent each of the wires resembles a fish hook (except for the barb), and acts in about the same manner. Under load, a fish hook will straighten out, the bend being the weak spot, and in quite a few cases

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[1] Full descriptions of method of making is given in U. S. Patent No. 1334244, assigned to American Steel & Wire Co.

ropes have pulled clear out of sockets, sometimes with disastrous results. When the torch is used, the danger is increased because an annealed rope wire has less than half of the original strength. In all cases, the stress on a rope socketed by this method is carried up to the point of bend, which is concealed beneath the lead or babbitt. There is no way in which the condition of the wires can be determined, but the number of cases of failure is sufficiently large to warrant discontinuing the method of bending over wires, use of torch, and babbitt or lead. The efficiency will vary from 40 up to 75 or 80 per cent. of the rope strength.

New Method.—This is the result of research work on the part of rope manufacturers seeking a more reliable method of socketing. This method is simple, cheap, and possesses the highest efficiency known to the rope art. It is therefore rated at 100 per cent. The directions for attaching sockets to wire rope are as follows:

The rope should be securely seized at the end before cutting off, and an additional seizing placed at a distance equal to the length of the basket of the socket away from the end. In the case of large ropes, this seizing should be several inches long and securely wrapped on with a special seizing iron; this is important in order that the lay of the rope should not become untwisted, otherwise the tension on the strands may not be equal when the socket is applied.

Take off the end seizing, and cut out the hemp center back to the other seizing and broom out the wires perfectly straight; *i.e.*; they should all be untwisted, not necessarily straightened.

The wires for the distance that they are to be inserted in the socket should be carefully cleaned in benzine, naphtha, or gasoline, and then dipped in a bath of commercial muriatic acid for 30 to 60 sec., or until the acid has thoroughly cleaned each wire.

Dip the wires in boiling hot water, to which has been added a small amount of soda to neutralize the acid, and insert the wires into the basket of the socket. Be sure that the socket lines with the axis of the rope.

Seal the base of the socket basket with putty, clay, or similar substance, and pour molten zinc into the basket of the socket until it is full. The zinc must not be too hot, or it will anneal the ends of the wires, particularly on the smaller ropes; about 800° to 875° should be sufficiently hot. When congealed, the socket can be plunged into cold water to cool off. If socketing is properly done, the wire rope, when tested, will break one or more strands between the sockets.

In order to have a check on the performance of the rope and be sure that no wires are slipping, the wires should project ½ in. above the zinc which is poured around them, thus allowing an opportunity for constant inspection of the safety of the fastening which is impossible with any other method of socketing.

The important points to be noted are to clean the wires thoroughly, to flux them with acid, and to use zinc, never babbitt or lead. Zinc adheres to the wires firmly; in fact, the wires are galvanized into a solid mass. This new method of socketing is fully described in Bulletin No. 75, U. S. Bureau of Mines, and is used exclusively by mines in South Africa for all their rope attachments.

Having provided for the best type of fastening, one other point should be noted.   The vibration of a rope in motion is free to pass and repass until it reaches the fastening, where it stops; the result is an early fatigue of the wires for a few feet near the attachment.   This may be guarded against by cutting off about 5 ft. (1.5 m.) near the cage end and resocketing or recapping at regular intervals.   Recommendations for this practice are also given in the bulletin just mentioned.   This distance may vary with conditions, but it is an easy matter to ascertain this by watching for broken wires at or near the socket; but after six months, a change of fastening is advisable regardless of whether broken wires show or not.   Vibration also stops when the rope reaches the head sheave and at the drum, both when the cage is at the surface and when it is on the bottom, so it is advisable to advance these points a few feet.

DETERIORATING INFLUENCES

As mention has been made of the necessity for safety first, it is necessary to determine what constitutes safety last, for a wire rope must be safe until it is removed, otherwise it is positively dangerous.   The question is how is the time for its removal to be determined; the answer is, by tests of the worst worn or broken portions of the rope condemned or taken off.   The South African mining law provides for a test, every six months, of the piece cut off when the rope is resocketted.   Whenever this test shows that the strength is below that required by law, the rope must be removed and a new rope put on.   In the United States, the state mine inspectors have the authority to order a rope taken off if in their opinion it is unsafe to run.   There is also a difference in requirements for ropes used for hoisting men.   In one of the largest metal mines in the United States, a workman inspects the man-hoist rope each day, and if he pronounces the rope unsafe, not a man goes into the mine until the rope has been replaced with a new one.   In almost all cases, the replacement of rope is done on a more or less arbitrary basis.   Very seldom is a piece of the rope tested, and the user is frequently groping around in the dark as to the exact facts.

Reference has already been made to the fact that rope strength increases for a time after a rope goes into service.   In one case, a 1½-in. 6 × 19 rope showed a higher strength on a section cut from the worst worn section than was secured from a piece cut from the end next to the drum, where there was no wear, and yet it was deemed necessary to take off the rope as unsafe.

The safety factor in a wire rope will be reduced by the following causes: (1) External wear; (2) breakage of individual wires; (3) corrosion by acid water, alkali water, electrolysis; (4) loss of elasticity; (5) fatigue in metal composing wires from regular or reverse bending, load variations

caused by shocks due to starting and stopping or overloading, torsion at or near attachments.

External Wear

External wear is the simplest phenomenon in connection with rope usage, and it occurs in all places to some degree in hoisting and haulage ropes. When a rope wears down smoothly without breakage of wires or corrosion, conditions are more or less ideal. It is possible for wires to wear down to nearly one-half their original diameter and yet for the rope to have more strength than the worn condition would indicate. It is an exceptional rope, however, in which the wires will wear down to one-half their original diameter. The worn spot is only on the crown of the strand, while the rest of the wire is its full section, for which reason the actual strength is greater than the value obtained by taking one-half the area. For example, if on a 6 X 19 rope, 72 wires are worn half through there will be 36 whole wires plus 12 inside wires, or 78 wires out of 114, or about 70 per cent. But if a section is taken at any given point at right angles to the rope, only two or three wires in each strand will show wear, or eighteen wires in all, resulting in a reduction of area of about 8 per cent., thus leaving the rope strength 92 per cent. of the original and not 70 per cent. if calculated in the other way. Actual test would show the values to be nearer 92 than 70 per cent., but here again it is well to have tests made to check up any rope proposition, because other factors are always present.

Breakage of Individual Wires

Breakage of individual wires occurs whenever the external wear has reached a point where the wire, no longer as ductile as originally made, will no longer stand the combined forces of bending and tension. Defective wire or poor construction may occasionally be indicated by broken wires. One wire that broke repeatedly in a rope strand would look as bad as if different wires were breaking, but the effect on the strength would be entirely different. One wire would reduce the strength of a 6 X 19 hoisting rope $\frac{1}{114}$ of the total strength, whereas several wires broken in close proximity to each other would reduce the strength directly in proportion to the number broken. A normal rope indicates the fact that it is worn out by a distribution of broken wires in a more or less even manner, always assuming equal wear throughout the portion of rope under consideration. If the broken wires are 5 ft. or 6 ft. (1.5 or 1.8 m.) apart, the resulting decrease in strength will be that of one wire only due to the binding action of the twist of the wires in a strand. Three or four wires broken in one spot are a far more dangerous indication than twice that number spaced 2 or 3 ft. apart, even if in the same strand. The whole question simmers down to the uniformity of the distribution of the broken wires as to whether failure of a rope is liable to result. The usual phe-

nomenon observed, however, is that where one wire breaks, those on either side are liable to become displaced and break, so that when wires begin to break a rope will bear close watching and inspection to be sure it does not become dangerous. Four or five broken wires in one place in a rope strand leaves the rope unsymmetrical, and in case of a shock or surge of load, that strand may give way and necessitate the immediate removal of the rope.

Corrosion

Rope corrosion is of frequent occurrence in mines where ropes become wet from mine water. Both acid and alkali water cause quick corrosion of a rope externally; and if the core becomes saturated with water internal corrosion will start, which may soon become dangerous. A rusty rope is stiff and the wires break quickly as they become cemented together with rust. The only remedy is to have ropes well lubricated when installed and apply the lubricant at frequent intervals, always being sure to dry off the rope so that lubricant will adhere firmly and keep the water from the core of hemp. Electrolysis goes hand in hand with acid and alkali corrosion. Accidents have been caused by running corroded ropes too long. The only sure way is to cut off a piece and examine the inside: no chances should be taken with a corroded rope, it is stiff, inelastic, and potentially dangerous.

Loss of Elasticity

Loss of elasticity is the result of corrosion, excessive stretch, etc., so that the resulting rope has not ability to withstand shocks like a new rope. An inelastic rope is therefore dangerous to keep in service.

Fatigue in Wires

Fatigue comes from the bending of a rope back and forth over sheaves and drums a large number of times. There are two kinds of bending, one known as regular and the other as reverse. In regular bending, a rope always bends in the same direction and straightens out. This does not have a very bad effect provided the diameter of sheave or drum is of reasonable size. Reverse bending consists of a bend in one direction followed by straightening of rope and in close proximity a bend in the reverse direction. Such bending greatly reduces the life of a rope by causing breakage of wires before much wear or service has been obtained. Bending causes fatigue more quickly than any other force that may be applied to a wire rope. It is of the utmost importance, therefore, to see that large sheaves and drums are used on all installations where long life is expected of a wire rope.

Load variations caused by shocks of starting and stopping, also heavy loads, assist in producing fatigue from tensile stresses; but they are not

as severe as bending stresses, because wire will stand repeated tensile stresses more readily than repeated bending stresses.

Torsional stresses at or near where ends of rope are fastened are another source of fatigue. Twist will frequently be crowded ahead by passing over sheaves, and in some cases a swivel will help to prevent the torsion becoming serious. In other cases on mine slopes and shafts, where swivels are not used, it is advisable, particularly in the case of a new rope, to unfasten the rope from the cage or skip occasionally, let any extra twist run out of the rope naturally, and then reattach the rope to the cage or skip. The tendency of a rope to twist is, of course, caused by the stretch, which is greatest during the first few days of operation and decreases until it becomes a small factor in the tendency to cause torsional stresses at or near the fastenings. If the overhead sheaves or turn sheaves around which a rope passes become indented with the impression of the lay of the rope, and a new rope is put on without first renewing the sheaves or turning out the groove, there may be a crowding back and forth of the lay of the new rope because the lay of the new rope will not quite match the lay of the rope it replaced. This crowding back and forth of rope lay may become dangerous because of the tendency of the rope to throw a kink if any slack is allowed to come into it for any reason. Should such crowding of the lay be noted, the rope system must be gone over and the cause of the trouble located and corrected. Excessive wear is also caused by running a new rope over sheaves already grooved with the impression of an old rope; this condition is often the most noticeable point in rope service.

LIFE OF HOISTING ROPE

The determination of the proper life for a hoisting rope and the expression of this factor in proper units of measurement are matters of vital importance to every rope user. This is particularly true in the case of mining and the operations closely associated therewith.

For the proper analysis of this problem, a consideration of the units of measurement of ultimate rope life, used in various places, is essential in order to determine which units, if any, come the nearest to expressing a fair and impartial measure of the value of any given wire rope. This is a subject toward which the attention of all rope users is being directed, as modern accounting systems tend more and more toward the unit system of cost expression. Items formerly grouped under large general headings are being divided into smaller groups and these in turn subdivided so as to show individual items in cost. Comparisons of all these items are being made by all large companies, and many of the smaller ones, in order to keep more closely in touch with the factors entering into them. This minute subdivision of cost items enables the foreman, superintendent, manager, and officials to see at a glance what items of cost are fluctuating

and the reason therefor is eagerly sought after in order to keep costs at the lowest possible level.

The following units of measurement are used in mining and allied interests for measuring rope service: (1) Tons hoisted or handled (weight unit), gross weight or net weight; (2) number of trips made (hoist unit); (3) days, months, or years service (time unit); (4) distance traveled, in miles (length unit); (5) composite factor—tons hoisted and length of service, No. 1 and 3 combined; (6) foot-tons work performed (work unit), gross weight or net weight.

Tons Hoisted or Handled

Most mines keep a record of the tons hoisted each day; so the number of tons handled by a pair of ropes on a balanced hoist is easily ascertained. In the case of a coal mine, for instance, the tonnage hoisted may consist of rock or slate plus coal or the record may include coal only and disregard the rock or slate. The amount of supplies, including timber, etc. lowered, constitutes a variable factor. The proper method, if this unit is to be used for measurement of rope life, would be to use the gross weight in tons handled, the ratio between the gross tonnage and the net tonnage being a measure of the efficiency of the operation; but considering it from a rope standpoint, the gross tonnage hoisted is the fairer method of measuring rope performance.

Number of Trips Made

This might be termed the hoist unit and, provided the hoists are all made from the same depth, the same weight is hoisted each time, and the same number of trips per hour, and the same number of hours per day, it might give a fairly reasonable measure of rope life. Such a unit of measurement would usually be applied to a hoist used exclusively for handling men or supplies, rather than a producing hoist. There is nothing to prevent the application of such a unit to a producing hoist, except that the term is less common and for that reason not as readily converted into other more common units for comparison.

Length of Service

In many places, the time unit has been used because it did not involve any compilation of tonnage records, simply keeping track of the date a rope is put into service and the date it is taken off. Subtracting the two dates gives the service in days, months, and years. This method does not take into account such variables as are certain to occur over a period of time due to various shutdowns and differences in the amounts handled during different periods of the rope life. It is not always possible to

make due allowance for these variables where memory is the only guide. On the other hand, the time factor has a distinct bearing on rope service. Certain large mining companies have found it advisable to prescribe a maximum length of time beyond which a hoisting rope shall not be kept in service, regardless of other factors. This rule has been laid down in order that corrosion or deterioration due to light operation or stoppage for any period of time may be properly taken care of. It is well known that a wire rope standing idle is liable to deteriorate owing to corrosion and hence be in a less efficient condition for subsequent operation. In cases of idleness, ropes should be dried, if they have been in contact with moisture, and well lubricated to prevent corrosion internally and externally. The time limit in the case of one company was 2 yr., but on account of the excellent grade of rope received and the lubrication and care taken of the ropes, the time was lengthened to 2½ yr. Previous to the adoption of the time-limit rule, some ropes were kept in service as long as 4 yr., but it was felt that the added care necessary on account of more frequent inspection did not justify the extra risk involved in keeping the ropes in service. This same company has now made changes at some of its mines so that ropes are worked more constantly and a larger daily tonnage hoisted, so that in some cases ropes are taken off in 13 to 16 mo. after having taken out as much tonnage, but in a shorter period of time. In the cases referred to, the time-unit rule never becomes effective as the tonnage hoisted and general rope condition are the governing factors.

One chief use of the time unit is that it enables a quick check to be made for rough comparison of any two ropes; and if there is a wide variation in service, it enables a mine superintendent to check up the tonnages handled and the number of trips made. It also serves as a rough guide as to when a new rope should be ordered provided a spare rope is not on hand.

Distance Traveled

The length unit, in the past few years, has come into prominence as it possesses certain desirable characteristics, more particularly in elevator service, but there is nothing to prevent its use or application to mining service where conditions are fairly uniform. The distance traveled by a wire rope is a measure of its life or performance, and in a way is supplementary to the time unit. The length unit is based on the premises that the distance a rope travels and the number of bends or sheaves passed over are a determining factor in the life of a wire rope, which is true depending on the influence of other well-known factors. In application, all that it is necessary to do is to attach suitable recording counters to sheave or drum and convert these revolutions into miles, or gear the counter to read miles directly.

Composite Factor

It will be readily agreed that time alone, regardless of work performed or vice versa, will not be a fair method of rope comparison; hence, the necessity of a composite factor. The application of a composite factor rests largely on the judgment of the person using it, but in cases where time of service varies greatly as well as the rate of mining operation, a proper value needs to be assigned to such variations; this value depends on local conditions.

Foot-tons of Work Performed

The preceding five units are an attempt to measure the work performed by a wire rope, but each has its limitations as to accuracy in measuring work done. In mechanics, it is customary to speak of foot-pounds of work done as a unit of measurement, and there is no reason why the same conception should not be applied to rope performance. On account of the large forces involved, it will be found more convenient to use the foot-ton as the work unit for measuring the work performed by a wire rope. This will properly be divided into two parts, one useful work and the other non-useful; the sum of the two will be the gross work performed. For making comparisons, the gross work performed would seem to be the fairest method, as it includes all work done whether hoisting cage or skip, loaded or empty.

The weight unit fails to take into account anything except net tons hoisted. In order to find the work unit, record must be kept of the number of trips, weight of cage, car, ore, coal, rock, supplies, and men hoisted, as well as distance hoisted.

In many cases, comparing the gross weight hoisted to useful load, it will be found that the rope is performing considerable work for which it gets no credit. It will also be found that the higher the percentage of the ratio of the useful load to the gross load hoisted, the more efficient is the hoist and the cheaper the cost per net ton hoisted. Considered in this light, many hoists could be materially improved in efficiency.

DISCUSSION

HORACE F. LUNT, Denver, Colo.—We have a rule in Colorado that hoisting ropes shall not be spliced where men are hoisted, but once in a while somebody uses a spliced rope for hoisting ore. Are there any definite data on that subject?

J. F. HOWE.—I do not know of any definite data, because the splicing is usually so long that it is difficult to make an actual test on spliced rope. For that reason, spliced ropes are confined mostly to haulage where there is a slight angle of pitch, where breakage would not occasion a serious

accident. I think that splicing is not allowed on the steeper slopes and shafts. I do not believe rope manufacturers would recommend splicing ropes under those conditions; that is, putting in an endless splice. I do not refer, of course, to a splice in the ends of a rope. A splice in the middle of a rope I would consider dangerous because it must bend and rebend so many times.

HORACE F. LUNT.—Two or three years ago, in a shaft, I think in British Columbia, a rope broke and dropped a cageload of men. It was afterwards discovered that this was due to internal corrosion. The rope was tested by an electromagnetic method. I do not know whether it would be practical to test a rope electrically without taking it out of the shaft, but if such a method could be developed, it would be a valuable means for discovering internal corrosion, which is likely to take place where there is acid water. I understand that these tests are quite delicate and would not be practical at an ordinary mine. Has this method ever been tried in a mine?

B. F. TILLSON,* Franklin, N. J.—It has been suggested that the excess length of rope be wound on a drum within the main drum; do not do it if there are more than two layers of rope on the drum. The incentive to do that is natural. When there are more than two layers, the rope will cross, bend, wind and unwind on the part that enters the drum and will wear at that particular point, so that the rope put inside is worthless.

Figuring the work on a rope has always been a puzzle to me. Does the rope do work when it is being lowered in the shaft? Should you include that in the gross load? How much work does a rope do when it is hanging idle in a shaft? How are you going to measure that? My plan is to figure the gross load on the trips hoisted and not to include the gross load of lowering.

Just before the war, the New Jersey Zinc Co. appropriated money for the investigation of devices for the magnetic testing of ropes and coöperated with the Bureau of Standards, which was doing some work on steel rails, in the development of a magnetic device for determining reductions in factor of safety of hoisting ropes. Because of the war, we were not able to secure results, but I hope that the matter will be taken up again. Doctor Burrows, who was then in charge of that work in the Bureau of Standards, has since developed a device that has been tested by the Otis Elevator Co.

* Mining Engineer, New Jersey Zinc Co.

Safety Devices for Mine Shafts

By Rudolf Kudlich,* E. E., Washington, D. C.

THE problem of eliminating the hazards of hoisting in mines has been with us since the industry passed its earliest stages, when coal and ore could be won from surface working and tunnels. At first, safety devices were introduced as means of decreasing the amount of labor necessary to produce the mineral, or to increase the amount that could be obtained from a certain opening. For example, crude ladders replaced winding footpaths or steps cut into the walls of the pit or shaft, decreasing the distance it was necessary to carry the product to the surface and eliminating, to some extent, the hazard introduced by the precarious footing afforded by such paths and steps. Later, instead of being carried from the pit on the backs of men, the product was hoisted in buckets or baskets on a rope passing over a pulley or around a windlass. Thus the capacity of the pit was considerably increased, and the hazard due to making many trips up and down the shaft was eliminated.

From such beginnings accident prevention in mine hoisting advanced with the importance of the industry, until at present hoisting compares favorably with other branches of the mining industry with reference to safe practices. As in the other branches, this progress has been accomplished along two general lines—by legislation, rules, and education, and by the installation of safety devices. Though due credit must be given the former, it is not so satisfactory as the latter. Legislation, to be effective, requires a policing and inspection system to guarantee enforcement, such as we cannot hope to inaugurate. Rules may be satisfactory so long as the employer controls the labor market, and dismissal or some other form of discipline is an effective means of insuring obedience to the rules; but where labor demand is in excess of the supply, or where labor organizations are in control, such measures are not effective. An educational campaign will draw attention to hazards and may awaken a desire to avoid accidents by exercising the proper care.

It is necessary, therefore, to place our reliance mainly on mechanical devices to prevent accidents, and to depend as little as possible on laws,

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rules or other means involving the human element. This does not mean that we hope to install mechanical devices that will absolutely protect the workman against himself, or that it is possible to perfect appliances that

TABLE 1.—*Number of Men Killed by Shaft Accidents in and about the Coal Mines of the United States, by States During Period Shown, Ending Dec. 31, 1913, for Which Continuous Records Are Available*

State	Period Ending 1913, Number of Years	Falling Down Shafts or Slopes	Objects Falling Down Shafts or Slopes	Cages or Skips	Other Causes	Total Killed in Shafts	Percentage of Total Killed	Number Killed per 1000 Employed
Alabama..................	21	5	2	1		8	0.43	0.02
Alaska and California.......	4					0		
Arkansas..................	9				2	2	1.87	0.05
Colorado.................	28	17	3	14	2	36	2.07	0.15
Georgia and North Carolina	5					0		
Idaho and Nevada..........	4					0		
Illinois...................	29	115	28	117	6	266	7.80	0.20
Indiana..................	19	31	2	36	2	71	10.46	0.23
Iowa.....................	26	35	6	22	2	65	9.00	0.20
Kansas...................	21	19	6	25		50	9.47	0.23
Kentucky.................	26	10	5	9		24	3.78	0.08
Maryland.................	23					0		
Michigan.................	14	2	2	6		10	10.20	0.22
Missouri.................	26	9	1	17		27	7.14	0.13
Montana.................	14			1		1	0.63	0.03
New Mexico..............	21					0		
North Dakota.............	6					0		
Ohio.....................	30	33	6	39	1	79	3.45	0.08
Oklahoma................	21	5	3	14	3	25	3.43	0.19
Oregon...................	5					0		
Pennsylvania, anthracite....	44	520	105	211	7	843	4.76	0.16
Pennsylvania, bituminous...	36	104	19	90	4	217	2.29	0.06
Tennessee................	23	1				1	0.12	
Texas....................	5				1	1	4.00	0.04
Utah....................	22	2		1		3	0.84	0.08
Virginia..................	5	1				1	0.40	0.03
Washington..............	25	6	3	2	2	13	2.03	0.12
West Virginia.............	29	43	2	22		67	1.41	0.07
Wyoming.................	6			1		1	0.41	0.02
Total...................		958	193	628	32	1811	3.74	0.12

* Albert H. Fay: Coal-mine Fatalities in the United States, 1870–1914. Bureau of Mines *Bulletin* 115.

will remain operative year after year without care; but better results can be expected from such devices, which will protect unless they are deliberately rendered inoperative, or are allowed to deteriorate to such an extent that they will not function.

To find where the greatest need of devices to prevent accidents exists, statistics should be studied to find how the accidents occur. Unfortunately, statistics covering accidents in metal mines are very incomplete, so that our assumptions must be based entirely on the number and causes of accidents occurring in coal mines. These assumptions will not be far wrong for metal mines, as the methods employed are not radically different. The principal hazards, falling down the shafts or being struck by objects falling down the shaft, or while riding on cages, exist in both cases.

Table 1, compiled by the Bureau of Mines from the various state mine inspectors' annual reports, shows the number of men killed by shaft accidents in coal mines in the United States for periods up to 44 years, over which continuous records are available, and ending Dec. 31, 1913. As this table shows only the fatal accidents and divides them only into four groups, and as it was well known that the large number of non-fatal accidents that occurred would indicate the lesser hazards, a compilation was made showing the number of fatal and non-fatal accidents in the four most important coal-mining states over a period of 10 years; this is shown in Table 2. These four states, Pennsylvania, West Virginia, Illinois, and Ohio, were chosen because they produced almost three quarters of the coal produced in the United States; therefore from the hazard standpoint they may be considered as truly representing the entire coal-mining industry; also because in the mine inspectors' annual reports of only those four states are accidents so fully described that they can be classified as to the cause of the accident and the responsibility for the accident fixed.

According to Table 2, the greatest number of accidents, 199, or almost 20 per cent. of the total, resulted from falling down the shaft. This immediately leads us to infer that the heads of shafts are not properly guarded, and that men working in the shaft do not properly protect themselves. The large number of accidents of this class is unfortunate, because these accidents are so easily prevented. Recent years have shown a decided improvement in the protection of shafts and a reduction of such accidents. Up to and during the period covered by these statistics, shafts practically unguarded were common. They were usually fenced at the sides, but the ends of the shaft were often open, or at best protected only by a chain, bar, or gate, which was supposed to be put in place when that shaft landing was not in use. But within the past few years, the properly protected shaft landing is becoming more common. By properly protected is meant that the shaft is entirely enclosed by adequate fences and gates, except when the cage is at the landing.

TABLE 2.—*Shaft Accidents in Coal Mines in Pennsylvania, West Virginia, Illinois, and Ohio, During Ten Years from 1904 to 1913, Inclusive. Data Obtained from State Mine Inspectors' Reports*

	Killed and Injured		Number Killed	Number Injured	Number Killed and Injured				Accidents	
	Number	Per Cent. of Total			Accident Due to Victim	Accident Due to Other Employees	Accident Due to Defective Equipment	Cause of Accident not Clear	Number	Per Cent. of Total
Falling down shaft............	199	19.9	164	35	58	3	17	121	192	24.0
Struck by objects falling down shaft......................	145	14.5	46	99	1	2	5	137	135	16.8
Overhoisting to top or bottom .	139	13.9	36	103		93	1	45	48	6.0
Caught under descending cage at foot of shaft.............	105	10.5	41	64	23	4		78	105	13.1
Falling from cage while riding .	101	10.1	84	17	30	12	2	57	87	10.9
Caught between cage and shaft timbers....................	78	7.8	59	19	45	7	1	25	76	9.5
Miscellaneous accidents while riding on cage..............	69	6.9	7	62	5	6	10	48	56	7.0
Shaft sinkers equipment.......	64	6.4	49	15	12	16	23	13	35	4.4
Miscellaneous accidents at foot of shaft and in sump........	37	3.7	24	13	12	11	1	13	33	4.1
Cage striking projecting landing keeps..................	21	2.1	5	16	1	5	1	14	9	1.1
Hoisting rope breaking........	20	2.0	19	1			20		4	0 5
Miscellaneous accidents at head of shaft..............	12	1.2	11	1	4	1	3	4	10	1.2
By hoisting machinery........	11	1.1	7	4	2		6	3	11	1.4
Total....................	1001	100.0	552	449	193	160	90	558	801	100.0
Per cent. of total..........			55.1	44.9	19.3	16.0	9.0	55.7		

To insure that the gates are opened only when the cage arrives at the landing and are closed as soon as it leaves the landing, it is absolutely necessary that their operation be entirely automatic. Experience has shown that men cannot be relied on to keep the gates closed; and in the case of landings where cars must be run off and on platform cages, the physical effort is too great for a man to open and close a heavy gate each time a car is caged.

Automatic gates have been developed to such a state that now a gate can be found to fulfill the requirements for each class of landings. For terminal landings, gates operated directly by the cage are suitable. At top landings a cross-bar extending across the shaft and connected to each gate is raised by the cage or a clamp on the hoisting rope; also brackets

or some similar device on the cage engage the gates and raise them. At bottom landings, a like arrangement is used, except that the downward motion of the cage raises the gate by means of a rope or chain passing over a pulley. At surface landings where coal is hoisted to an overhead dump or tipple, a more complicated operating mechanism is necessary. Several gates are available for this purpose, operated by air or steam pistons, the admission valve of which is controlled either by the cage or by the hoist. When the cage is being hoisted past the landing, the admission valve is opened and closed so rapidly that the motion of the piston is not sufficient to raise the gate; but if the cage is stopped at the landing, the valve is held open and the gate is raised. For intermediate landings, similar mechanism electrically operated and controlled by the cage has been successfully installed. Other gates opened directly by the cage, and under the control of the engineer, require no external power for their operation but require attention by the engineer; therefore, they are not so fool-proof as the air, steam, or electrically operated ones.

The gates should be placed as near the shaft as possible, so as not to interfere with caging; they should operate in a vertical rather than a horizontal plane both to reduce the space required and to eliminate the possibility of sweeping men into the shaft when closing. They should be at least 5 ft. (1.5 m.), preferably 6 ft. high, to prevent men leaning over them, and should be sufficiently strong to resist the ordinary rough usage, and so reenforced and braced that they will not be jarred apart. Springs or other cushions should be provided to reduce the jar as much as possible.

A small number of these accidents resulted from falls from some point in the shaft while work was being done on the timbers, guides, or piping. These are generally due to the use of improperly constructed staging, or because the men worked without staging. No device can be developed that will eliminate such accidents; they can be avoided only by the exercise of care on the part of the men who build and use the staging.

Of the 1001 accidents 145 were caused by objects falling down shafts. There are two methods of eliminating such accidents—preventing objects from falling and keeping men out of the danger zone. A few of the objects fall from the surface, where they are accidentally thrown or fall into the shaft; a few fall from timbers in the shaft, where they have lodged at some time; but the majority are pieces of coal or rock falling from the cars as they are being hoisted or dumped at the top. To prevent the latter, the more modern cage dumps are so designed that the car is carried some little distance from the shaft before the actual dumping period commences, so there is little likelihood of any coal falling back into the shaft; this is quite a contrast to the older cages, which dumped directly at the edge of the shaft, almost every car dumped being followed by a rain of coal down the shaft.

Foreign objects falling into the shaft at the surface can generally be

prevented by suitable fences and toe boards, but those falling from timbers can be eliminated only by unfailing care on the part of the shaft repair men and inspectors.

The best remedy is to keep the men out of the danger zone at shaft landings; this is quite possible by the use of automatic caging devices. These devices are of benefit in so many ways that it is hard to understand why they are not more generally installed. From the safety standpoint, the automatic device eliminates entirely the need of men working directly at the shaft when hoisting coal. The empty cars are automatically released from the cage, when it is landed, and levers set at a distance from the shaft control stop blocks which, when released, allow the loaded car to run by gravity on the cage, where it is held by self-operating blocks. From the operating standpoint, these devices usually reduce the number of men required at the landing; reduce the amount of work they are required to do, so that they are able to keep the same pace during the entire shift; and, except in rare instances, where a caging crew has worked together so long that the men coöperate like parts of a machine, the speed of caging can be considerably increased.

The third largest class of hoisting accidents are those caused by over-winding. These can also be prevented by mechanical devices, though more complicated apparatus is required than in either of the preceding cases. This apparatus has the advantage, however, of being well protected from the weather and from dirt, and is under the direct care of the hoisting engineer, a man more familiar with machinery and better able tomaintain it.

The earliest forms of apparatus to prevent overwinding, or overwinds as they are commonly called, were very simple; in fact, so simple that they did not prevent the trouble. One of the earliest devices consisted of a bell or gong rung by a trigger in the shaft so placed that the cage struck it just before or just after passing the landing. This bell was supposed to remind the engineer to shut off his throttle and apply his brake, if he had not already done so. Next, the trigger instead of ringing a bell released weights that closed the throttle, reversed the engine, and applied the brake; this was all done instantaneously after the cage had passed the landing, without any previous slowing down. Though this device prevented the ascending cage from being carried into the sheaves, provided there was ample overhoisting space, it did not prevent the descending cage from striking the bottom so hard as to injure the men riding. The stresses set up by suddenly reversing the engine and applying the brake when the engine was operating at full speed were so severe that several engines were wrecked by this device.

Next mechanisms were attached to the engine; these could be adjusted to suit conditions and, in general, were quite an improvement on the earlier types. They closed the throttle before the cage reached the

landing and brought the reverse lever gradually to the central point, thus slowing down the engine preparatory to the application of the brake when the cage was at the landing. There were two principal objections to this device. If set so that the hoist could land a loaded cage with full steam pressure, it could not do so if the pressure was low, as it did not have sufficient momentum when the throttle was closed to carry the loaded cage to the landing. When men were being hoisted, the speed of the descending cage would not be sufficiently reduced when it reached the bottom, and the impact of the cage on the keeps or landing timber was so great as to injure the men.

The most modern overwinds have overcome these objections. In these, the speed is controlled by a governor during the entire hoist, maximum speed being permitted until a certain point near the landing is reached, where, if the speed has not been sufficiently reduced the power is cut off. If at a farther point the speed is excessive, the reverse is centered, in the steam hoist, or the controller thrown into reverse, in the electric hoist, and when the cage reaches the landing the brake is applied. In this way the speed is properly governed during the entire hoisting period, but at no time is the engineer's control of the hoist interfered with, unless he has not kept it under control.

These overwinds are also, if desired, fitted with devices by which a low-speed limit is set when hoisting men, and the surface landing instead of the tipple is made the terminal point of the hoist. This attachment must be made operative by the engineer, but as a check on him a signal light at each shaft landing automatically indicates whether or not it has been set.

These devices are accurate, well built, and can be relied on to function properly, but one weak spot has developed in several installations. The overwind is usually a separate piece of mechanism driven from the drum shaft by chain and sprockets. Generally, ordinary malleable-iron Ewart chain is used, which stretches and wears until it becomes so loose that it skips a tooth or leaves the sprocket entirely. This is a simple mechanical fault that can be remedied by substituting gear and shaft drive, or a better type of chain.

The 105 accidents caused by men being caught under the descending cage occurred in four ways: Men were caught under the cage while clearing out the sump; while crossing the shaft instead of using the passageway around the shaft; by falling or slipping into the sump while caging cars; or by being pushed or falling while trying to get on the cage to ascend the shaft. Accidents of the first class can be prevented by probably as simple a safety device as we can contrive—a substantial bolt or bar slipped through a hole bored in a shaft guide a short distance above the entry level. If the cage is inadvertently lowered, it will come to rest when the guides or runners on the cage strike the bar, and the

men in the sump will be protected. The other accidents would have been prevented had the gates or caging devices described been in use.

Accidents listed as Falling from Cage While Riding and Caught Between Cage and Shaft Timbers could have been prevented by the same remedy—providing the cages with gates. This is one instance of a radical difference in practice in metal-mine and coal-mine hoisting. Cages for hoisting men in metal mines are commonly provided with gates, while those used in coal mines are only rarely so equipped. In metal mines, cages are usually very small, compared with those in coal mines, and a larger number of men are crowded on, making gates necessary to prevent the men being crowded off. The cages in coal mines have larger decks and the number of men carried at one time is in most states limited, by state laws, to ten or twelve, so that there is ample room without crowding, and hand holds are usually considered sufficient. That they are not is shown by the fact that 179 men were killed or injured by falling from the cage or by being caught between the cage and the shaft timbers. These accidents are generally the result of jostling or playing while riding on the cage, or of trying to get on or off the cage after the signal to hoist has been given; they show clearly that the men must be protected against themselves.

In a large number of mines, cages are equipped with bars or chains, which are sometimes placed across the ends of the cage when hoisting men, but more often they are not. When used, these bars or chains prevent men being pushed off the cage, but they do not prevent men trying to get on after the cage has been "rung away." Though it is a radical departure, and it is claimed that gates would interfere when hoisting coal, it is recommended that all cages used for handling men be fitted with gates, and that these gates be used whenever men are being carried.

The miscellaneous accidents that occurred while riding on the cage, as well as those that occurred at the foot and head of the shaft, were due to such a large variety of causes that it will not be possible to go into details as to how each could be avoided. Nevertheless, if a shaft be provided with the devices discussed here, the greater part of these accidents would have been avoided.

It may be questioned whether accidents occurring while sinking a shaft should be included in shaft accidents. This work is usually done by contract, and the methods employed are not usually under the control of the mine management; in many states, the work is not under the control of the mine inspector. I might say, however, that the principal offender is the crosshead, or billy. This is usually a simple wooden frame traveling between guides to steady the rope to prevent the bucket from swinging. As a rule, the crosshead simply rests on clamps on the rope, so that the rope is free to pass through the crosshead if the latter is

kept from descending. Occasionally, the crosshead sticks on, say, a lump of ice, or in a "tight place" in the guides, and its downward motion is arrested momentarily. After the rope and bucket have passed downward some distance, the crosshead may come loose, and falling may strike the rope clamp with sufficient force to break the rope or attachment. Several safety crossheads have been successfully used, which effectively prevent this accident. A catch is provided which automatically grips the rope and locks the crosshead and rope together, except when the former is resting on the stop-locks at the foot of the guides, and the grip on the rope is released.

It is only in anthracite mines where coal is hoisted from several levels, and in metal mines where ore is hoisted in cars, that landing fans or keeps are used, so that this class of accident is rare. The cause is carelessness on the part of the foot tenders, who neglect to withdraw the keeps when going from an upper to a lower level, or of the engineer, who forgets to which level he is lowering the cage. The remedy is a signal system that indicates on the hoisting engineer's depth indicator, by light or some other means, just which keeps are in or out, showing the engineer at all times how far he can lower the cage.

At least one such system has been in satisfactory operation in the shafts of one large anthracite company, wherein a movable mark on the indicator dial indicates the level being served at the time; red and green lights on the target indicate the position of the keeps at each landing, and the landing keeps and car blocks at each landing are interlocked, so that the keeps must be withdrawn to set the safety block protecting the shaft.

The average miner fears more the breaking of a hoisting rope than any other accident which may occur in a shaft. It is gratifying to note that these accidents are so few. Only four times in ten years in the states covered by this compilation did a rope break while men were being hoisted. There were, no doubt, many cases while coal was being hoisted, but we have no means of obtaining this information. The laws require only that accidents in which men were killed or injured be reported, and do not require records of ropes used, as do those of some other countries. We are naturally somewhat protected when hoisting men, because the static load on the rope is considerably less when hoisting men than when hoisting materials, and the stress due to acceleration is also less on account of the slower rate of speed at which men are hoisted.

The English miner says, "The best safety catch is a good rope," and lays his entire stress on hoisting-rope inspection and testing, often entirely omitting the safety catches from his cages. I would amend this motto to read, "The best preventative for falling cages is a good rope." Did we use such good ropes that there would be no danger of their breaking, and was there no danger of the cage accidentally striking an obstruc-

tion on the guides or in the shaft, there would be no need of safety catches; but as that is a goal we cannot attain, it is necessary to equip cages with these devices. But naturally the proper thing to do is to so care for our hoisting ropes that there is as little likelihood as possible of the safety catches being brought into play. It is better to discard a rope months before it is worn out than to use it one day too long and have it break in service. The cost of ropes per ton of material hoisted is so small, and the cost of a rope, compared with the cost of damages caused by a rope breaking, is so slight, that the cost of discarding a rope on the first sign of weakness rather than using it to the limit will amount to a very low cost of insurance.

The types of safety catches now in use have certain shortcomings, but if properly cared for they are of great value in preventing disastrous results if hoisting ropes break under certain conditions. Unfortunately, many mining men consider them of little value, install them on cages simply because they are required by the mining law, and then neglect to give them the proper attention. If properly adjusted, lubricated, and maintained, most of the modern safety catches will effectively prevent the cage from falling if the rope breaks between the cage and the head sheave while the cage is being hoisted. It is under these conditions that the rope usually fails. If, however, the rope should fail near the drum, with a considerable length of rope hanging over the sheave to retain the tension on the drawbar springs, or while the cage is descending, the probability is that the catches will not hold the cage, though they may so retard the velocity of the falling cage that the damage is minimized. But to render absolute protection under these conditions, some radical changes from the present mine-cage safety catches must be made. The elevator manufacturers have practically solved this problem, but these devices are more complicated than the mining man thinks necessary, one or more additional ropes in each shaft are required, and the entire operation depends on niceties of clearance, alignment, and working fits that are difficult to maintain under the severe conditions existing in mine shafts. The office-building type of passenger elevator with metal guides, speed governor, etc., has been installed as man hoists in mines, but in only a few cases.

Accidents caused by machinery, that is, by hoisting engines and sheaves, could have been prevented had the moving parts been protected with railings or guards such as are in common use in other industrial plants.

To sum up the entire situation, we can safely say that almost all the accidents commonly occurring in shaft hoisting can be avoided by the installation of suitable safety devices, and that with few exceptions such safety devices have been developed and can be used without interfering with the actual operation of hoisting in mine shafts.

Shot-firing by Electricity

By N. S. Greensfelder,* E. M., Wilmington, Del.

The firing of explosive charges by electricity dates back to 1745 when a Doctor Watson is said to have used an electric spark for igniting gunpowder. His method failed in practical application because, to ignite blasting powder, heat must be applied for a period longer than the duration of an electrostatic spark; Wheatstone found that this does not exceed $\frac{1}{24000}$ sec. The usual result, therefore, of an electrical discharge in gunpowder is to scatter the grains, without igniting them. Benjamin Franklin partly overcame this difficulty by confining the powder and crushing grains that lay in the line of discharge. In 1822, after the primary cell had been discovered by Volta, Robert Hare, professor of chemistry at the University of Philadelphia, noticed the heating of a short piece of wire placed in the circuit of a voltaic battery. Later work by Oersted, Faraday, Wheatstone, and Henley resulted in the development of the magneto blasting machines.

Fulminates of gold and silver are thought to have been discovered by Cornelius Drebbel, about 1600. Pepys, in his diary for November 11, 1663, recounts a conversation with Doctor Allen, who told him about aurium fulminans "of which a grain . . . put in a silver spoon and fired, will give a blow like a musquett and strike a hole through the silver spoon downward, without the least force upward."[1] This fallacious belief that explosives act downwards only is still held by many who should know better. Fulminate of mercury was discovered by Howard in 1799. J. Shaw of Philadelphia is said to have invented a fulminate-of-mercury cap in 1814; his idea was later adopted by gunmakers in England and France. The principal advantages of mercury fulminate for use in detonators are the simplicity of exploding it and the great shock developed by its explosion.

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* Mining Engineer, Hercules Powder Co.
[1] Marshall: "Explosives," 2d ed. 1, 36, 1917.

After the invention of the mercury-fulminate cap and the development of practical blasting machines, electric igniters and electric blasting caps were introduced and improved by manufacturers in different countries. In America and England, efforts of inventors were directed principally to detonators that were fired by heating produced by an electric current; in other countries systems of ignition by sparks were further developed. Two types of detonators, commonly called high-tension and low-tension, were thus evolved.

High-tension igniters, or detonators, are not used in this country and are practically obsolete. A spark passing between two metal poles ignites the priming material, such as a piece of guncotton or a composition of potassium chlorate and antimony sulfide. The priming substance in the low-tension type is ignited by the heating, to incandesence, of a bridge of fine wire, usually platinum, platinum-iridium, or cupro-nickel alloy, embedded in the priming substance. This is the type of electric blasting cap used in this country. Its resistance can be tested before it is introduced into a circuit; whereas the only way in which the high-tension type can be tested is by firing it.

"For various reasons mercury fulminate seems destined to hold its place as the essential ingredient in high-explosive primers. Mixtures of fulminates with potassium chlorate are generally used for charging detonators. The admixture of potassium chlorate not only reduces the quantity of the more expensive mercury fulminate, but makes a composition which runs more easily in the loading machines and, according to the tests of Storm and Cope, mixtures containing up to 20 per cent. potassium chlorate are more efficient than the mercury alone. The explosive reaction by which mercury fulminate decomposes may be written as follows:

$$HgC_2N_2O_2 = Hg + 2CO + N_2$$

Addition of an oxygen carrier, such as chlorate, oxidizes the CO to CO_2 thus increasing the total energy of the explosion. Potassium chlorate itself decomposes exothermically and adds additional heat to the reaction."[2]

Apparatus for Electric Shot-firing

Electric Blasting Caps

The electric blasting caps used in the United States are copper cylinders A, Fig. 1, open at one end, containing an explosive charge B that is detonated by the heating of a fine bridge wire E when an electric current is passed through it from wires D sealed into the end of the cap

[2] G. B. Taylor and W. C. Cope: Initia-Priming Substances for High Explosives. Bureau of Mines, *Tech. Paper* 162.

opposite the explosive charge. Electric blasting caps, either No. 6 or No. 8 strength, can be obtained with copper or iron wires. The maximum length of the iron wires is usually 8 ft. (2.4 m.); copper wires are furnished in any desired length.

Waterproof electric blasting caps are designed for depths of water up to 30 ft. (9.1 m.) but not for submarine blasting. They have enamelled wires insulated sufficiently for use in sea water. When blasting is done in depths greater than 30 ft., and where considerable time may elapse between the loading and firing of the shots, submarine electric blasting caps should be used; these are usually manufactured only on special order. Nothing weaker than a No. 8 cap is recommended for submarine blasting.
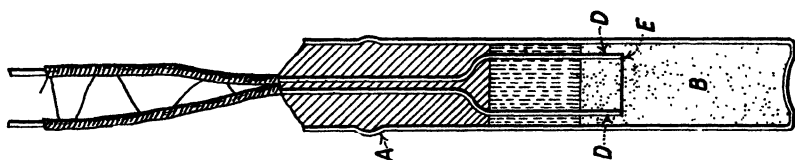
FIG. 1.—ELECTRIC BLASTING CAP.

Delay electric igniters consist of copper cylinders made on the same principle as an electric blasting cap, but instead of an explosive charge at the end of the capsule, a piece of high-grade fuse is crimped firmly against the priming composition. These fuses vary in length from 2 to 12 in. (5 to 30 cm.). When the igniter is used for firing high explosives, a blasting cap is crimped on to the end of the fuse; for black blasting powder, the blasting cap is unnecessary. The current passing through the cap ignites the fuse. Different delays in detonating the explosive are obtained by varying the length of the fuse in the igniter.

Delay-action exploders are similar in principle to delay-action igniters except that a blasting cap is securely crimped on to the fuse and the exploder waterproofed by the manufacturer. A hole is left through the sulfur plug of the exploder so that the gases from the burning fuse may escape. This hole is closed by a shot valve and a waterproofing compound, which prevents any water from entering until the interior gas pressure is sufficient to keep it out.

Electric Squibs

Electric squibs are used to fire blasting powder. They resemble electric blasting caps in appearance but the shell is made of pasteboard coated with a waterproof solution. The charge is a pellet of specially compressed powder, which is protected by a cork placed in the end of the squib; this cork is removed before the squib is placed in the charge of

powder. The squib, when fired, shoots a long hot flame into the charge of powder. The electric squib can be placed in the center of the charge, insuring a quick and strong explosion, with less smoke than when an ordinary squib is used.

Blasting Machines

The blasting machine commonly used in the United States in recent years is a small, portable, series-wound, two-pole, direct-current generator. The armature, of the Siemens type, and the field are solid iron. A long rack with a handle is meshed with a pinion on the shaft of the generator. The external circuit is connected in parallel with a switch that short circuits the generator. As the rack bar is forced downwards, current flows from the generator and is short circuited by this switch; at the end of the stroke, the switch is opened by the rack, and the current is forced, by a comparatively high voltage, through the external circuit.

During the war, blasting machines were used by the Ordnance Department and Engineers Corps of the army, but as the common industrial type did not meet all the requirements for military service, the Bureau of Standards was asked to design a generator of approximately the same size and weight, but with greater capacity and reliability. As a result, there was developed a portable generator capable of firing 100 electric blasting caps connected in series, without danger of failure. The weight of this machine was no greater than that of the earlier type rated to fire 30 electric blasting caps. Thousands of these new machines were ordered for the army and gave satisfactory results. They are now replacing, to some extent, the older type of blasting machine on industrial work.

Some interesting information was obtained by the Bureau of Standards while doing this work. First, the current required to fire the electric blasting caps and the time between the application of the current and the breaking of the circuit by the explosion of the fastest cap were determined. A fine copper wire was tied around each cap to be tested in such a way that it would be broken by the explosion of the cap. A weak electric current passing through this copper wire, as well as the actual firing current passing through the bridge wire inside the electric blasting cap, were recorded on an oscillograph. From the photographic records the instants at which the firing circuit was closed, at which the fuse wire was melted by the current, at which the outer wire was broken by the explosion of the cap were determined to 0.0001 sec. The following conclusions were then drawn:

1. Several types of caps tested have approximately the same current-time characteristics.

2. A current of at least 0.4 amp. is required to insure certain firing of a single cap.

3. With small currents, the explosion occurs before the resistance wire is fused by the current.

4. With large currents (above 3 amp.), the wire is fused by the current before the explosion occurs.

5. The measurements at the large currents indicate a fairly constant time interval of 0.004 sec. between the heating of the wire to the ignition point and the explosion of the charge.

6. With currents of 2 amp. and below, the maximum variation in firing time among different primers is greater than this 0.004 sec.; consequently, there is some danger that the circuit may be interrupted by the explosion of the quicker caps before the slower ones reach ignition temperature.

7. In the tests with three caps in series, all three caps fired in all trials at 1 amp. or above.

The conclusions reached from these tests were that the small current (only 0.9 amp.) delivered by the old-style machines explained the failures that occurred when using them. The blasting machine designed by the Bureau of Standards delivers 1.5 amp. at 300 volts when connected to a circuit of 100 electric blasting caps in series, and the current is correspondingly greater when fewer caps are used. With the new machine, the caps are fired in about 0.01 sec. with practical certainty.

Pocket-size Blasting Machines.—Small blasting machines with a capacity of from one to five electric blasting caps are very convenient for certain work. A small blasting machine that will fire five electric blasting caps consists of a permanent magnet, in the field of which a coil is revolved by twisting a handle that is detachable from the machine. This type of blasting machine is particularly well suited for much coal-mining work.

Leading and Connecting Wires

Connecting wires are used to join the wires of electric blasting caps when they are not long enough to reach adjoining holes; leading wire is used for completing the circuit between the connected blasting caps and the source of current. These wires, as well as the cap wires, may be either copper or iron. Copper leading and connecting wires are preferable and should always be used on large shots. If iron wires are used, they should be of a size corresponding to the resistance of the equivalent length of copper wire, inasmuch as they have a much higher resistance per foot. Insulated copper wire, No. 20 or No. 21 Brown & Sharpe gage, is suitable for connecting wire; No. 14, Brown & Sharpe gage, insulated copper wire either single or duplex is commonly used for leading wire.

Duplex leading wire consists of two insulated copper wires with an outside insulation. It weighs approximately twice as much as the same length of single wire, and is often more convenient to handle. Where

the leading wire is apt to be damaged by flying rock, however, it is better to avoid duplex wire. Short circuits are caused by damaged paired cables. The insulation makes it difficult to locate such short circuits in duplex leading wires; it is, therefore, simpler and better to use instead, two separate well-insulated wires of the proper diameter. With this arrangement, the chance of having a short circuit is reduced and any injury to the wires more readily located. It is better to suspend leading wires near the top of tunnels or drifts. If they are laid on the floor, moisture or contact with rails and other metallic articles may cause short circuits.

When connecting electric blasting cap wires, the ends of the bare wires should be scraped with a knife, the wires joined with a long twist, and twisted together tightly to keep down resistance to a minimum. When connecting the ends of the electric blasting caps or the connecting wire to the main leading wires, the detonator or connecting wire should be wrapped tightly around the end of the leading wire about 1 in. (25 mm.) from the end; then the end of the leading wire bent back and one or two turns of the connecting wire made around the loop.

Bare joints in the circuit should be protected against short circuiting on wet work. This can be done in mines by supporting the wires from the roof, or, if this is not practicable, by laying the wires on stones or sticks on the ground. An additional precaution, when the wires are on the ground, is to wrap all joints with tape.

Apparatus for Testing Blasting Circuits

Each electric blasting cap should be tested before it is used; it is also advisable to test blasting machines occasionally to determine their capacity. Electric blasting caps and electrical connections are tested with galvanometers and ohmmeters; blasting machines are tested with rheostats.

Galvanometers

A type of galvanometer commonly used by blasters is illustrated in Fig. 2. The cell and dial with a movable hand are contained in a case of metal and hard rubber, which is enclosed in a leather case with straps for carrying over the shoulder. A silver-chloride dry cell is used. The current passes through high-resistance coils, so that when connection is made to the galvanometer terminals the current is less than one-tenth of that required to fire an electric blasting cap. When the galvanometer is properly used, several thousand tests can be obtained from one cell. Replacing an exhausted cell is a simple operation.

To test an electric blasting cap or a blasting circuit the wires are placed on the terminals of the galvanometer and the deflection of. the needle is noted. No definite figure can be given for the distance the needle will deflect, as this varies with the age of the cell. The blaster

soon learns the deflection to expect from different circuits. As the cell
is used it constantly grows weaker, which causes a gradual diminution
in the deflection of the needle on the galvanometer. When one electric
blasting cap is tested with a new cell the needle should deflect almost
completely across the scale. The greater the number of caps, the longer
the wire on each cap, and the greater the length of connecting and lead
wire in the circuit, the greater will be the resistance and, consequently,
the less will be the deflection of the needle. After some practice, the
blaster will know the amount of deflection to expect from any circuit
that he tests. If the needle moves too far
across the scale, there are probably short
circuits or leaks; if it does not move far
enough, there is too much resistance in the
line, probably caused by poor connections; if
it does not move at all, the circuit is broken.
To find where the trouble is, the galvanometer
is connected with different sections of the
circuit, until the part that contains the poor
connection, short circuit, or broken wire is
found.

One method of doing this is as follows:
Assume a series connection with five electric
blasting caps in which the galvanometer

FIG. 2.—INTERNAL CONSTRUC-
TION OF GALVANOMETER.

test at the blasting machine shows that the
circuit is broken. Connect the galvanome-
ter in the circuit, using the leading wire and the first blasting cap. If
the needle deflection shows that this circuit is all right, include the
second cap in the circuit. Continue to test, adding a cap each time
until the galvanometer again shows a broken circuit; the trouble is then
known to be in the last cap or connection tested. If the needle does not
deflect at all, the cap is broken; if it does not deflect far enough, there is
too high resistance, probably caused by a bad connection; if it moves
too far, there is not enough resistance, probably because of a leak or short
circuit, which can then be easily fixed.

Ohmmeters

While a galvanometer shows the condition of the circuit, it does not
give an accurate test for a number of loaded holes. A reading is obtained
as soon as the current from the cell in the instrument passes through the
circuit. This, however, is not a sure indication that the current is
passing through the wiring and the caps as it should, because part of it
might be flowing through leaks in the insulation, grounds, iron supports,
etc., and does not reach the caps. In fact, if there is a short circuit, the
reading will be better than usual.

For testing where electric blasting caps are connected in series, an ohmmeter is the most reliable and accurate instrument. Ohmmeters can be obtained, mounted in a small portable case. They consist essentially of a slide-wire Wheatstone bridge, with the scale divided to read either in ohms or in per cent. of a fixed resistance value. One model. made by a well-known manufacturer, has a galvanometer mounted in the rubber plate, and battery and galvanometer keys are provided.

To show how trouble is detected with the ohmmeter, assume that in a series connection the resistance of the leading wires in the circuit, either calculated or determined by testing with the ohmmeter, is 6.4 ohms; the resistance of the electric blasting caps is 1.5 ohms each; and that there are 20 caps connected in series. The total resistance of the circuit is then 36.4 ohms. If the ohmmeter does not show approximately this figure there is trouble somewhere in the circuit. This may be analyzed as follows:

1. A reading greater than 36.4 ohms indicates poor connections in the circuit.

2. A reading less than 36.4 ohms indicates that some of the caps have been omitted from the circuit or the circuit is not properly connected up.

3. A reading about 6.4 ohms indicates a short circuit in the leading wires, with no current passing through the caps.

4. A reading about 3.2 ohms indicates a short circuit in the leading wires with current passing through about only one-half of the total length of the leading wires and through none of the caps

By this method, trouble is detected at once and the source quickly discovered, but these tests hold good only with series connections. The ohmmeter can be used for testing each series separately in a series-parallel or parallel-series connection, but is not sufficiently delicate for a straight parallel connection, where the resistance is decreased a slight amount by each additional cap added to the circuit.

Rheostats

Rheostats used for testing blasting machines consist of coils of high-resistance wire connected to six binding posts. The resistance of the entire length of wire is sufficient to test 100 electric blasting caps with 30-ft. (9-m.) wires, with a sufficient factor of safety to allow for the leading and connecting wire in the circuit. Between adjacent binding posts there is connected wire with a greater resistance than the number of electric blasting caps stamped on the instrument between the posts.

If it is desired to determine whether a blasting machine will fire 40 caps, let us say, one post of the blasting machine is connected to the binding post on the left of the figure 40 on the rheostat; one wire from an electric blasting cap is then connected to the post on the right of the

figure 40 and the other wire from the blasting cap is connected to the second post·of the blasting machine. Sufficient leading and connecting wire should be used to permit the operator to be a safe distance from the cap when it is fired by pushing down the rack of the blasting machine. If the cap does not fire, it is not safe to use the blasting machine to fire 40 blasting caps. A blasting machine can be tested for various numbers of caps by making connections to the posts on either side of the number used. To test for 35 blasting caps, connections are made to the first and fourth posts, the total resistance of which is $5 + 10 + 20 = 35$. The resistance between posts one and six is equivalent to the sum of all the numbers included between them, or 100 blasting caps.

Calibrating Galvanometers

A galvanometer can be calibrated by testing with a rheostat; the resistances between the various posts of the rheostat are obtained from the manufacturer. The galvanometer is connected between posts 1 and 2, 1 and 3, 1 and 4, 1 and 5 of the rheostat; then by comparing the galvanometer readings with the known resistances between the posts of the rheostat, the complete scale of the galvanometer can be calibrated, in ohms. It is necessary to recalibrate the instrument from time to time as the cell grows weaker. A fairly close check is obtained with the calibrated galvanometer by comparing the total resistance of the electric blasting caps, leading and connecting wire in a circuit with the reading obtained on the galvanometer.

Limitations of Plate Test

Little dependence can be placed upon the plate test sometimes used for testing blasting caps and electric blasting caps. In this test, the cap is exploded with its bottom or side in contact with an aluminum or lead plate; the strength and condition of the cap is then judged from the resulting appearance of the plate. There are a number of reasons why this test is not a reliable one. The indentation in the base of the cap, if any, the diameter of the cap, the nature of the charge and of the metal cylinder that contains it, the moisture in the charge and other variable factors all have an effect on the plate test. This test is useful only for detecting caps that have become defective and for testing certain detonators of the same grade, strength, and manufacture.

CONNECTIONS FOR ELECTRICAL FIRING

Series Connection

When the caps are connected in straight series, the current flows from one cap to the next, and so on through all the caps. If one cap should fire an appreciable length of time before the others, the circuit is

broken and the other caps will probably not fire. For this reason the series connection is the least desirable, but it must be used when firing with a blasting machine because the current generated by these machines is of short duration and not sufficient to fire a group of caps connected in other ways. A current of at least 1.5 amp. should be used, regardless of the number of caps in series, to assure a sufficiently large factor of safety. The resistance of each cap used can be obtained from the manufacturer, and the required voltage figured by Ohm's law (volts = current × resistance), using 1.5 amp. for current.

To illustrate the calculations for a series connection suppose that it is desired to know what power is required to fire 25 No. 8 electric blasting caps with 6-ft. (1.8 m.) iron wires, with the source of power 1000 ft. (305 m.) from the shot, assuming that the cap wires can be connected to one another without using connecting wire. The current for straight series connection is 1.5 amp. As the resistance of each cap is 1.343 ohms, the resistance of all the caps is 25 × 1.343 = 33.57 ohms. The resistance of No. 14 copper lead wire is 2.525 ohms per 1000 ft. Lead wire resistance is 2 × 2.525 = 5.05 ohms. The total resistance of the circuit is 33.57 + 5.05 = 38.62 ohms. The required voltage (by Ohm's law) is 1.5 × 38.62 = 57.93 volts.

In large shots connected in series, the whole circuit can be tested with an ohmmeter before firing. The theoretical resistance can be accurately figured from the known resistances of the leading wire, connecting wire, and blasting caps; which should check up fairly closely with the reading on the ohmmeter. Series connections can also be tested with a galvanometer; this is not as accurate or reliable a test as that obtained with an ohmmeter, but it usually enables the blaster to detect and locate short circuits, broken wires or poor connections.

Parallel Connection

The parallel connection is the best and should be used wherever convenient and possible. In this connection, each cap is independent of all others and one defective cap cannot affect the action of the other caps in the group. Each cap is connected across the two lead wires, and differences in resistance of the caps do not affect the result, because with a sufficiently strong current every cap detonates, no matter what its resistance may be. This connection cannot be used when firing with a blasting machine, because more current is required than the machine can furnish. Neither can the entire circuit be accurately tested with an ohmmeter or a galvanometer, as can the series connection. About 0.75 amp. per cap is a safe figure for the current required with the straight parallel connection. With a large number of caps, the amperage required is relatively high so that this arrangement can be used only with a com-

paratively high current supply, but the straight parallel connection reduces the probability of misfires to the minimum.

The voltage required is figured by Ohm's law as in the series connection; the following problem illustrates the method of making the calculations: What power is required to fire 50 No. 8 electric blasting caps with 6-ft. copper wires, when 2000 ft. of No. 14 copper leading wire is required from the shot to the source of power and No. 20 copper line wire is used along the shot, with the holes spaced 4 ft. apart in a single line, as in blasting a ditch? The current required is $0.75 \times 50 = 37.5$ amp. The resistance of one cap is 1.343 ohms; therefore, the resistance of the 50 caps in parallel is $\dfrac{1.343}{50} = 0.027$ ohm, which is practically negligible. The resistance of 2000 ft. of No. 14 wire used for leads is 5.05 ohms. With the 50 caps spaced 4 ft. apart, the distance covered by the shot is 200 ft. Therefore, $2 \times 200 = 400$ ft. of, say No. 20 copper connecting wire is necessary for connecting the shot. The current in passing through this connection diminishes from full strength at one end to practically zero at the other. Therefore, we can approximate the amount of voltage required by using one-half of the resistance of the total wire in the line, viz., of 200 ft. of No. 20 copper wire which is $0.2 \times 10.15 = 2.03$ ohms. The total resistance of the circuit is $0.027 + 2.03 + 5.05 = 7.1$ ohms. The required voltage $= 37.5 \times 7.1 = 266.25$; approximately 270 volts.

Parallel-series Connection

The parallel-series connection, shown in Fig. 3, is better than the straight series connection, because it reduces the number of caps connected in straight series, and thereby reduces the chance of the circuit being broken by a more sensitive cap firing in advance of the others. The more closely this connection can be made to approach the straight parallel,
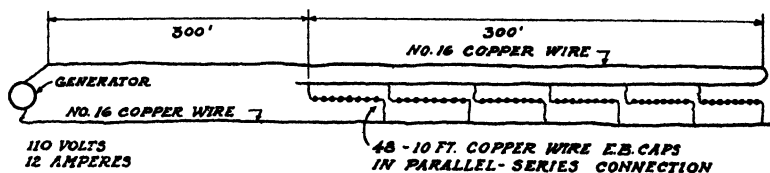
FIG. 3.—PARALLEL-SERIES CONNECTION.

the better. This is done by connecting as many groups of caps in parallel as the available current will permit. We recommend having 2 amp. per series and as many series as possible.

In the problem assumed in Fig. 3, there are 12 amp. and 110 volts at the generator. There are 48 holes to be fired approximately 5 ft. apart, in round numbers requiring about 300 ft. of connecting wire.

The generator is 300 ft. from the shot so that 600 ft. of No. 16 copper leading wire is necessary; combining this with the 300 ft. along the shot makes a total of 900 ft. of copper wire. The resistance of 1000 ft. of No. 16 copper wire is 4 ohms; the resistance of 900 ft. is 4 × 0.9 = 3.6 ohms, the resistance of the leads. The drop in voltage would equal 12 amp. × 3.6 ohms. = 43 volts approximately. The voltage at the shot would be 110 volts − 43 volts = 67 volts. In order to put the 12 amp. through the shot with 67 volts, we can have a maximum resistance of 67 ÷ 12 = 5.6 ohms approximately. This means that the resistance of the 48 electric blasting caps and the wire to which they are connected shall not exceed 5.6 ohms. Each 10-ft. copper-wire electric blasting cap has a resistance of 1.2 ohms. There are six series of eight caps each. As the resistance of each series is 8 × 1.2 = 9.6 ohms, the combined resistance of the caps in six series connected in parallel is 9.6 ÷ 6 = 1.6 ohms. We are allowed 5.6 so that there are 5.6 − 1.6 = 4.0 ohms left for the connecting wire. We must choose a wire 300 ft. long, the resistance of which will not exceed 4 ohms. From a table of resistances, we find the nearest to be No. 21 wire.

With this connection, it is desirable to have approximately the same resistance in each series. Consequently, approximately the same number of caps with wires of equal length should be used in each series; this number should not vary more than one cap. The lead wires should be connected so that they will cross opposite ends of the shot, as shown, otherwise the series farthest from the shot might not receive sufficient current. This system is open to the same objection as the straight series, but as there are fewer caps per series, there is less possibility of misfires. It is a simple connection to make and works successfully in practice.

Series-parallel Connection

The series-parallel connection is next best to the straight parallel connection. This system, shown in Fig. 4, approaches the parallel
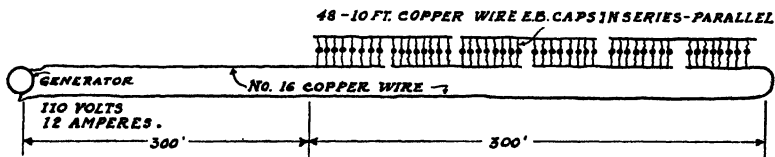
FIG. 4.—SERIES-PARALLEL CONNECTION.

connection in having each cap connected to the line separately, independent of other caps. The different groups of caps are connected in series in the firing circuit. A current of 1.5 amp. for each cap in a group is recommended; that is to say, if eight is the maximum number of caps in any group the minimum current to figure on is 12 amperes.

In a problem similar to that just given, the lead wires should be connected to opposite ends of the shot as in the parallel, and always to the shortest leg. Again, 900 ft. of lead wire is required, and the drop in the line is 43 volts, leaving 67 volts at the shot, with a current of 12 amp. The maximum resistance allowable in the caps and connecting wire is 67 ÷ 12 = 5.6 ohms. With a current of 12 amp. available and 1.5 amp. figured for each cap in parallel, there can be 12 ÷ 1.5 = 8 caps in parallel in one group; for the 48 caps, six groups are required. The resistance of one cap with 10 ft. of copper wires is 1.2 ohms; the resistance of each group, therefore, is 1.2 ÷ 8 = 0.15 ohm. For the six groups, the resistance is 6 × 0.15 = 0.9 ohm. This leaves 5.6 − 0.9 = 4.7 ohms for the wires along the shot to which the cap wires are attached. We have assumed that two wires each 300 ft. long are used. For all practical purposes, we can figure on one length of line wire, viz., 300 ft., with a resistance of not over 4.7 ohms. From a table, No. 21 copper wire is found to be the nearest to this.

We have determined, therefore, that with 110 volts and 12 amp. at the source of power, not over 900 ft. of No. 16 or larger copper leading wire, and not over two lengths of 300 ft. each of No. 21 or larger connecting wire along the shot, six groups of caps in series, each group consisting of eight caps in parallel, can be fired. There is large enough a factor of safety in these calculations to permit firing a few more caps, but it would be better not to do so. It is advisable to have the same numbers of caps in each series, but if necessary the number may be varied by not more than one. Always have the smallest number of groups in series and the greatest number of caps per group in parallel that can be arranged; in other words, make it as near a straight parallel connection as can be done with the current available.

FIRING FROM POWER CIRCUITS

Direct current is preferable for electric shot-firing and should be used wherever available. No varying factors are introduced, and the current, voltage, or resistance computed by Ohm's law can always be relied on, if the shot is properly connected.

Alternating current can be used, but its use is apt to complicate the calculations in making connections. Currents less than 25 cycles should not be used, and a 60-cycle current is preferable for 0.014 sec. is required to fire an electric blasting cap, and it is impossible to make each cap absolutely uniform. The heating and cooling of the bridge wire produced by the slower alternations may cause some cap to fire in advance of the others, thus decreasing the effectiveness of the shot and causing misfires with series connections. Other factors introduced are the various phases of different generators and the inductive resistances of motors, when these are included in the load. If, at the time of the blast

everything were shut off between the generator and the shot, these complications would not arise; but on large operations usually it is not practicable to do this.

Misfires with alternating current might be caused by a drain on the power line to supply other purposes so that insufficient current is left to fire the shot. To prevent this, the available voltage and amperage should be measured at the point where the lead wires from the shot connect to the power line; Ohm's law can then be applied, as there is no inductive resistance in the shot itself. Care should be taken that when the shot is fired the load on the power line is no greater than at the time the readings were taken. This difficulty may be eliminated by shutting off all loads on the line other than the shot at the time of firing, but it is not always practicable to do this.

Another cause for misfires, which may also occur when firing with direct current, is unknown and unanticipated resistance from poor connections or other causes, thus reducing the voltage. In one instance a poor connection introduced an unanticipated resistance of about 200 ohms, which however was detected and located with a galvanometer before it was attempted to fire the shot. Unexpected resistances may be caused by poor connections, dirty or loose switch contacts, or by short circuits from grounded lead wires, which, without care, is likely to occur on rainy days. An excellent precaution is to have a voltmeter across the line at the switch to check the voltage just before firing. Defective caps, another possible cause of misfires, may be detected by testing with a galvanometer before the primer cartridges are prepared.

Voltages higher than necessary should be avoided particularly in gaseous and dusty coal mines; sparking, caused by high voltage, may result in fires or explosions. Some instances have been reported where the leading wires fused before the caps fired. Unless necessary to perform the work, it is better not to use over 250 volts for electric shot-firing in coal mines.

ELECTRIC SHOT-FIRING IN UNDERGROUND METAL MINING

There are many advantages of shot-firing by electricity in underground metal mining. The fumes from burning fuse are eliminated; where large rounds are fired in poorly ventilated places this improves working conditions considerably. Misfires, caused by flying rock cutting off the fuse from other holes in the round, are avoided. In wet holes, misfires are often caused by water penetrating the fuse and damaging the powder or leaking into the cap; electric shot-firing eliminates this cause of misfires. Electric blasting caps not only resist water in wet holes but, because they are sealed, when stored in damp places they do not deteriorate as rapidly as open blasting caps. With electric shot-firing, all of the cut holes can be fired simultaneously and the remaining holes

fired in the exact rotation desired; this is a great advantage in pulling a tunnel or drift round. After the round is fired electrically, the men can return to the shot without delay, where there is good ventilation; whereas, to avoid possibly injury from hang fires where fuse is used, it is safest not to return to the shot for a considerable length of time, unless all of the holes are counted as they fire.

Electric shot-firing is particularly well adapted for most shaft sinking. Lighting a round of holes in a shaft is no child's play, even under the most favorable conditions. Seconds are apt to seem like minutes and it is only natural that the shot firers should try to give the signal for hoisting them out of the shaft in as short a time as possible. If the fuse gets wet on the end, if the lights are extinguished by water dripping down the shaft, or if anything else unforeseen occurs to cause delay, misfires usually occur. When shots are fired electrically, there is ample time to make all connections properly and to take every possible precaution against misfires. The men are relieved from mental strain, because they know they can take all the time they need and still be in a safe place before the round is fired.

In stoping, it is possible to break the rock finer by shooting a number of holes simultaneously with electric blasting caps than when the holes fire one at a time; crushing expense can often be reduced in this way. In timbered stopes, there is a danger from fire resulting from burning fuse touching the timbers. One fire thought to have been thus started necessitated flooding one of the country's largest mines and pumping it dry again when the fire was extinguished. Wiring stopes for electric shot-firing involves some extra effort, which is often the reason why it is not used. There are conditions, however, such as those just mentioned, that make it advisable to fire electrically in stopes, in spite of any inconvenience in installation.

"Electric shot-firing is preventing waste and reducing costs at a number of large tunneling operations. On the Hetch Hetchy water supply project in California, two bare copper bus wires supported by wooden plugs protruding from drill holes on each side, are stretched across the face of the tunnel. All of the electric blasting caps are connected in parallel to these bus wires; one wire from each cap is connected to one of the buses, the other cap wire to the other bus. The wires of the delay-action exploders are of different color so that they can be easily distinguished in the dim light of the tunnel. After all of the caps are connected, the shift boss readily checks up the connections by counting the number of wires connected to each bus. If all of the wires going to each bus are of like color, and if the number of wires connected to each bus is the same as the total number of holes in the round, the connections are properly made. Special delay-action exploders are used on this work, giving about double the standard interval of time

between shots. After the caps are correctly connected to the buses, two spools of connecting wire on reels are used to connect the bus wires with the main leads. Shots are fired from a safety switch, the cover of which cannot be closed until the switch is opened. From the switch to a point about 50 ft. from the face, the lead wires are carried on insulators on the opposite side of the tunnel from other electric wires; this is to avoid induced currents or short circuits. This system of electric firing helped to establish excellent progress records. A notable example is the advance of 776 ft. (236.5 m.) made in one month at the Priest tunnel.[3]"

Electric Firing in Coal Mining

The United States Bureau of Mines has for years recommended electric shot-firing because of its greater safety. Open lights for lighting fuse and the flame from the burning fuse powder are dangerous in gaseous and dusty mines. In a western coal mine in which, a few months before, a disastrous explosion had occurred, the shot-firing was done at night after the men had left the mine, the shot-firers being equipped with permissible electric lamps and Davy safety lamps for testing for gas. Fuse and caps were used for firing, however, and in each room the shot-firer struck matches to light a short piece of fuse, which acted as a "spitter" for lighting the shots. The flame from this spitter and the initial spit from the fuses were sufficient to ignite gas in the workings; the safety-lamp precautions, under these circumstances, therefore, were a farce. The use of matches and the flame from the burning fuse powder could have been easily eliminated by electric shot-firing. The only objection was the shot-firers' reluctance to change methods to which they had become accustomed, and their opposition to carrying a blasting machine through the mine. Even when told of the simplicity of electric shot-firing and of the light blasting machines, less inconvenient to carry than the fuse and loose blasting caps they carried, the attitude of the shot-firers did not change. The objection of some workmen to a change of method, even though it reduces the hazard of their occupation can, of course, be overcome by the management taking a firm stand; after the change has been made to electrical firing, the shot-firer is often its strongest advocate.

One of the most dangerous practices in coal mining is the use of "short fuses," sometimes not over 6 in. in length. At mines where the miners load and fire their own shots, even the utmost effort to eliminate firing with short fuses is unavailing as long as miners are permitted to fire with fuse. A sure way to eliminate this dangerous practice is to adopt electric shot-firing.

[3] N. S. Greensfelder: "Eliminating Waste in Blasting." Wilmington, Del., 1922. Hercules Powder Co.

In some anthracite mines, a considerable saving in the time required for firing the holes in a room is being made with delay-action electric blasting caps. Kynor[4] says that with delay electric blasting caps the time for firing the holes in an ordinary chamber face was reduced 70 per cent. A typical round of holes in the face of a room at this mine is illustrated in Fig. 5. The cut holes *A* are first fired with instantaneous electric blasting caps. Then holes *B*, *C*, *D*, and *E* are charged, primed with delay-action electric blasting caps, which allow an interval of 6 sec.
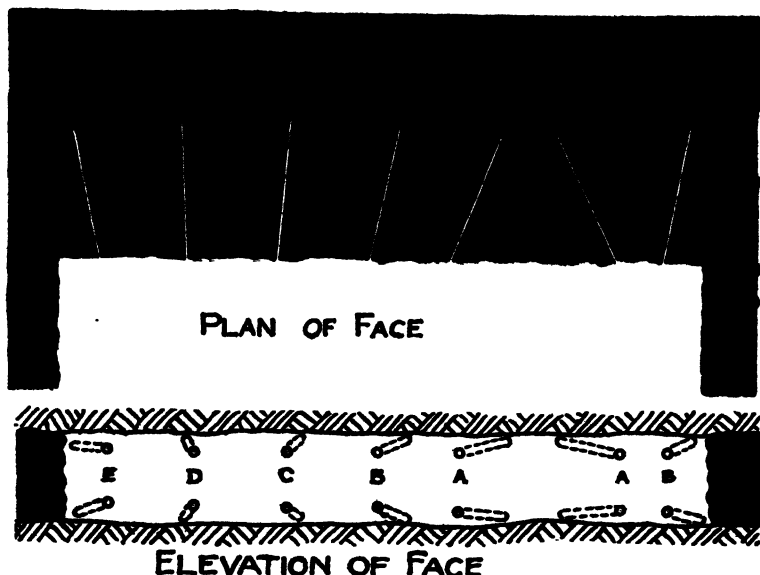
FIG. 5.—ARRANGEMENT OF DRILL HOLES FOR ELECTRIC SHOT-FIRING WHEN USING JACKHAMER AND MECHANICAL LOADING AT AN ANTHRACITE MINE.

between shots, and fired with one stroke of the blasting machine. The caps in holes *B* detonate first and are followed at 6-sec. intervals by holes *C*, *D*, and *E*, respectively. Formerly it was necessary for the miner to go back into the smoke after firing every hole or set of two holes separately; now it is necessary for him to return into the smoke only once. Delay-action electric blasting caps are reducing costs at other anthracite and some bituminous coal mines.

INSIDE FIRING SYSTEMS

Shot-firing inside the mine can be accomplished both from power circuits and by a blasting machine. Either system is safer than firing with fuse, in gaseous or dusty mines, but does not safeguard the lives of

the men as completely as do outside firing systems. Midget blasting machines, weighing less than 5 lb. (2.3 kg.) and capable of firing five electric blasting caps, are the most convenient for shot-firers to carry. They can be carried strapped over the shoulder or placed in a coat pocket.

The following systems of shot firing are used to meet various local conditions; electric shot-firing can be used in any of them.

1. The holes are drilled by miners, who carry the explosives and caps; they load the holes and fire them either during or after the regular working hours.

2. The holes are drilled by miners and loaded under the direction of shot-firers, who fire the holes either during or after regular working hours.

3. The miners take their own powder and caps into the mine and drill and load the holes; all holes are fired by special shot-firers after the miners have left the mine.

4. The miners take the powder into the mine but do not carry caps. After the holes are drilled, they are loaded by shot-firers, either alone or with the assistance of the miners; the shot-firers carry the caps and do all the firing after the miners have left the mine.

5. The shot-firers handle both powder and caps. The miners drill the holes but the loading and firing are done by shot-firers, usually after all other men have left the mine.

Increased safety and efficiency are usually obtained where the holes are loaded and fired by specially trained shot-firers after all others have left the mine.

OUTSIDE FIRING SYSTEMS

Outside firing systems are those in which power derived from a source outside of the mine is turned on at the surface after every one has left the mine. In the best-designed systems, switches are placed near the entrance of rooms and entries and near the point underground where the power-circuit enters the mine. There should also be provision at this point for breaking the circuit when men are in the mine, to guard against possible firing by discharges of lightning. With this arrangement of switches, the circuit can be closed step by step from the working place to the entrance of the mine. Adequate precautions are thereby provided against the firing of shots by stray currents, lightning, or accidental connection of the generator to the line. A system of wiring for an outside firing system used at a western coal mine is shown in Fig. 6.

Essential Features of Outside Shot-firing Systems

Recommendations for outside shot firing systems are summarized by the Bureau of Mines as follows:[5]

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[5] H. H. Clark, N. V. Breth, and C. M. Means: Shot-firing in Coal Mines by Electricity Controlled from the Outside. U. S. Bureau of Mines, *Tech. Paper* 108.

A sure method of checking men in and out of the mine should be used; any system based on memory is not as safe as one that gives visible evidence of the number of men underground.

The source of power should have a momentary capacity of not less than 30 kw. for a mine of average size; it may supply either alternating or direct current.

Where a large number of electric blasting caps are spread over a great distance, larger voltages may be necessary; but where sufficient for firing, 250 volts are recommended.
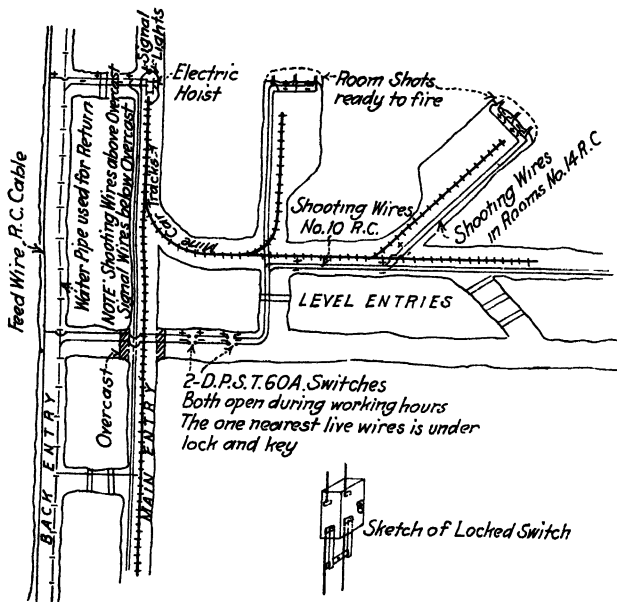
[FIG. 6.—DIAGRAM OF WIRING FOR OUTSIDE FIRING SYSTEM USED AT A WESTERN COAL MINE.

The shot-firing panel should be well mounted and housed and provided with two single-pole circuit breakers, a double-pole switch, and an ammeter; there should also be a switch for disconnecting the panel from the source of power.

All shot-firing conductors should be insulated and all parts of the circuit supported on insulators; dry wood is satisfactory for the insulators of room wires, but is not recommended for any other part of the circuit. Conductors should not be less than No. 14 (B. & S.) gage in rooms, No. 10 in cross entries, and No. 8 elsewhere.

Arrangements should be made for opening a large gap in each side of the shot-firing circuit, at or near the point where the circuit enters the mine; this gives protection against accidental firing by lightning discharges.

There should be not less than four switches between the source of power and the point of firing; at least three of these switches should be locked. The cross-entry switches should be enclosed in locked boxes.

The electric blasting caps should be connected in parallel, and the connections between the caps and the room wires should be made by men specially assigned to this duty.

ELECTRICAL FIRING AT OPEN-PIT MINES AND QUARRIES

At open-pit mines and quarries, the usual practice is to shoot simultaneously a number of holes containing relatively large charges of explosives. Better results and greater efficiency from the explosives are obtained in this way, than if the holes fired one at a time. Simultaneous detonation may be accomplished either by the use of electric blasting caps or cordeau-bickford detonating fuse.

Cordeau-bickford and methods for using it have been described by Harrison Souder.[6] Where convenient it is advisable to detonate the cordeau with an electric blasting cap. The average rate of detonation of cordeau is about 17,500 ft. per sec.; when it is placed along an explosive charge in a deep hole the entire column of explosive is detonated at a rate equal to that of the cordeau. This increases the shattering effect of the slower explosives, and, consequently, decreases the cost of secondary blasting. When the explosive charge is not continuous, but is placed at different points along the hole, cordeau furnishes a particularly convenient and efficient means of firing. The charge can be broken as many times as desired, because the cordeau extends along the entire length of the hole and, therefore, detonates all of the explosive. In the deeper holes it is necessary to use countered cordeau and often advisable to use double countered cordeau neither of which breaks as easily as the plain cordeau.

An Ohio company producing crushed stone found that the amount of explosive necessary was reduced more than 10 per cent. by replacing electric blasting caps with cordeau. Missed holes were also practically eliminated. At the Nevada Consolidated Copper Co., where churn-drill holes are first chambered by dynamite and then loaded with black blasting powder, it has been found that the holes can be loaded more quickly and with greater safety with cordeau than with fuse and blasting caps and the danger from unexploded caps and powder in missed holes has been eliminated.

Although cordeau has certain advantages over blasting caps in firing deep churn-drill holes, electric blasting caps are still used at many mines and quarries. Their initial cost is less and, if the work is properly done, they give good results. In holes containing large charges of explosives, a good precaution is to place two or more electric blasting caps in each

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hole, or two caps in each separate charge in the hole if several charges, separated by stemming, are used. If the holes are wet, waterproof caps should be used; if there is over 30 ft. of fluid in the hole, submarine electric blasting caps are desirable. Care should be taken in loading to prevent injuring the insulation or breaking the wires. Frequent galvanometer or ohmmeter tests should be made when loading large charges. Electric blasting caps should not be handled during an electrical storm, as in a number of instances they have been detonated by the lightning discharges. If the ground is damp, the joints should be taped or raised off the ground to prevent grounds or short circuits. Each circuit should be tested for resistance with a galvanometer or ohmmeter, and if a blasting machine is used, its capacity should be tested with a rheostat, before an attempt is made to fire the shot. The different connections that can be used and calculations required are given under the head Connections for Electrical Firing.

DEEP-WELL SHOOTING BY ELECTRICITY

In the past, the most common explosive for shooting oil wells was liquid nitroglycerin.[7] It possesses great strength and a high rate of detonation and is easily detonated because of its extreme sensitiveness. The latter quality, which is a favorable one after it is loaded in the well, makes the handling of nitroglycerin for well-shooting an extremely dangerous occupation. Solid explosives have been developed that give good results in deep-well shooting and are much safer to handle than liquid nitroglycerin. They can be transported by any public carrier under I.C.C. regulations.

Several ingenious methods of firing charges of these solid high explosives have been devised. After the charge is lowered in the hole, it is fired with a squib. The most common and effective squibs are the Jack or some type of electric squib.

The Jack squib is suitable for open holes where there is no danger of caving and where the casing does not have to be moved, because of its proximity to the shot. One form of Jack squib is a tinned iron tube from 3 to 6 ft. long, in which is placed one or more cartridges of dynamite, primed with a No. 6. or a No. 8 blasting cap and fuse. The primed cartridges are packed tightly in the tube with sand and the squib is weighted on the bottom. When ready to shoot the well, the fuse is lighted and the squib is dropped down the hole. It comes in contact with the main charge on the bottom, which it detonates when the squib explodes. Jack squibs are not recommended where there are more than 600 ft. of fluid in the well.

Where wells are drilled through caving material or where it is necessary

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[7] J. Barab: "Deep Well Shooting." Wilmington, Del. Hercules Powder Co.

to pull the casing before shooting, an electric squib is required. An ordinary electric squib satisfactory for shooting wells with fluid heads of 200 to 300 ft. consists of a small shell about 3 in. in diameter and 15 in. long, filled with torpedo gelatin and primed with a cartridge of 60 per cent. straight nitroglycerin dynamite in which two electric caps are inserted and connected.
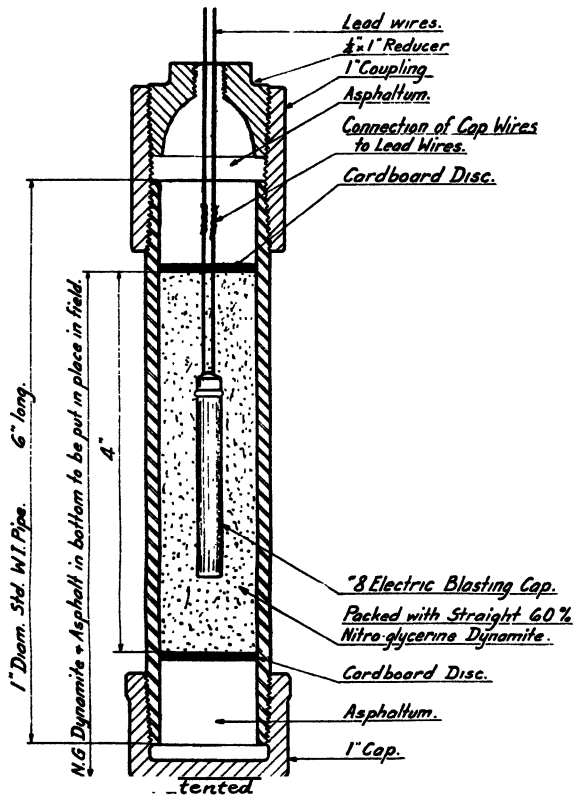
FIG. 7.—HIGH-PRESSURE ELECTRIC DETONATOR FOR OIL-WELL SHOOTING.

For wells where there is a greater fluid head, specially constructed electric detonators are advisable; one that is suitable for high fluid pressure is shown in Fig. 7. A small shell, similar to that just described, is packed with torpedo gelatin primed with this special electric detonator, and lowered until it comes in contact with the main charge. No. 14 copper wire, well insulated, should be used for lead wire. The caps should be properly waterproofed and all connections carefully made.

Casing Squibs

When it is necessary to shoot a string of casing in two, a special casing squib should be used; a good type is shown in Fig. 8. It contains a
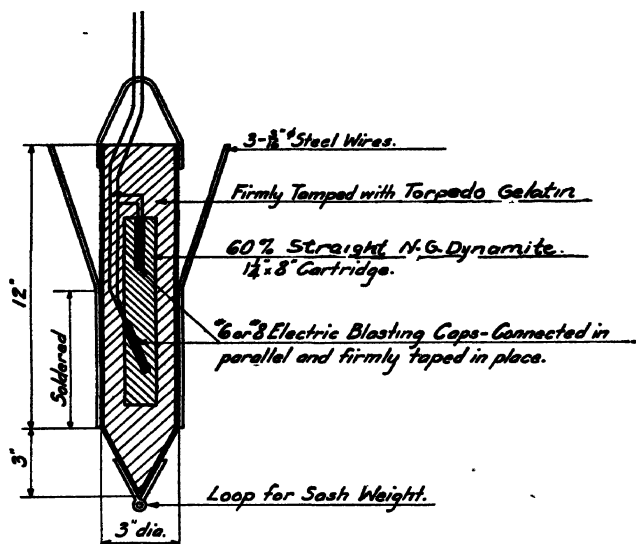
FIG. 8.—CASING SQUIB.
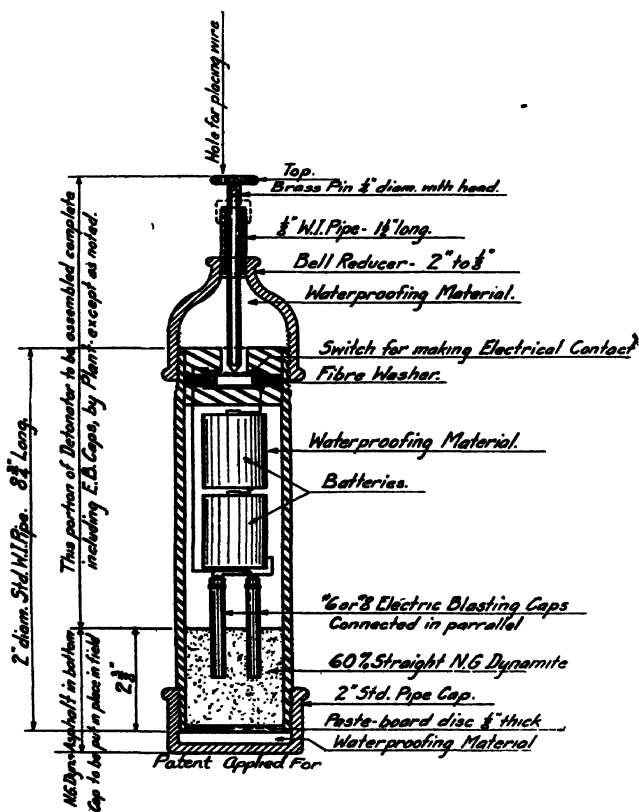
FIG. 9.—SELF-CONTAINED HIGH-PRESSURE ELECTRIC DETONATOR.

small charge of torpedo gelatin primed with 60 per cent. straight nitro-glycerin dynamite and electric blasting caps. A sash weight is attached to the bottom of the squib and the antennæ that project upward from the bottom are bent out. The squib is then lowered to the desired position, and as it is raised up and down at this point, the antennæ catch on the nearest collar joint. When in this position, the wire is fastened to hold the squib in place and the squib is fired from any convenient source of current.

Special Self-contained Electric Detonator

A self-contained electric detonator, particularly adapted for shooting torpedo gelatin under high fluid pressure in wells, has recently been developed. It can also be used satisfactorily where Jack squibs are used or for casing shots and trouble work.

This detonator, shown in Fig. 9, is lowered on a stout copper or iron wire to the desired position. When ready to fire, a 1-in. pipe, about 15 in. long is dropped over the wire. When the pipe strikes the firing head, contact is made which explodes the detonator. The chief advantage of this squib is the saving in cost of the double insulated wire required in regular electric firing.

ACKNOWLEDGMENT

The author wishes to acknowledge valuable assistance, in the preparation of this paper, by J. H. Horlick, Jr., J. Barab, and E. M. Symmes.

DISCUSSION

W. S. WEEKS, Berkeley, Calif.—Some years ago two students at the University of California did some research work on the subject of electric blasting and detonators. Anybody who has studied these will agree that the caps should be connected in parallel wherever possible. We were able to put two detonators in series and beginning with a very low current and gradually increasing it we could shoot one of those detonators without the other; then the other detonator would be connected on the line and the current gradually built up until that one would shoot, showing there was nothing wrong with the detonator. These men found that detonators required different amounts of current, or different amounts of heat, and that if all are in series the ones that require the least heat will shoot first, thus causing misfires in the others. The little platinum bridge that causes the detonator to be fired is the point where the heat-generating current passes in. We tried to determine the cause of the difference in resistance of different caps and made many measurements of the platinum wire but there was practically no difference in the diameter of the wire. This difference in resistance is, probably, the result of differences in

length and in the soldering. The different resistance of the bridges has no effect on the time of firing; that is, one cannot say the detonator having the higher resistance will shoot first; there is absolutely no relation between the resistance of the bridge and the time of the firing.

The delay-action exploder is merely a platinum bridge which lights a short piece of fuse that, in turn, lights a detonator. In the cut holes, it is desirable that charges go off simultaneously, in order that one can help the other. We found that if instantaneous exploders, that is those in which the bridge immediately lights the fulminate and they go off instantly, are placed in the cut that the blasting of the cut will snuff out the fuses in the delay-action exploders before they have time to start. As a result there has been developed what is called a no-delay exploder; this has a very slight delay which gives the fuses in the delay-action exploders time to start burning before the shot goes off. In our experiments we were able to use the instantaneous exploders in the cut by carrying three wires to the face from a three-pole switch. There were one negative and two positive poles, which made two distinct circuits, one carrying the delay-action exploders and one carrying the instantaneous.

On this three-pole switch, two of the knives were made long so that the delay-action exploders were lighted first, and then as we completed the closing of the switch, the instantaneous exploders were lighted.

Mining Methods at the Ashio Copper Mine

By Masayuki Otagawa, Tokyo, Japan

THE mining methods adopted in Japanese mines are less known to the mining world than those of other countries, owing to the geographical remoteness, but they present many features of interest to mining engineers.

The Ashio copper mine is at the town of Ashio, in Tochigiken, about 109 mi. (175 km.) from Tokyo by the Imperial Government railroad. The area of the concession owned by the Furukawa Mining Co. is 5000 acres (5023 ha.). The mine was formerly divided into three, viz. Honzan, Kotaki, and Tsudo, which are distant 3 mi. (4.8 km.) from each other as shown in Fig. 1. However, in 1914, when the writer was general manager, the three mining plants were amalgamated into one with the result of simplifying the working projects and increasing the operating efficiency.

The mine was visited by the Institute party in November, 1911, when K. Inouye was the general manager. Shortly afterward, the writer was appointed manager, and, in 1915, he was succeeded by Dr. K. Asano. At the end of 1917, K. Kibe became general manager and, after two years, was succeeded by the present manager, I. Sugimoto. For some time Dr. J. Kojima was acting general manager. It is needless to say that the gentlemen mentioned are all members of the Institute, as is also Baron T. Furukawa, president of the company.

Since the discovery of the enormous masses, called "kajika" or "bullhead" deposits, which the writer has described elsewhere,[1] the mining methods have been varied to meet the different conditions presented by these deposits.

THE ASHIO ORE

The country rock of this mine consists of the middle Chichibian slate and quartzite into which has been intruded a post-Tertiary plagioliparite neck more than 2 mi. in diameter, in the shape of an inverted cone which narrows downward to a relatively small diameter in depth.

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[1] *Mining and Metallurgy* (December, 1920) 54.

Nearly all of the deposits are veins, and they occur for the most part in the liparite, although a few are found in the sedimentaries. It has been generally believed that there are two principal series of veins, one striking north 30 to 60° east, called the Yokomabu series, and the other striking north 80 to 85° west, called the Shinsei series. There are more than 200 lodes, and among them 40 of the former series and 60 of the latter are now being worked. They intersect each other, forming rhombic nets of veins. They are from 1 to 16 ft. (0.3 to 4.8 m.) thick, 400 to 6000 ft. (121.9 to 1828.8 m.) long, and usually have steep dips. The lodes are more numerous in the upper levels, due to the fact that some of them unite on dip and some die out.

The chief vein minerals are chalcopyrite and pyrite, but zinc blende, arsenopyrite, galena, and pyrrhotite are also of frequent occurrence. Bornite, chalcocite, cuprite, malachite, and pisanite, with occasional azurite and native copper, are observed in the oxidized zone. Although the principal ore is chalcopyrite, it is believed that there is a commercially important amount of secondary chalcocite enrichment in the upper levels. There is a small amount of gangue, consisting of quartz, clay, and chlorite in the upper levels, with less quartz and more calcite as depth is gained.

Some of the deposits in the sedimentaries seem to be metasomatic bedded veins and form a network of ore stringers, while others are irregularly arranged masses, which are called "kajika" (bullhead deposits) by the miners because they are found behind the vein walls, as a bullhead is frequently caught by lifting or removing pebbles in the river bed.

Brief comment on the discovery of these kajika deposits may be of interest. As long ago as 300 years, they were known and worked at Sunokobashi near Tsudo adit. At that time they were mined, or rather excavated, in an open cut. The name of one such open pit, Senjojiki (thousand mats, one Japanese mat being 6 ft. by 3 ft.) still remains, and the size of the bonanza could well be imagined for the "gold screen" of these days. Up to the present time six principal kajikas have been discovered, viz., the Deai, Kosei, Taisei, Tsuruhi, Ninenhi, and Tengu, each named from the vein and level in which they are encountered, as shown in Fig. 1, where the kajikas are numbered from 1 to 14.

The largest kajika, the Tengu, which was discovered at the end of 1919, is an enormous mass, a metasomatic bedlike deposit along a plane of folding in the quartzite, and is a belt-shaped oreshoot dipping 30° to 50° north-northeast and pitching 15° east to its dip. It has now been developed about 100 ft. (304.8 m.) long in strike, 30 to 50 ft. wide in true thickness, and 1000 ft. in vertical extension.

Generally speaking, kajika deposits are found at or near the junction of two or more fissures, having resulted from the sulfide enrichment
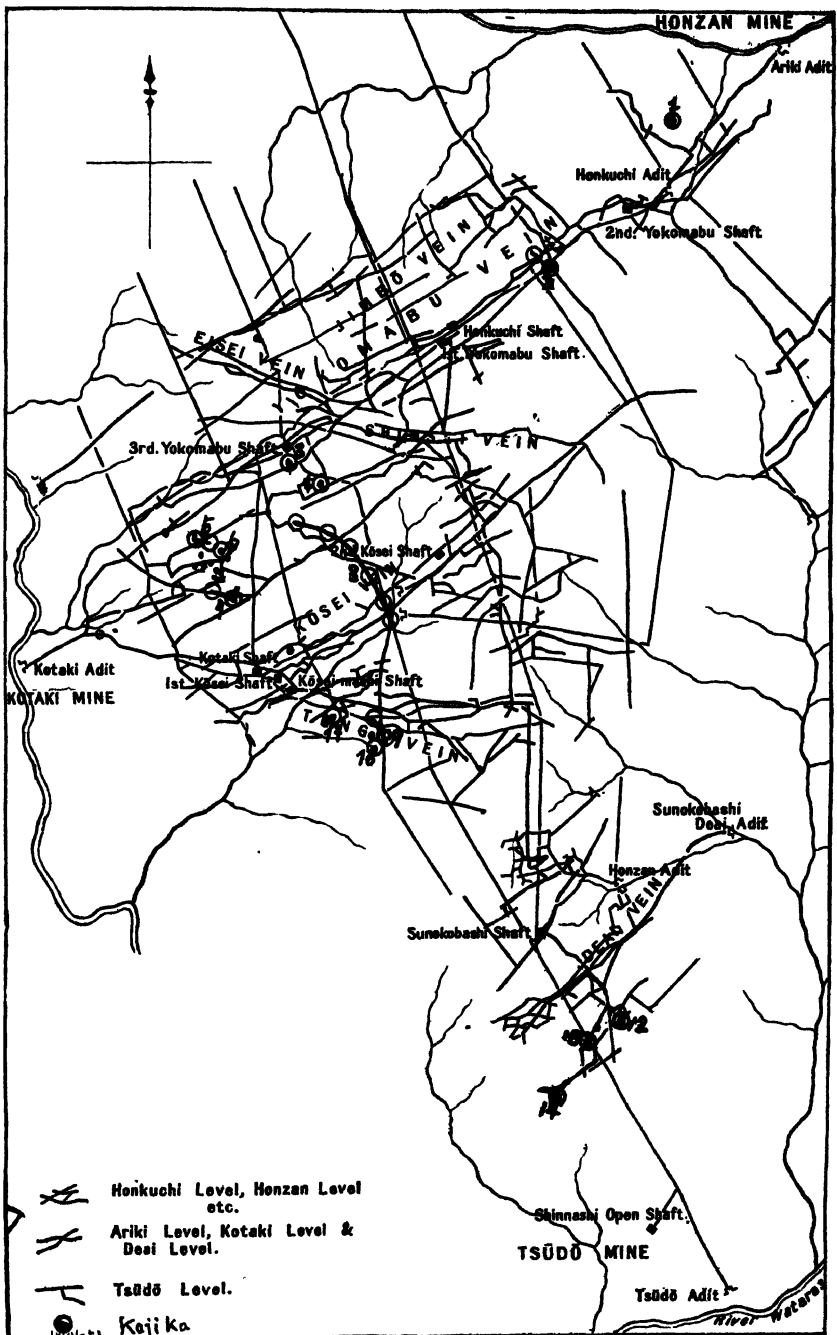
FIG. 1.—UNDERGROUND MAP OF ASHIO COPPER MINES.

and replacement of included slate fragments in the liparite. As already pointed out, the mine possesses a great many fissures which have been grouped into two main systems. However, H. V. Winchell has expressed the opinion that there is still another system of fractures which were complementary or produced at slightly different periods. As the writer also agrees with Mr. Winchell's view, it may be considered that the kajika deposits have no direct genetic relation to the ordinary fissure fillings in the liparite.

The kajika ore is chalcopyrite with neither gangue nor pyrite. In the higher zones, the chalcopyrite has been altered to bornite and chalcocite. The sulfide ore contains from 30 to 50 per cent. copper in the rich part; the average is generally more than 15 per cent. The boundaries of the deposits are not well defined, since there is usually a marginal chloritic and sericitic alteration. It is worthy of note that the commercial copper ores of the United States are mainly chalcocite, as illustrated in Arizona, Montana, Utah, Nevada, and New Mexico, whereas chalcopyrite is the principal source of copper in Japan.

MINING METHODS

The mine is developed by adit levels and blind shafts and is at present being worked to the depth of 3500 ft. (1066.8 m.) below the surface. The daily output is 1000 tons of copper ore, mined from veins and kajikas in nearly equal amount.

Overhand stoping is employed in the veins. Each stope is 6 ft. (1.8 m.) high, 10 to 15 ft. long and more than 2½ ft. wide, depending on the thickness of the vein. The system of mining the kajikas is mostly by stepped-face, square sets. As shown by Fig. 2, development drifts are in either wall or the deposit with raises to the level above, and the sill floor is breasted out from the drift to each wall. The room is supported by round square-set timbering, 6 ft. square, center to center, both cap-wise and girt-wise and 6 ft. high. The sill floors are 7 ft. high. The diameter of the timbers for post, cap, and girt is 10 in.; on the main sill, 12 in. Since the ground is firm enough, the sets alone are relied upon to support the walls without filling. The ore is drawn off through inclined chutes into cars on each level and trammed to the shaft station. In most cases, cages are used in the shafts, but a skip with 2.5 tons capacity is installed in the Yokomabu shaft No. 2, which has a depth of 1080 ft. Kajika deposits are too wide for stulls, and the ground is fairly strong, but requires prompt support to prevent slabs falling from the back and walls; so this method of mining is justified by the lower cost of exploitation.

Although there are still many mines worked by the square-set system

in the United States, in spite of the growing preference for shrinkage, filling and caving systems, there are few mines in Japan where the conditions are suited to the square-set method.
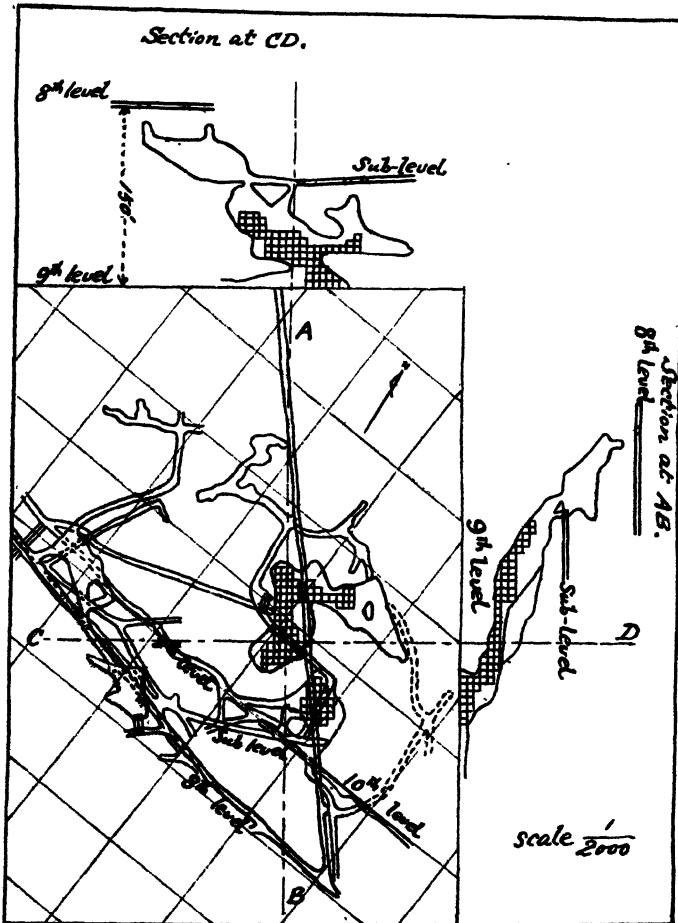
FIG. 2.—PLAN AND SECTION OF TENGU KAJIKA, SHOWING SQUARE-SET WORKINGS.

THE ASHIO DRILL

The excavation is by hand and rock drills. There are several kinds of rock drills, which have been in use for many years. In fact, it will be noted that the Ashio mine was the first one where machine drills were extensively employed in Japan. Among them, Leyner Ingersoll No. 26, Ingersoll stoper B. C. 21, Jackhamer B. C. R. W. 430, and Ashio drill are chiefly used at present, the latter being designed and manufactured by the engineers of the mine. Its construction is much like that of the Jackhamer, having a butterfly valve, but it is much simpler and weighs only one quarter as much. The following table gives comparative figures:

	JACKHAMER	ASHIO
Length, inches..........................	18	11.833
Weight, pounds..........................	40	11.5
Cylinder diameter, inches...............	2.25	1.625
Diameter of steel, inches...............	0.875	0.75
Performance per minute, inches..........	9	5.5
Cubic feet free air per minute..........	69	50

FIG. 3.—GENERAL VIEW OF ASHIO SMELTER.
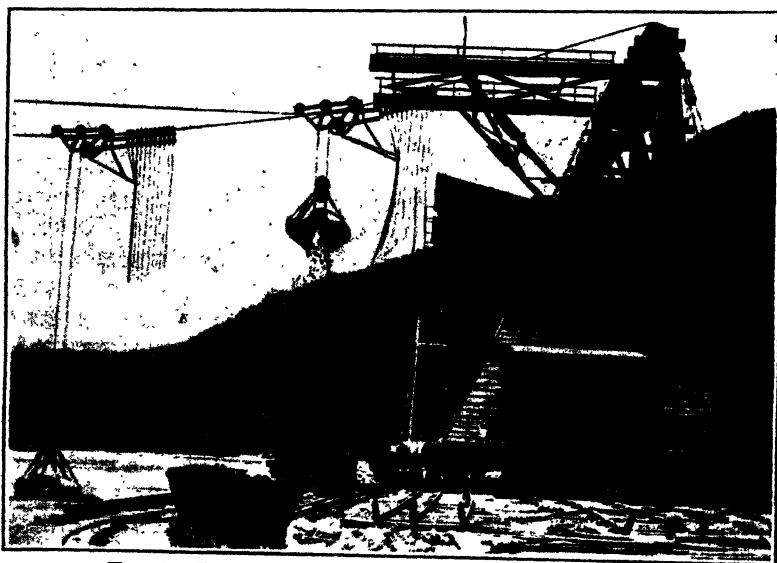
FIG. 4.—BUCKET DREDGER IN GIRIMIKI SETTLING POND.

Although this drill is not suitable for very hard rock, it is profitably used for soft ores, such as that of the kajikas, and also for the stoping of narrow and poor veins where hand drilling is too expensive. The cost of breaking ground by the Ashio drill, including wages, explosives,

compressed air, etc., ranges from $1.50 to $2 per 10 cu. ft., depending on the nature of the material broken.

In Conclusion

The Ashio mine has an assured tonnage of ore for many years to come. None but those who have seen the kajikas can realize the size and richness of the ore which has been found since the "Senjojiki" days. Rich ore is encountered constantly at every turn. News of the new find leaked out to the public at about the time of the terrible crisis that for a while threatened to shatter the financial structure of Japan's national economy. The appearance of the new bonanza thus resulted in the effective demonstration of the strength of the Furukawa firm, and enabled it to continue producing when the copper market had sunk to the bottom of depression and when the cost of production at several other mines was far above the market price for copper.

In the middle of January, 1921, when the writer had the privilege of welcoming Sherwood Aldrich, president of the Ray Consolidated Copper Co., at Tokyo, he referred to the formation of a Japanese copper producers' association with a view to curtailing output and warding off suicidal productions and competition, and then expressed the hope of seeing a new Ray in the Land of the Rising Sun as well as in the Mighty Country of Stars.

Japan owes a vast debt to America in mining and metallurgy, since the works in Japan were modeled after those of the United States. It is the earnest desire of the writer that Japan shall always preserve a sympathetic peace in copper matters as well as in industry and commerce at large.

DISCUSSION

HORACE V. WINCHELL, Los Angeles, Calif. (written discussion).— This property is noteworthy because of both its antiquity and its modernity. It was discovered and worked, I believe, about 320 years ago. To the Japanese, surrounded by monuments and structures of hoary age, such workings call for no particular attention, but to American engineers without foreign experience a mine with a producing record of more than a century is a phenomenon. On the other hand, under the progressive management of the author, and others of our Japanese members, the property has been extensively developed and equipped with every modern appliance so that it represents the most refined stage of mining, milling, smelting, and refining methods of today.

Geologically, it does not resemble any copper deposits with which I am familiar but possesses several unique features. The occurrence of the

veins in the neck of liparite which has the shape of an inverted cone, vary-
ing in diameter from about 2 mi. upon the surface to perhaps less than
2000 ft. at the lowest developed level, is only one of its special features.
This liparite, which is roughly circular in horizontal cross-section, con-
tains a marginal zone around its periphery in which are found innumer-
able fragments of included "Chichibian" slate of all sizes from minute
particles to masses as large as a city block. The famous kajika deposits
represent fragments of the slate in all stages of replacement by chal-
copyrite. Where the replacement is complete, the bedding planes are
obliterated. Where it is only partial, the banding can still be observed
and the tenor in copper is proportionately less. We thus find kajikas of
all sizes and all degrees of copper content, up to that of pure chalcopyrite.
The kajikas have a sporadic distribution usually within a distance of 500
ft. from the marginal contact of the liparite with the slate and quartzite;
they are not in the veins nor apparently genetically associated with them
and hence are subject to no rule of distribution nor discovery by the
application of any structural conditions. One kajika may contain 10
tons of 5 per cent. ore and another 1000 or 20,000 tons or more of 25 per
cent. ore. The occurrence of these peculiar deposits lends interest to
the work of prospecting but also introduces complications in the mining
methods, which have been well described.

Another feature is the fact that the veins are more numerous in the
upper levels. The mine has been developed by three tunnels, which enter
the mountain from different erosion valleys and which reach a depth of
about 2000 ft. beneath the apex of the mountain. From these tunnels
vertical shafts were sunk to a still greater depth of about 1600 ft. in
1917. There were at that time, I believe, seventeen timekeepers' offices
underground, each consisting of a large room excavated at a convenient
point, electric-lighted and furnished with conveniences for keeping track
of underground operations. There is an excellent system of electric
haulage and the tunnel entrances are supplied with cut-masonry arch-
ways suggestive of some of the best and oldest shafts and tunnels in
Pachuca and other old camps in Mexico.

In addition to a modern flotation plant, the tailings from which are
carried over the mountain by a bucket conveyor, this mine has a smelter
which is carefully watched under Government regulation to guard
against smoke damage. It has extensive hydro-electric power develop-
ments, for which purpose is utilized a waterfall with a vertical drop of
more than 1000 ft., and an electrolytic refining plant. The village of
Ashio has a population of about 30,000 and is beautifully situated, and a
visit to the property is rewarded by the opportunity of viewing beautiful
scenery and enjoying the courteous hospitality of the officials of the
Furukawa Co. My visit to this property will long linger in memory as
one of the most delightful of my experiences.

D. W. BRUNTON, Denver, Colo. (written discussion).—To the members of the Institute who were fortunate enough to visit Japan in 1911, this paper will be a pleasing reminder. At that time the railroad had not been built, so we made the trip on horseback over the mountains and down the valley of the Watarso River to the mines. The views along the valley and at Ashio resembled closely the scenery on the Roaring Fork of the Grande and Aspen, and at almost every turn someone would call attention to some striking resemblance to Colorado's most interesting and beautiful mining camp.

The hospitality shown left us little time and less inclination for detailed mine studies, but we were profoundly impressed with the immense size of the orebodies and their wonderful richness. More striking, however, were the evidences of careful thought and study in the selection of the mining and metallurgical equipment of the property. Order, system, and superlative cleanliness were evident everywhere, both on the surface and underground. Development was well ahead of extraction in all parts of the mine and I have never seen better work in drifting and stoping nor cleaner workings. The machinery at the mines and smelter was of the most improved types, many of the mechanical devices being especially designed to meet local conditions. Every provision was made for the comfort and safety of the workmen and the condition of the entire plant reflected the highest credit on both management and operatives.

MASAYUKI OTAGAWA (author's reply to discussion).—Although the writer did not intend to describe, under the present title, the milling and smelting processe adopted at the mine, the property is equipped with every modern appliance in these respects as stated by Mr. Winchell. Until a half century ago, all mines in Japan were worked by the Government in an uneconomical way. In 1877, I. Furukawa, the founder of the company, undertook to work the Ashio copper mine, and proved that it could be worked economically and successfully by a private corporation. Early in 1893, he introduced the bessemer converter, when C. Kimura was the general manager; this installation of the bessemer process in a copper plant was next to the Butte installation, so far as satisfactory operation was concerned. Also the ore has been treated by the flotation process since 1913, with improved extraction and reduced cost of milling.

Since the kajika deposits have been extensively developed and worked, the milling and smelting processes have been advantageously affected, because the rich and compact nature of the ore simplifies the operations in dressing and reduces the quantity of fuel and flux required in smelting.

In addition to the typical kajika deposit in liparite, there are kajikas that replace quartzite; and this deposit is considered very important for the development of the mine. The quartzite found in the upper

reaches of the Watarase River runs along the bed down to Sunokobashi where it crosses the river and extends to the Kotaki mine. (*vide* Fig. 1.) The width of quartzite ranges from 30 to 60 ft., and the kajikas are found at the fracture, where the former is crumpled by faults. This type of kajika consists of chalcopyrite and is usually associated with pyrite; bornite and chalcocite sometimes occur. It is specially characterized by its high content of copper, showing 60 per cent. for the best specimen and averaging 15 per cent. generally.

While in the United States the problem of injury to vegetation by smelter fumes was not given much consideration at the time, as early as 1897 the famous investigation of the mineral poison problems of the Ashio copper mine was carried out by the Government Committee. The engineers of the company had to pay attention to the mine-water and smoke problems to prevent any damage they might do, and installed several plants, in addition to various construction and forestry works, in order to comply with the Government regulations and special orders in accordance with the reports of the government investigators.

For the treatment of the smelter fume, in 1897, the company built the desulfidizing tower, which was replaced, in 1915, by a large dust chamber using the Roesing wire system. The gases passed from the chamber to four chimneys and were diluted by the forced admission of fresh air and then discharged into the atmosphere. In 1916, the Cottrell electrical precipitation process was introduced. In this way, besides the abatement of the smoke nuisance, arsenic and bismuth are being recovered.

As described by Mr. Brunton, when the Institute party visited the mine in 1911, the horse haulage and aerial tramways were mostly used for surface transportation, although the electric locomotives were employed for main adit haulage. In 1913, the Ashio railway was built, which connected the mine with the metropolis. The late Dr. R. Kondo organized and presided over the Ashio Railway Co., and he was succeeded by the writer, who was the president of the company until the railway was purchased by the Imperial Government in 1918.

Steam-shovel Operation at Bisbee, Arizona

BY H. M. ZIESEMER* AND GEORGE MIEYR,† BISBEE, ARIZ.

PRIOR to 1909 that mountain of porphyry known as Sacramento Hill had hardly been touched though it had always been thought to contain ore. During that year, exploratory work was started by underground methods; drifting and raising continued until 1913. It was proved that a mineralized zone existed, the limits of which had not been determined; so, because of the lower costs of exploratory work and the greater desirability of vertical sections, it was decided to prove the extent of the ore zone by churn drills. Drilling continued until 1917, when it seemed almost certain that the horizontal limits of the orebodies had been determined. Ore continued in depth in one direction beyond a point where it was economical to explore by churn drilling; this extension is still being developed by underground methods. The irregularities of the boundaries of the ore zones, together with the variable grades of ore encountered, made it necessary to drill on the corners of 100-ft. squares. Two ore zones were proved, known as the East and West Orebodies.

The Sacramento Hill porphyry stock intruded into the Dividend fault. The fault ran roughly east and west with a dip to the south, having pre-Cambrian schists on the north against Paleozoic limestone on the south. This intrusion was accompanied by a silica breccia on the south periphery, along the limestone contact side. The ore lay in the silica breccia and in the porphyry, some distance from the breccia. No ore was found in the heart of the porphyry mass on the north side, along the schist contact.

The West Orebody is elliptic in shape, having a major axis of 1200 ft. (366 m.) and a minor axis of 800 ft. (244 m.); these diminish with depth until, at 300 ft., the axes are 500 ft. and 300 ft., respectively. The ore consists of chalcocite, some bornite, chalcopyrite, and considerable pyrite in a siliceous gangue. About 30 per cent. of the copper is caused by

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enrichment of the pyrites. The estimated tonnages of the West Orebody are:

	Tons	Per Cent. Copper	Grades, Per Cent.
Smelter ore	1,296,120	5.960	3.50 and higher
Milling ore	6,557,640	1.525	0.80 to 3.50
Total	7,853,760	2.257	

In determining the method of mining to be adopted for the West Orebody, costs and information relative to caving, shrinkage, slicing and open-pit mining were obtained from the different mines of the Southwest, where these methods had been worked successfully on various types of low-grade deposits.

Should the Ohio-Inspiration caving method be adopted for mining the West Orebody, it was estimated that it would cost 33⅓ per cent. more than the costs obtained at the Inspiration mine, where this method is used, because of the relatively small tonnage to be handled. The ore did not lend itself readily to the caving method because of the hard and soft strata, which would increase the cost of repairs or losses of the small finger raises. From results obtained by the caving method up to that time, it was estimated that only 90 per cent. of the copper out of 100 per cent. of the tonnage in reserves was being recovered, due to the dilution from the waste capping. A characteristic of the ore in Sacramento Hill was its irregular surface; therefore, it could not be expected that even 90 per cent. of the copper contents would be recovered, particularly as much of the ore against the capping is of a higher grade than the average. Neither did the method lend itself to a clean separation of the direct smelting ore, which occurred in pockets and stringers throughout the orebody, from the lower grade milling ore. The same conclusions were drawn with reference to the shrinkage system of mining, so it was decided that these methods could not be considered.

There remained for consideration the underground slicing and the open-pit steam-shovel methods, both of which appeared favorable. By either method, the orebody with its irregular outline, having high-grade stringers running into the capping, could be recovered; and both gave an opportunity for fairly close sorting, on a large scale, of the different classes of ore which occurred irregularly throughout the orebody. A comparison of net returns from the two methods showed in favor of the open-pit mining. The estimated cost of mining the orebody by slicing was based on figures obtained from the Miami Copper Co., where this method had been used on a large scale. Costs from the Chino Copper Co., Utah Copper Co., and the Nevada Consolidated Copper Co., where steam-shovel mining was carried on, were used as a guide in arriving at an estimated cost for open-pit work. The estimated profits from steam-

shovel mining were based on sending ore to the mill with a minimum grade of 1 per cent. copper. There remained the possibility of sending lower grade material to the mill, thereby increasing the profitable tonnage while the total cost remained the same. Comparative estimated costs were based on the 1915 scale of wages, with the probable chance for an increase, which would affect the slicing cost more, in proportion, than the open-pit cost, as only about one-half as many men would be needed for the open-pit mining as would be required for the slicing method, to produce the same tonnage. For the open-pit mining, about two-thirds of the men employed would be unskilled laborers. This would work to the advantage of open-pit mining, should a shortage of skilled labor occur for the slicing method.

In the early stages of the slicing method, when men are working for costs and to put the mine in shape to deliver the required tonnage, small stringers of good ore which run into the capping might be passed by. This would not be the case with steam-shovel work, where it is necessary to remove all material. A few small stringers of good ore left unmined would soon amount to a 100-ft. cube, or 85,000 tons, meaning a considerable loss. If the tonnage of ore within the orebody proved to be greater than that estimated, the total cost of mining the orebody by slicing would be increased by the actual cost of placing the increased tonnage at the mill or smelter, while with the open-pit work the total mining cost would remain the same.

In all underground caving methods, it is anticipated that the surface capping will follow directly upon the removal of the ore. The hard capping lying above the ore in Sacramento Hill made it extremely uncertain whether the capping would follow or not. In case it should not follow, there would be the possibility of serious accidents or an additional cost to remedy the difficulty.

The open-pit mining allowed a greater elasticity with less cost, for increasing or decreasing, on short notice, the tonnage desired. And, should a complete shutdown be necessary during the period of operation, as has actually occurred, with the underground method considerable expense would be incurred in closing the mine, keeping it in repair while closed, and reopening it, that would not be incurred were the open-pit method used. All the unknown chances appeared to favor open-pit mining, but the difference in estimated costs between the two methods was not great, and any errors in estimates might affect costs and the net profit based on either method. Before a final decision was reached, L. E. Foster, formerly assistant superintendent of the Chino Copper Co., at Santa Rita, N. Mex., was called in as consulting engineer and, after his report and recommendations were made, it was definitely decided to mine the West Orebody by steam shovel. Plans of attack were laid out and the order for equipment placed.

The plan of opening the pit might be divided into three stages: Work located between the 5360 and 5656-ft. elevations (all elevations refer to sea level), the 5180 and 5360-ft. elevations, and the 4940 and 5180-ft. elevations.

The work between the 5360 and 5656-ft. elevations consisted in attacking the hill from the side and removing the apex down to an elevation of 5480 ft., and below that elevation to the 5360-ft. elevation, removing a semicircular mass in the hillside; between the elevations of 5180 ft. and 5360 ft., to divide the pit in half by through-cuts, removing one half through east approaches and the other half through west approaches; below the 5180-ft. elevation and down to the 4940-ft., the bottom of the pit, to carry a spiral track on the rim on a 2½ per cent. grade.

FIG. 1.— SOUTH SIDE OF SACRAMENTO HILL BEFORE OPERATIONS WERE STARTED.

It was on the first stage, between the elevations of 5360 (1634 m.) and 5656 ft. (1724 m.), where the greatest difficulties were encountered. Fig. 1 shows a south view of the Hill before operations were started and Fig. 2 shows a north view. These views show the location of the hill with reference to the civic improvements—homes, railroads, public highways, and mining property. The company's shops and other buildings were located on the south rim and had to be removed for the pit operations. The top of the hill, with its rough, jagged, and precipitous outline consisted of an iron-stained siliceous breccia, or the contact material between the limestone and porphyry. The porphyry, on cooling, had caused this mass to become very badly fractured. Before any track could be constructed on the hill, it was necessary to blast down this apex. The broken material served as a fill for railroad beds and formed a slope from which shovel operations could be started. In many places, the hill was too steep to

allow tracks to be constructed for shovel operations without burying the loading track. Fig. 3 shows the point to which this work was carried

FIG. 2.—NORTH SIDE OF SACRAMENTO HILL BEFORE OPERATIONS WERE STARTED.
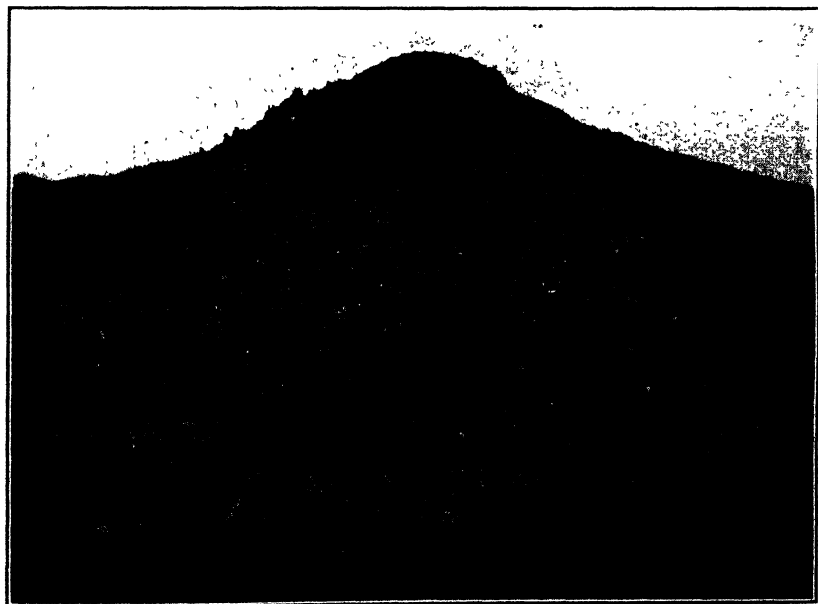
FIG. 3.—SHOWING METHOD OF ATTACK.

This work was accomplished by working up from the bottom, drilling and blasting each exposed face of rock. Compressed-air machines

were used and holes were drilled horizontally 15 to 20 ft. (4.6 to 6 m.) deep. As each series of shots was fired, the drills were moved up on the broken rock lying against the remaining cliff and the cycle repeated. When the top was reached, a number of churn-drill holes, 40 to 85 ft. (12 to 26 m.) in depth, were loaded and shot with the last series of machine-drill holes, using about eleven tons of powder. This practically completed the slopes, except for shooting down certain points and breaking up the many boulders in the mass of broken rock.
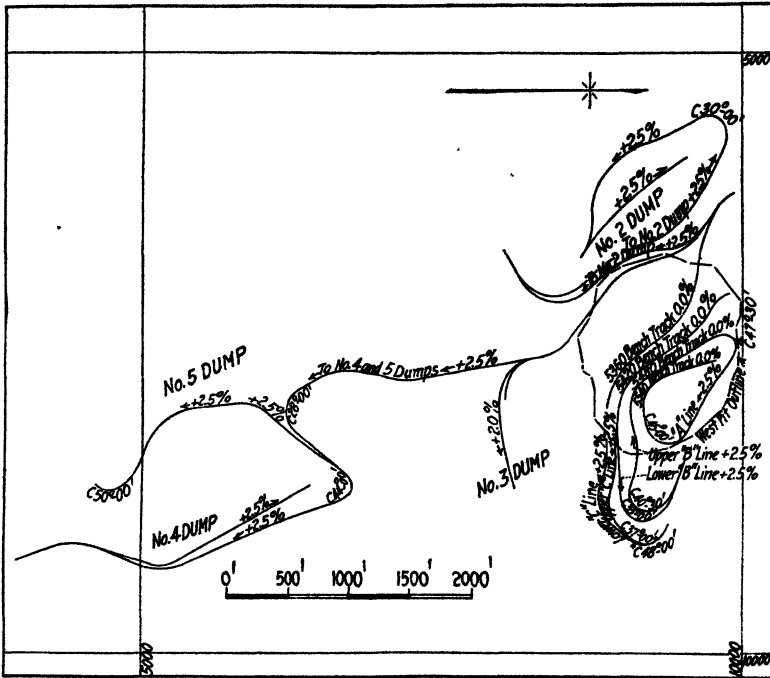
FIG. 4.—PRELIMINARY TRACKS, 5360-FT. TO 5540-FT. BENCH.

Fig. 4 shows the track system to be put in for the first stage of operations. In deciding on 60-ft. (18-m.) benches for this stage of operations, common practice at the various steam-shovel operations in the Southwest served as a guide. It has since been proved that 30-ft. benches would have been more economical with rock of the character encountered. The track runs level across the 5360-ft. bench and then starts up on a 2½ per cent. grade to the dead end of a switchback between the upper and lower C line tracks. From there, it climbs again on a 2½ per cent. grade until it reaches the pit line on the 5420-ft. bench, from which the track is run out level for this bench. From that point, it continues on a 2½ per cent. grade over the upper and lower B line switchback to the east edge of the pit, where it runs level across the 5480 ft. bench, continuing on a 2½

per cent. grade, making a complete circuit of the hill, reaching the 5540-ft. bench at a point on its western edge.   Most of this line was roughly graded by hand, closely following the irregularities of the ground.   The alignment was later made by fills.   Some curves ran as high as 65°, being later decreased to 40° curves for operations.

To establish the 60-ft. benches it was necessary to make a cast through on the 5360-ft. bench to make room for a loading track.   Then a second cut was loaded out; this made room to catch the cast from the bench above.   This cycle was repeated on the 5420-ft., 5480-ft., and 5540-ft. benches, see Fig. 5.   In working the 5540-ft. bench back, the danger was
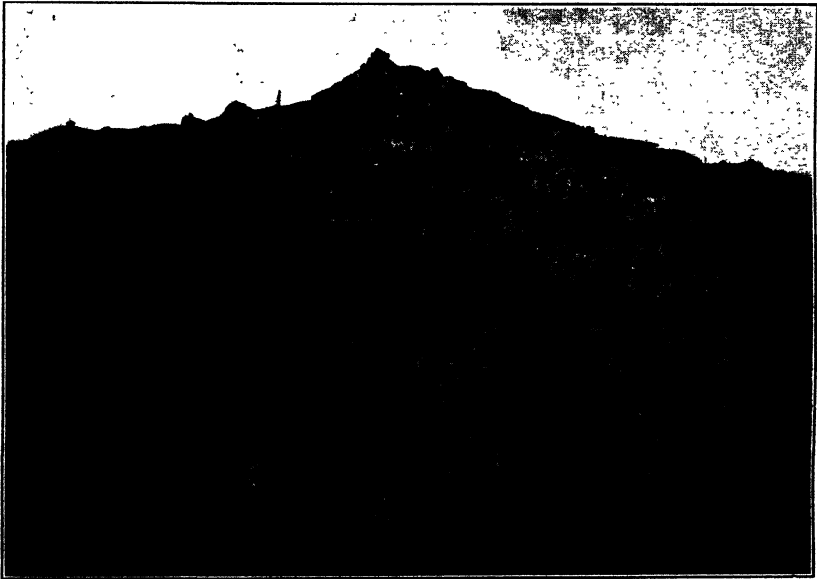
FIG. 5.—SHOWING METHOD OF ATTACK.

not as great when advancing against the slope of the hill as when the slope of the hill receded from the advance.   In working against the slope, it was possible to blast with fairly heavy charges without danger to life and property on the north side of the hill, but when the slope receded from the direction of advance, there was always the danger of boulders being dislodged from the hillside or being thrown by the blast amongst the traffic and homes located below.   It was necessary to undercut the toe of the bench by short air-drill rounds and put in plugger drill rounds, from the slope side of the top edge, in order that all material would fall toward the bench side.   This work was carried to a point, where it became necessary to stop as any blast might dislodge rocks that would endanger life and property below.   The comb of rock thus left was taken down with

the bench below, as it was worked back in the manner just described. The only serious accidents occurred from premature explosions, caused by flinty particles raveling from the sides of the holes during loading, falling down the hole and igniting the powder.

The disposition of waste in the early stages of operations proved very expensive. The only available areas for dumps near the shovel operations were located on hillsides. These areas were reached by adverse grades of 2½ per cent. and track alignments having high curvatures. The location of these waste disposal areas is shown in Fig. 4. Though the hauls were short in comparison to a location having favorable grade

FIG. 6.—SHOWING PROGRESS OF WORK.

to the loads and much greater distance, haulage costs proved greater, because of the small number of cars that could be handled at one time. This necessitated more trains being kept in service to supply the shovels.

Operations on the first stage, between the 5360-ft. and 5656-ft. elevations, have removed, to date, all material above the 5420-ft. bench and have advanced the 5360-ft. bench in the clear for the bench below. On the second stage of opening the pit, between elevations of 5180 and 5360 ft. enough material has been removed to allow down-cutting operations to start below the 5270-ft. bench. This is the objective point to which the pit had to be opened in order to allow the required tonnage for the mill to

come out, see Fig. 6. The general arrangement of tracks, locations of
the various dumps, and mill site are shown in Fig. 7.
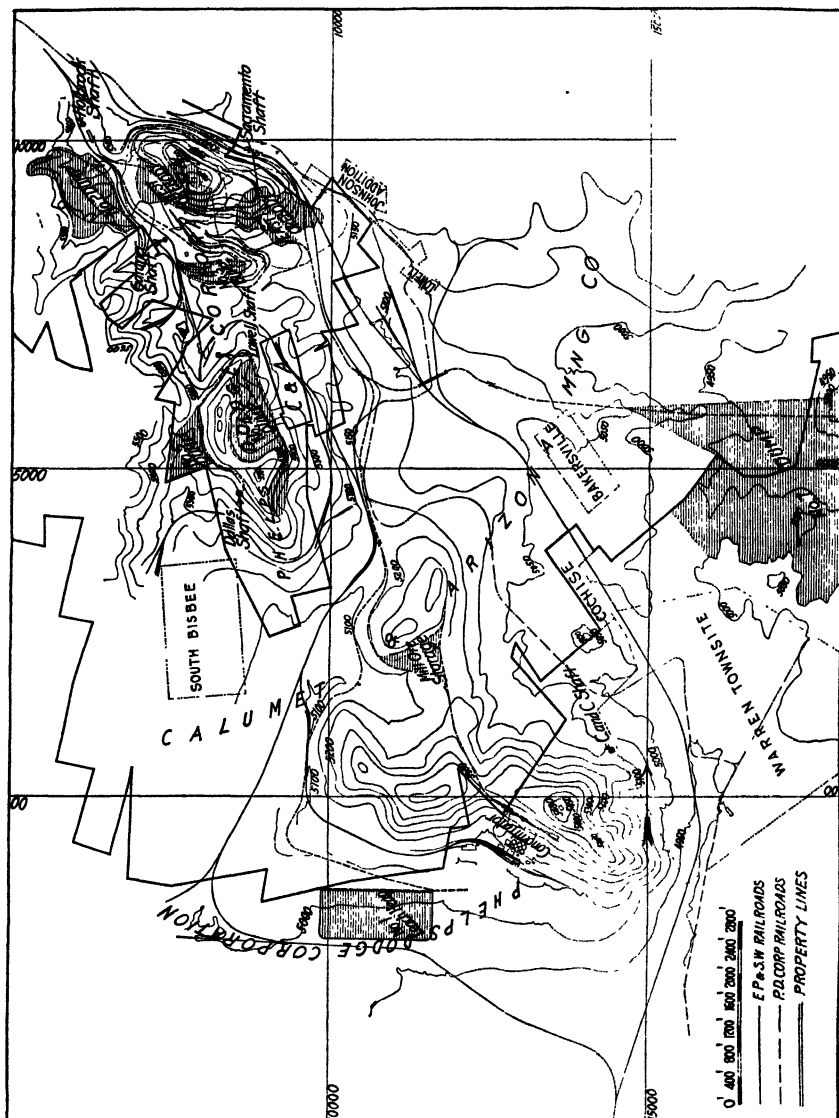
FIG. 7.—GENERAL ARRANGEMENT OF TRACKS, DUMPS, ETC.

STEAM-SHOVEL OPERATIONS

Steam-shovel operations may logically be considered under the follow-
.ing major subdivisions: Breaking ground, shoveling, haulage, dump
maintenance, and sampling. The following discussion of methods and

costs will follow this outline, with a brief description of equipment used in the operation.

Breaking Ground

Drilling.—In breaking the material for rapid handling by the shovels a combination of bank and toe holes is used to prepare the ground for the powder charge. The bank holes are put down by churn drills. Collar

A.—BANK HOLES
B.—TOE HOLES
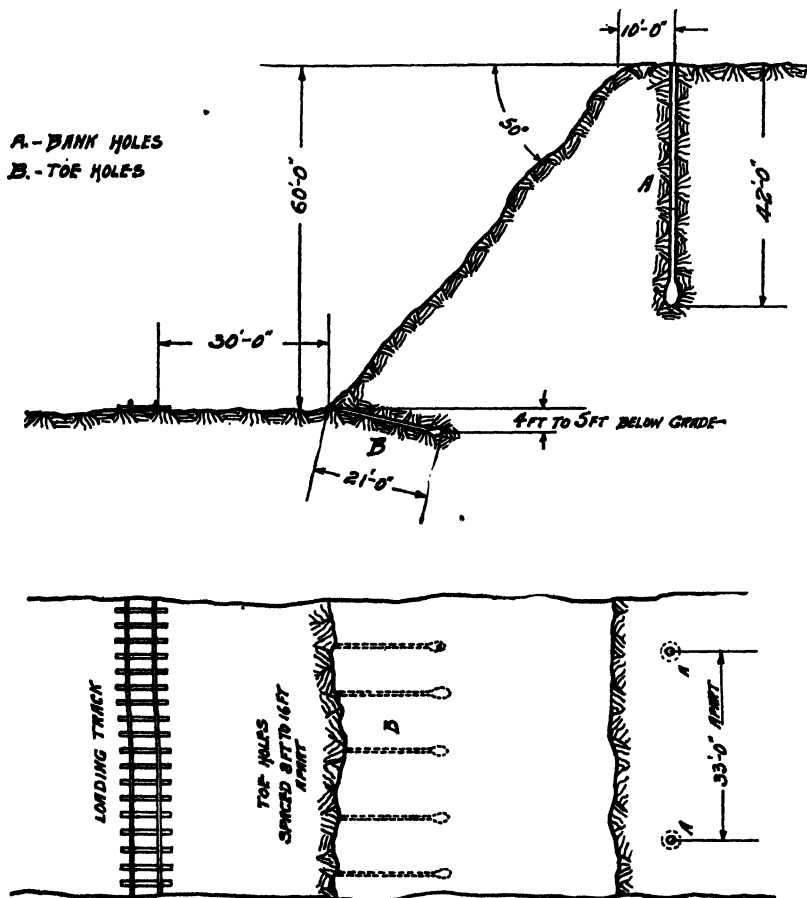
Fig. 8.—System of Bank Holes.

casing, in lengths of 5 to 8 ft. (1.5 to 2.4 m.), is always used and it is occasionally necessary to case 20 ft. (6 m.) or more when the material tends to ravel. The casing is driven with the string of tools. The toe holes are put in with the Burley machines. The average footage per shift for the air drills, in all classes of ground, is 27 feet.

The arrangement of the holes on established benches of 60-ft. height is given in Fig. 8. The system is not rigidly followed, the holes usually

being placed to conform with the shape and hardness of the bank.   With
the system illustrated, the toe holes are depended on to give the required
bench advance at the toe and to maintain a level bench floor.   The func-
tion of the bank holes is to slope the bank, lessen the burden on the toe
holes, and to make the bank safe for the shovel to go under.   A slope of
50° is considered desirable and is maintained as nearly as is practicable.
In breaking the ground for undercutting to establish the 5270-ft. bench
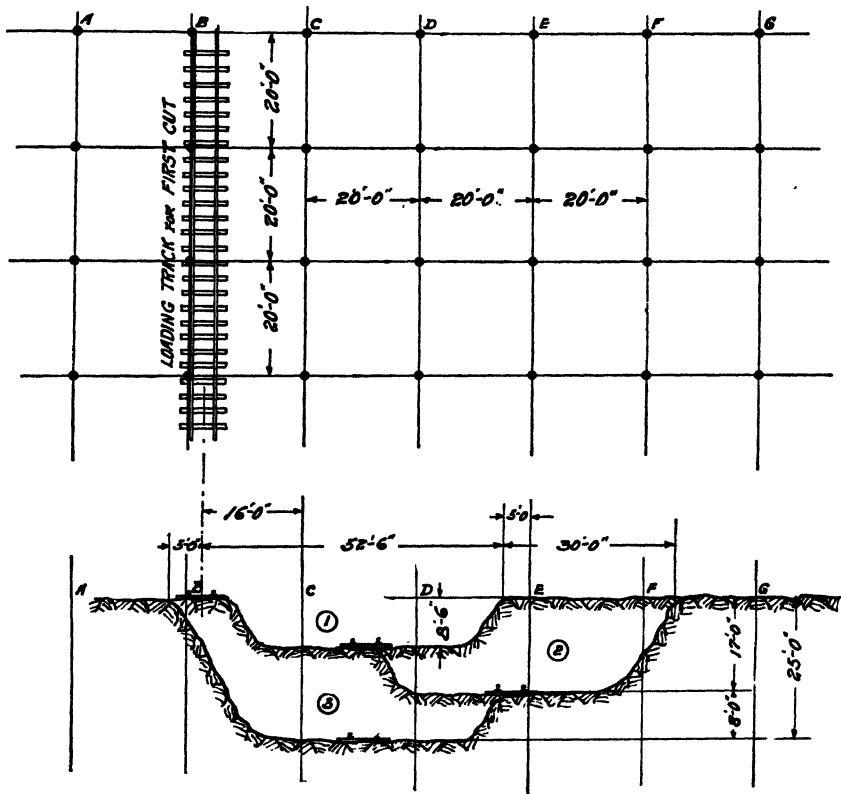
FIG. 9.—SYSTEM OF UNDERCUTTING.

45 ft. below the 5315-ft. level, the work was done in sections of 25 ft. and
20 ft.   An illustration of the 25-ft. section will describe the method used.
    First, three rows of churn-drill holes C, D, E, Fig. 9, were put down
5 ft. below grade, or 30 ft., at regular spaced intervals of 20 ft. each way.
The loading track for the first downcut was placed, approximately, 16 ft.
from the C row, or just far enough away not to be disturbed by the blast-
ing.   These three rows of holes were then blasted and the first cut, approxi-
mately 8 ft. 6 in. in depth and close to 50 ft. in width (this being the
maximum depth and width of cut possible with the shovel) was taken.

The loading track was put in the bottom of the first cut and the second 8 ft. 6 in. was taken. Then row *B* was drilled and blasted and, with the loading track in the second cut, the third drop cut was made. In order to allow time for drilling the next section, a row of holes *A* and *G* was put down and shot and a cut taken on each side of the third cut.

In the undercutting just described, practically all the powder was confined in the bottom of the holes; only enough powder was placed in the barrel, about 12 ft. from the top, to shatter the collar. Excellent results were obtained by the use of this method; 2.19 solid yards (1.8 cu. m.) per pound of powder were secured in breaking the material and 1144 solid yards per 8-hr. shovel shift were loaded in moving a total of approximately 450,000 yd. (344,000 cu. m.).

It has been the general opinion that shooting the full depth of two or more cuts is wasteful of powder and that the lower cuts would be so consolidated by the time they were reached that it would require a second shaking up with the powder before shoveling. It has been demonstrated that, in certain cases at least, the powder results are satisfactory and that the ground was well prepared for the shovel. The advantage of shoveling in eliminating delays is obvious and of great importance.

The following table gives the average yearly performance of churn drills, in feet per 8-hr. shift, and cost per foot:

YEAR	FEET PER SHIFT	COST PER FOOT
1918.....................	12.5	$1.74
1919........	20.0	1.14
1920....................	24.2	0.94
1921 (9 mo.)	31.2	0.55

Blasting.—Bank holes are sprung heavily to take a charge of from 300 to 800 lb. (136 to 363 kg.) of powder, depending on the nature of the ground; Cordeau Bickford, an exploding fuse, is used as a detonator. The end of the fuse is weighted and lowered to the bottom of the hole before the charge is put in and about 1 ft. of it is allowed to project from the hole. The charge is then poured in and the hole filled to the collar with screened dirt. The fuse is set off by a No. 8 cap on the exposed end, detonated by a hand dynamo, or blasting box.

The toe holes are sprung to take a charge of from 75 to 150 lb. (34 to 68 kg.) of explosive and are loaded and fired in strings, the number of holes depending on conditions. Loading has been made safer and faster by the use of brass tubes through which the powder is introduced; the tubes are removed before blasting. Two electric blasting caps are placed in each charge, each separately connected in parallel to a pair of heavy lead wires. The holes are tamped with fine material. The charge is fired by electricity from the regular power circuit on the hill.

Under the system described, the loading track is laid in to an average distance of 30 ft. (9 m.) from the unbroken toe before the toe holes are fired. If the toe-hole blast is normal, the broken material will fill the pit without covering the track. The shovel first cleans up the toe-hole material, then it advances and faces up the bank as far as it can reach until the jackarms are opposite the bank holes. It then backs up 40 or 50 ft. and the bank hole is shot into the pit. The shovel then advances again, cleans up the muck from the bank-hole blast, and trims the bank. The work thus progresses throughout the cut, the toe holes being fired in strings ahead of the shovel and the bank holes one at a time into the pit. Considerable shovel time is lost when blasting bank holes, an average of 40 min. being required for each blast.

The method just described is the one also used on the 45-ft. benches, the bank or slope holes being put down between 27 and 30 feet.

On the 30-ft. benches, toe holes will be eliminated and bank holes, drilled 5 ft. below grade, will be used. These holes will be drilled and shot in series, well in advance of the shovels. This method will give good powder results and will avoid shovel delays due to blasting, which has been one of the most serious problems to contend with on the higher banks.

In general blasting operations, it may be said that distributing the powder uniformly through the ground would make finer breaking and would save delays in shoveling. However, the comparative cost of drilling and loading holes bored by churn drill or air drill must be balanced in deciding on a method of shooting.

The height of bank affects both methods of shooting and shovel results. High banks are to be avoided and, if the ground is hard, 30 ft. is the limit; softer banks may be permitted at greater height. In shoveling, the work is safe with a bank that can be reached and trimmed with the dipper. It has been demonstrated that with a 30-ft. bank more cars can be loaded for every 6-ft. advance of the shovel than on a 60-ft. bank. Also, if there is a mixture of ore and waste, sorting can be better done on lower banks.

In the above work, a gelatine 40 per cent. powder, 1⅛ by 8 in. sticks, was used in springing holes and shooting boulders. Both nitro-starch and ammonium-nitrate powders have been used in bank and toe holes. The selection of a powder to use depends largely on the nature of the ground and the results obtained, in terms of yards broken per pound and cost per yard. The following figures give the powder efficiency, in terms of solid yards per pound of powder used, and the cost per yard broken:

YEAR	SOLID YARDS PER POUND	COST PER YARD
1918	1.43	$0.158
1919	1.72	0.115
1920	1.97	0.100
1921	2.12	0.101

Shoveling

The operating crew consists of nine men: engineer, craneman, fireman, and six pitmen. The loading capacity of the shovels in favorable material has never been reached. The record 8-hr. shift is 124 cars, or about 2350 solid cubic yards. The following table gives the average yards per shovel-shift, gallons of fuel oil per yard, and the cost per solid yard:

	1918	1919	1920	1921
Solid yards per 8-hr. operating shift........	582	828	769	1038
Gallons of fuel oil per solid yard........... ..	0.81	0.65	0.60	0.56
Cost per solid yard........................	$0.236	$0.181	$0.157	$0.130

Haulage

The broken material is separated into four products, as follows: Direct smelting ore, milling ore, leaching ore, and waste. To date, waste has been the principal material handled, although small tonnages of direct smelting ore have been shipped to the smelters at Douglas, about 180,000 tons of milling ore have been placed in stock pending starting of the concentrator, and 375,000 tons of low-grade ore have been placed on the dump for heap leaching.

In the earlier stages of the operation, it was necessary to locate waste dumps on adjacent hillsides. Although the distance was comparatively short, the haul was against 2½ per cent. grade and proved expensive. Later, plans were made to establish a dump-site north of the town of Warren, which would provide ample room for all future waste. Although this site necessitated a longer haul, the grades were favorable to loads. Nearly all of the waste was taken to this dump in 1921, at a material saving in transportation cost, as indicated by the comparison shown below. After the concentrator starts, the smelting and leaching ores will be taken to the crushing plant, located at the concentrator, to be crushed before leaving for their respective destinations.

The following table gives the cubic yards hauled per locomotive shift, the gallons of fuel oil used per yard, and the cost, including the maintenance of the dumps:

	1918	1919	1920	1921
Solid yards per 8-hr. operating shift...........	237	338	361	450
Gallons fuel oil per solid yard................	1.45	0.95	1.03	0.87
Cost per solid yard........................	$0.333	$0.239	$0.254	$0.177

Dump Maintenance

In the early stages of operation, the dumps located on nearby hillsides were comparatively small, necessitating frequent moving of tracks and consequent excessive labor costs. With the establishment of the large

dump near Warren, which will have a face approximately ½ mi. (0.8 km.) in length and an average depth of 100 ft. (30 m.), the cost of dump maintenance should be materially reduced.

Sampling

The banks are sampled in 25-ft. blocks along the toe, in advance of the shovel, to furnish some estimate of the probable grade of the material, and to aid the sampler in making the separation. The critical samples are taken from the cars while the train is loading at the shovel. The sampler, an American, works on top of the train, using a round-pointed shovel and throwing the sample into a portable bin on the ground beside the track. A Mexican helper breaks up the sample and shovels it through a 1½-in. splitter with three banks, which reduces it to one-eighth its original size. Two bins are provided so that the work can be continuous. When the train is ready to leave the shovel, the sample has been taken and it is only necessary to shovel the sample from the last car through the splitter, sack it, and send to the assay office. About 500 lb. (227 kg.) of sample are taken from each train of six cars, of which about 60 lb. (27 kg.) reach the assay office.

The assay plant is built on two cars, so that it can be moved easily and kept as near as possible to the ore shovels. One car contains the crushing plant: jaw crusher, rolls, splitter, electric hot plate for drying, a disk pulverizer, and a fan with pipe connections to take all the dust from hoods over the machines and to the outside. The other car contains the laboratory equipment: balance, hot plate, sink, filter stand, and colorimetric machine. The cars are both the same size, 12 by 26 ft., and there is ample room for the plant described.

All assays are by the colorimetric method and only coppers are run. Each day composites are made up of all the ore samples and sent to the main laboratory to be checked by the KI method. On the average, results from the two methods agree within 5 per cent. on the total copper. The assay plant is operated by one man who does all the bucking and assaying and exercises a certain amount of control over the trains.

Material loaded by the shovels is classified and separated into four products: smelting ore, milling ore, leaching ore, and waste. With the aid of the toe samples, the smelting ore and most of the milling ore are recognized by the sampler on sight; and in all cases where he is sure he gives a train disposition ticket to the brakeman when the train leaves the shovel. The dividing line between milling ore and leaching ore, and that between leaching ore and waste, however, cannot be determined with sufficient accuracy by sight alone. If the sampler is not sure, he does not give the brakeman a disposition ticket, but marks his sample ticket "set out," rushes the sample to the assay office, and leaves it up to the assayer. All trains pass over the Lowell switch on the way to the dumps.

If the brakeman has no disposition ticket, the train is held on a siding until released by the assayer after the sample has been assayed. The switchtender at Lowell then issues the disposition ticket, without which the train cannot be dumped, and the train goes to the proper dump. Actual delay to "set out" trains at Lowell will average about 11 min. and all errors in disposition can be avoided.

EQUIPMENT USED

Churn Drills

Seven Sanderson Cyclone drills, having a spudder motion, are in use. They are driven by single-cylinder gasoline engines rated at 18 hp. and are self-propelling. The crown sheave is 32 ft. above the ground and the maximum combined length of the bit, stem, and rope socket is 26 ft., allowing the bit to be lifted 6 ft. above the ground. The complete string of tools weighs approximately 1300 lb., and the maximum fall when striking is 28 in. Standard bits are of dumbbell section 5-in. material dressed out to give a 5.875 in. (149 mm.) bit of reamer type.

Air Drills

The equipment consists of fifteen heavy air drills, all Sullivan piston or "Burley" model with 3.25-in. (82.5 mm.) pistons, and a stroke of about 6 in. They operate on 85 lb. of air pressure supplied from the central power plant. Round steel, 1.375 in. in diameter, is used with a change interval of 2 ft. and a maximum length of 23 ft. Standard starters are 3 ft. long with a gage of 3.125 in., the gage decreases 0.1875 in. with each change throughout the set_of eleven drills, the smallest bit being 1.5 inches.

Plugger Drills

Eighteen Sullivan Plugger drills, Type D. P. 33, are in service for trimming banks, breaking boulders, and drilling high places in the floor of the pit.

Steam Shovels

Seven shovels are regularly used for loading the broken material. They are all the Bucyrus Co., Model 88 C, which are standard machines designed for such work. They weigh 103 tons, are equipped with 3.5-yd. dippers, locomotive boilers, and use saturated steam at 125 lb. gage pressure. The maximum lift is about 75,000 lb. Main hoist-engine cylinders are 12 by 15 in. and the swing-engine cylinders 8.5 by 8 inches.

Due to the hard ground and the high banks, considerable damage was done to the main frames, so all had to be reenforced over the front trucks. Also, the size of the A-frame legs was increased from 6 to 7 in.

It is proposed in the future to equip all shovels with locomotive-type superheaters.

Locomotives

Fifteen locomotives are in use, all manufactured by the Porter Co. They have four wheels, all drivers, a wheel base of 7 ft. 6 in., and are of the saddle-tank type. Three use saturated steam, weigh 106,000 lb., and can exert a tractive force of 20,436 lb. The remaining twelve have superheater equipment, weigh 109,000 lb. and have a tractive force of 23,063 lb. All use a gage pressure of 175 pounds.

Dump Cars

Seventy-five all-steel dump cars, built by the Western Wheeled Scraper Co., are in use. Fifty have a rated capacity of 20 cu. yd. and weigh about 55,400 lb. each. The remaining cars have a rated capacity of 25 cu. yd. and weigh about 60,700 lb. each. These cars dump and right themselves by compressed air, the door rising as the car dumps over, allowing the broken material to slide out underneath.

Water Softener

The question of suitable water for the boilers was important. At first, the city water was used without treating, but the resulting scale and sludge greatly reduced the boiler efficiency. The problem was satisfactorily solved by treating water from the Junction mine, containing 43 soap degrees of hardness, by the Reisert barium-carbonate process. The hardness was reduced to 7 or 8 degrees with practically all the sulfates removed and the product was a first-class boiler water.

Electric Welder

One of the great time and money savers on the job is the Westinghouse electric-welding outfit. This is a 60-volt, 300-amp., portable machine, using metallic electrodes. It is used for welding all kinds of breaks, such as engine frames, dipper bails, drawheads, swing circles, flues, etc., and in building up old dipper fronts. The results have been very satisfactory and the outfit is considered a necessity.

Powder Car

To provide a safe means of transporting powder from the magazine to the hill, an armored car built especially for this work is used. It is built of wood reenforced by 1/8 in. steel. It is 9 ft. 3 in. wide by 22 ft. long and is built on regular railroad-car trucks.

Wrecker

For lifting heavy castings, moving tracks, bad derailments, and for general steam-shovel repair work, a wrecker made by the Link-Belt Co. is used. It is self-propelling, has a 46-ft. boom, and has a lifting capacity of 20 tons on a 12-ft. radius.

GENERAL

In starting a large shovel job, heavy costs are to be expected while building up an organization; in this case, the troubles were increased by the shape of the hill and the difficulties of getting the first benches

Superintendent

Statistician

Operating Foremen
Bench Foremen

Master Mechanic
Repair Foremen

Blasting Operations   Dumps   Steam Shovels   Locomotives
Powder Foreman   Main Line Tracks   Engineers   Foreman
Air Drills Blasting Tool Shop   Roadmaster   Crane Men   Engineers
Boss Powdermen Sharpeners   Track Gangs   Firemen   Repair Men
Machine Men Laborers Helpers   Bosses   Pitmen   Hostlers
Helpers   Dump Men   Hostlers
Nippers   Bull Gang

Train Movement   Pipe Fitting   Bench Tracks   Churn Drills
Yard Masters   Boss   Foremen   Foremen
Laborers   Gang Bosses   Drillers
Chief Dispatcher Brakemen   Trackmen   Helpers
Repair Men
Dispatchers Switchtenders   Nipper

FIG. 10.

removed. The improvements in costs and increase in efficiency are due both to improvement in the physical condition of the ground handled and the development of a good working organization. The paramount factor in an organization is efficient coöperation. The spirit of coöperation and the incentive to efficiency have been given the foremen, bosses, and men by the posting of graphic charts giving the daily cost and labor efficiencies in all departments. Another factor is the labor bonus based on the yardage per man-shift whereby every employee benefits on a percentage basis. Fig. 10 gives the essential points of the organization and the following table gives the number of solid cubic yards moved per man-shift and the labor costs per yard:

	1918	1919	1920	1921
Solid yards moved per man-shift...............	7.84	11.74	12.02	15.28
Cost per solid yard, labor.....................	$0.590	$0.387	$0.419	$0.312

The following table gives the average yearly cost per solid yard:

	1918	1919	1920	1921
Breaking ground	$0.357	$0.239	$ 0.214	$0.173
Steam shovels	0.236	0.181	0.157	0.130
Haulage expense	0.333	0.239	0.254	0.177
Dump expense	0.040	0.031	0.031	0.023
Miscellaneous expense	0.008	0.008	0.027	0.016
Total	0.974	0.698	0.683	0.519
Monthly labor bonus	.....	0.004	0.023	0.029
Total, direct	0.974	0.702	0.706	0.548

The Aztec Mine, Baldy, New Mexico

BY CHARLES A. CHASE,* DENVER, COLO., AND DOUGLAS MUIR,† SIMON, NEV.

(San Francisco Meeting, September, 1922)

THE Aztec mine is not widely known, by reason of its isolation and the relative insignificance of its tonnage; financially, however, it has an enviable record and geologically it is extremely interesting.

The town of Baldy, Colfax County, the mine camp, at an elevation of 10,000 ft. (3048 m.) is 8 mi. west of Ute Park, the terminus of a branch of the A. T. & S. F. Ry. The mining tract was designated by the Maxwell Land Grant Co. as the Aztec Reservation, and lies on the east slope of Mt. Baldy, the summit of which, 12,491 ft. (3807 m.), is one of the highest in the state.

Rich copper float was found on the northeast slope of the mountain in 1865, and discovery work was begun the next year. At the same time, gold placers were found on Willow Creek, on the southwest side of the mountain, and the Elizabethtown settlement followed shortly. Gold washing lasted many years. In 1867, placer gold was found also on Ute Creek, the tracing of which to its source led to the locating of the Aztec mine by Lynch, Dogherty, and Fosley in the following year.

The early production came from the contact of shale and sandstone or from sandstone immediately above the contact. Exact record of production is lacking, but L. C. Graton[1] gives estimates ranging between $1,250,000 and $1,500,000, of which about $1,000,000 was taken out in the first four years.

After the exhaustion of the early bonanza, production continued intermittently until 1909, when the Maxwell Land Grant Co. undertook systematic development, through four adits at intervals of 75 ft. vertically or approximately 100 ft. on the dip of the contact, which was thought to contain generally a quartz-pyrite vein. The two intermediate adits almost immediately reached ore, the most southeasterly in Fig. 2. This ore, however, was not mined until 1911–12; its grade was about $20 and

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it was mostly quartz. Adits No. 2 and 3 were discontinued after the ore was found and Nos. 1 and 4 were then driven.

Late in 1911, the southeasterly orebody on No. 4 level was reached. This was typically silicification and enrichment of flat-lying beds of sandstone from narrow and steep fractures; shaly layers in the sandstone seemed to account for vertical localization of the enrichment. The type was like that of enrichment in flat beds in the Black Hills of South Dakota, except that the verticals were fainter and the beds less subject to replacement. Development northwesterly provided a meager ore supply into 1914.

In that year, it was decided to cross-cut from No. 4 level to the old workings lying to the west and slightly above the level. This work almost at once opened a bonanza, consisting mainly of altered shale enriched with free gold to a value of $50 to $100 per ton; 80 to 90 per cent. of the gold could be amalgamated and another 5 to 10 per cent., with a little pyrite, was readily caught on a Wilfley table. Although this new orebody was only 500 ft. distant from the old workings, its discovery, by downward prospecting along the contact, would have been delayed by the presence of folds and water-bearing fissures in the intervening barren ground. Subsequent development of this orebody has failed to find the slightest continuity with any other ore, and it shows no indication at the surface.

From 1912 to 1920, the mine produced $1,680,718 in bullion, varying from 856 to 889 fineness in gold and 105 to 142 in silver, and $243,079 in concentrates. Shortly after the discovery of the bonanza, the cost of producing gold was as low as $2 per ounce, but in 1919 it rose to $13.50 because of the large amount of timbering required and other operating difficulties.

In addition to our own conclusions, gained by several years' connection with the mine, the one as consulting engineer and the other as superintendent, we have the advantage of observations recorded by a number of other investigators. L. C. Graton[2] studied the district, in 1905, for the Geological Survey. In 1916, Edward H. Perry and Augustus Locke reported on the geological conditions at the mine. In 1918, O. H. Hershey made a study of the faulting systems and the ore occurrence. In the following description, dealing principally with the geological features of the Aztec mine, in addition to our own observations we have drawn freely from the written reports of these engineers.

GEOLOGY AND MINERALIZATION OF DISTRICT

As described by Mr. Graton, the Cimarron Range owes its elevation to faulting with accompanying upbending of the flanking strata. Unlike

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[2] *Op. cit.*

most of the high ranges in the northern part of New Mexico, its crest consists of sedimentary strata, although the pre-Cambrian basement is exposed in places. The sedimentary rocks belong chiefly to the Cretaceous and are broadly divisible into Dakota (?) sandstone, Colorado, Montana, and Laramie (?). Cutting the Cretaceous rocks are numerous intrusive masses of monzonite porphyry, which in many places is quartz-bearing. The faulting occurred at the close of the Cretaceous and was accompanied or closely followed by the intrusion of porphyry.

The primary gold deposits are of two types: quartz veins and contact-metamorphic deposits in calcareous sedimentary rocks; most of the gold has been obtained from the former. These veins are closely associated with intrusions of porphyry but, owing to the abundance of sedimentary rocks, many veins appear to be wholly contained in them. A common position for these veins seems to be at the contact of sills of porphyry with shales of Colorado and Montana age; several such veins are known on the Baldy side of the mountain, notably the Rebel Chief group. These veins and those like the Aztec vein, which occur between the Montana shales and the Laramie (?) formation, are locally spoken of as contact veins. While they do actually lie at the contact of different rocks, their position is probably due solely to the fact that fractures formed more readily at such places. These veins apparently differ not at all from those that do not occur at contacts, and it is probable that the character of the veins is not influenced by the fact that their walls are of different materials. In composition, these are essentially quartz-pyrite veins. Unquestionably, the veins have some relation to the porphyry and probably represent products derived at slightly later time from the same source as that of the porphyry. In other words, the ore-bearing solutions are believed to be the final product of magmatic differentiation, the most abundant and characteristic product of which was monzonite porphyry.

The contact-metamorphic deposits are fewer in number, and have produced less than the veins. They occur in sedimentary rocks at or near the contact with quartz-monzonite porphyry. The metamorphic effects are different in different deposits and are probably dependent on the chemical character of the sedimentary rocks. In the Ajax deposit, the original rock was probably a calcareous shale.

STRUCTURAL GEOLOGY AT AZTEC MINE

Mr. Hershey describes the structural geology. The original plane of division (locally known as the contract) between the sandstone series and the Montana shale, in places, remains in its original condition, but generally there has been slipping of the rigid sandstone over the soft shale, producing at the top a thin gouge or selvage of 6 to 8 in. of soft

FIG. 1.—NORTHEAST-SOUTHWEST SECTION OF AZTEC GOLD MINE.

LEGEND

Sandstone
Shale
Porphyry
Ore 1866–1900
Ore 1912–1920
Diamond Drill Holes

0   100   200   300   400   500
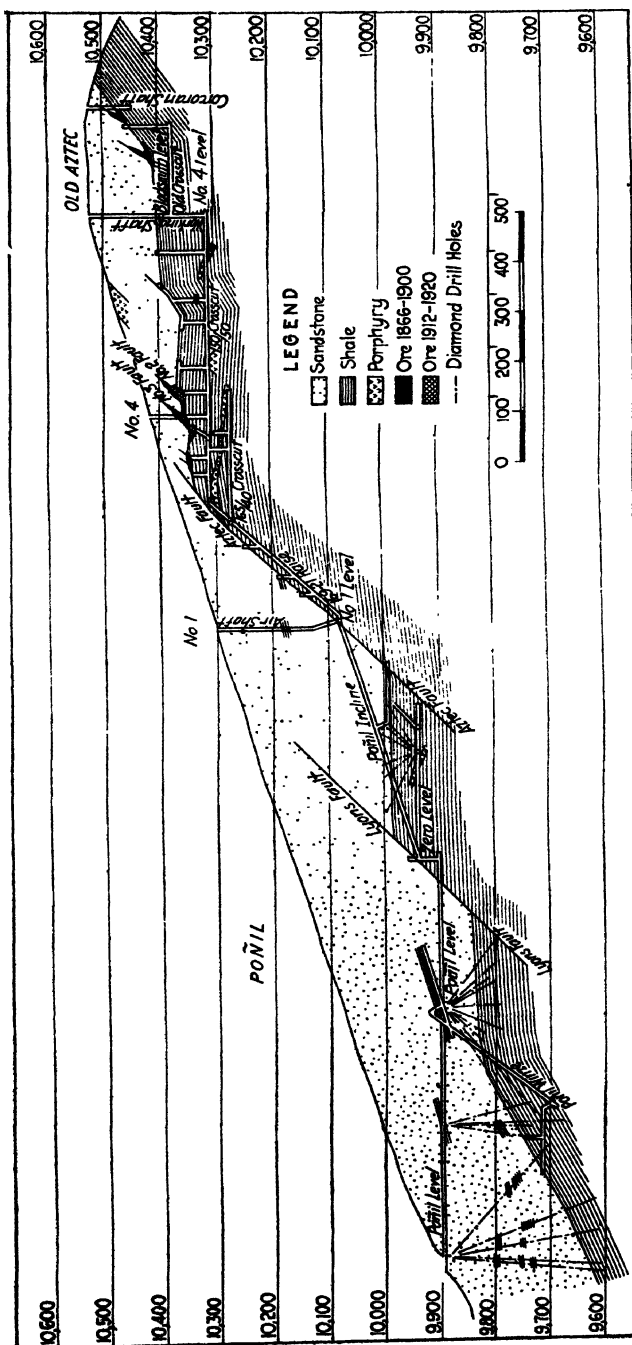
crushed shale. Where there has been no actual displacement, that is, cutting across of the beds, the contact is regarded as original. He recognizes two main periods of faulting, Fig. 1. The first period produced a group of faults of relatively small displacement. While they fault the sandstone, they do not penetrate deep into the underlying shales, but curve to low dips and practically follow the original contact. The second period produced two notable faults of large displacement, called the Aztec and the Lyons fault. The former dips northeastward from 20° to 45°, generally about 35°. Nearly everywhere it has a thick, soft gouge under a hard sandstone roof. The shales under it have been partly crushed to a depth of several feet. In places the disturbed zone contains large fragments of sandstone under the main gouge. In Fig. 1, it appears that the displacement on the plane of the Aztec fault has been about 550 ft. (167 m.). The Lyons fault, as exposed in the Pofiil workings, is accompanied by a black gouge and dips northeastward at 35°. It is a normal fault and appears to have a displacement of 450 ft. (137 m.). Grooving and striation, observed on both fault planes, is practically straight down the dip.

THE AZTEC OREBODIES

Mr. Hershey distinguishes three types of ore deposits, probably formed at three distinct periods. The first is the so-called "shale ore." This is remarkable for its high gold content, whereas contact-metamorphic ore deposits usually carry gold in relatively small quantities. A somewhat similar deposit appears to occur at the Ajax mine on the southwestern slope of Mt. Baldy. There the gold is distributed through a dark, heavy, finely granular rock, consisting of nearly colorless pyroxene, amphibole, epidote, magnetite, a little zoisite, scapolite, and specularite. The Aztec shale ore is a dark green, heavy, finely crystalline rock in which the microscope might show the same minerals as are present in the Ajax ores. In the field, the Aztec ore seems to consist of greenish grains that may be zoisite or a small green garnet; there is also considerable calcite in places. The gold is embedded in the green mass in ragged grains up to nugget size, which can often be seen with the unaided eye. There is little pyrite or other sulfide present. The ore occurs at the original contact or a few feet below it and has been involved in the crushing of the shale, for it abounds in curved slickensided faces with a greasy feel. Underground, it is difficult to distinguish from the shale.

The second type of mineralization is that accompanying faults Nos. 2 and 3, Fig. 1. Its essential minerals are calcite and pyrite. The ore occurs in seams and bunches scattered through a zone of broken ground accompanying the fault gouge. Other carbonates, such as rhodochrosite, may be present and probably a little quartz. Much of the pyrite is coarsely crystallized and there may be a small amount of chalcopyrite

with it, but this is not easily recognized underground. The gold content runs from $8 to $50. The metal is fine grained and not visible. Where these fault veins lie at low angles, practically coinciding with the shale-ore lenses, the gold content is relatively high. Good ore continues up along the fault into the sandstone but gradually weakens. Down the dip from the shale-ore lenses, the fault veins contain much pyrite but little gold, and soon pass into non-commercial sections. These facts lead to an inference that the mineral-bearing solutions ascended along the fault
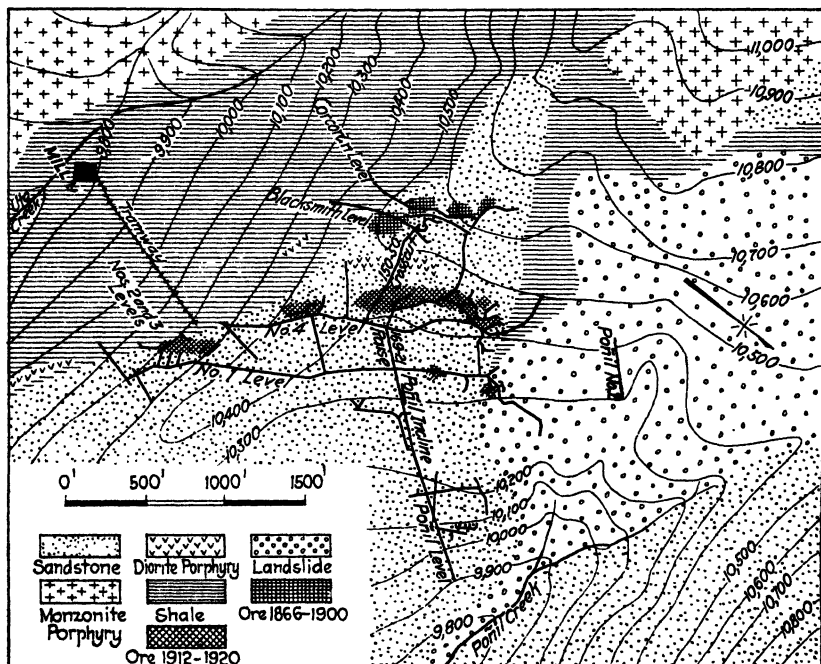
FIG. 2.—PLAN OF AZTEC GOLD MINE.

veins and that they derived their gold from the shale ore. Hence fault veins that do not touch shale ore are not likely to carry commercial ore; and if a fault vein is found to carry commercial ore, it may indicate the presence of shale ore somewhere down along the dip.

The third type of deposit is genetically connected with Aztec, Lyons, and other faults of that system. It occurs almost exclusively in sandstone and its most characteristic mineral is chalcopyrite. The gold content is relatively low, and free gold is rarely or never visible. Movement along the fault fractured the relatively brittle sandstone, forming little fissures in which the solutions circulated. The resulting mineralized zones vary from a few inches to many feet in thickness.

The orebody at the extreme northwest that was commercially important seems to us to be of a different type from those described. It was extremely high-grade ore having the characteristics of contact metamorphism. It lay just between the contact of shale with overlying sandstone partly altered to quartzite, and though seemingly a part of neither, it extended a short distance upward along the steep, small fractures, fading away into the sandstone. Beside masses of coarse and fine native gold, this ore contained quartz, limonite, garnet, pyroxene, and other silicates, a little pyrite and chalcopyrite and always an appreciable amount of bismuth telluride. The latter mineral, tetradymite, was identified by A. J. Weinig. Native gold in coarse particles was embedded in it, yet practically uncombined with tellurium. The gold amalgamated readily. A similar occurrence of ore having the same mineral components and apparently derived by similar processes at Dolcoath mine in the Elkhorn mining district, Montana, is described by W. H. Weed.[3]

The quartzite close to the ore is hard, glassy and somewhat stained on its fractures with a brilliant green and blue film of very recent copper oxidation products. While the ore is quite hard and tough, the shale beneath it is soft. It is barren, strongly iron-stained throughout and sometimes copper-stained in the uppermost few inches, just under the quartzite. The change from high-grade ore to barren shale under it is sharply marked. The hardening alteration of the shale so pronounced under the bes tdevelopment of shale ore is entirely absent here and an opposite condition exists, notwithstanding the presence of the dioritic sill and its much closer approach to the shale-sandstone contact.

At the time Mr. Graton visited the property, the Old Aztec orebody alone was known. He describes the main underground workings as being near the contact of the shales of Montana age with overlying coal measures, which are commonly regarded as Laramie. The ore is in some respects unusual. It occurs mainly in coarse sandstone just above the contact with the shales and it consists of a number of veins of which a few are at the actual contact with shales and nearly all are parallel to the bedding of the rocks. The lode thus formed strikes about west-north-west and has a variable dip, which averages about 25° to 35° northeast. A few veins, which are apparently branches or offshoots from the main lode, cut upwards almost vertically through the sandstone. The walls of the narrow fractures in the sandstone, which constitute the veins, are covered with a thin layer of dark material, presumably limonite, and this is said to have been fairly bristling with free gold in the rich ore. It seems probable that this was once rich auriferous pyrite, which was later oxidized. The stopes from which the ore of the main lode was

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[3] W. H. Weed, U. S. Geol. Survey, 22 *Ann. Rept.*, pt. 2, p. 506.

extracted are disconnected, and as all are not on the same fracture they overlap in places.   Several of the stopes of the main flat lode extend out to the surface of the steep hillside.

Derivation of the Orebodies

In describing the shale ore, the first of his three groups, Mr. Hershey says that this is obviously shale which has suffered contact metamorphism.   The minerals it contains are those formed by contact metamorphic action.   The causative igneous rock is not definitely known but somewhere a mass of porphyry may have been intruded across the original contact and the metamorphism spread outward from it along the shale layer under the sandstone.   The porphyry may have been removed by erosion or it may exist in some unexplored section.

The dioritic sill that underlies the principal productive orebodies on the No. 4 level, and which is separated from them by 40 to 100 ft. of much less altered and absolutely barren shale, might have produced the metamorphism; assuming that the ore represents a layer of shale that was slightly calcareous and thus amenable to an alteration different from that of the non-calcareous shale.

At present it is known only that the shale-ore lenses seem to occur in a relatively narrow belt elongated in a north-northwest direction.   The orebody as a whole lies flat but in detail is somewhat undulating, a common form being that of a trough.   The shale orebody is much complicated by the presence of faults Nos. 2 and 3 and a fissure running obliquely between those faults.   Toward the northwest the Aztec fault appears to have encroached upon the orebody and perhaps dragged part of it down below No. 1 level.   At the extreme northwest, the Aztec fault seems to swing sharply toward the north, thus permitting another lens of shale ore to appear under it.

In connection with Mr. Hershey's suggestion that metamorphism may have originated in a porphyry dike on or intersecting the sandstone shale contact, it is worth noting that diamond drilling did encounter intrusion at a point 2500 ft. northwest from the principal orebodies and lying on the contact between the two sedimentary rocks.

Perry and Locke noted that the most promising localities for the occurrence of shale ore were the intersections of east-dipping faults of small displacement with flat-lying contacts of shale and sandstone, including among the latter not only the main Montana-Laramie contact, but those existing at the boundries of shale beds within the sandstone.

They described the ore as chiefly an alteration of shale at or close to the contacts of shale with overlying sandstone.   It is soft and greenish gray, consisting largely of chlorite, carrying scattered grains of pyrite

and native gold, with occasional gray metallic specks which may be telluride or selenide of gold. Rhodochrosite and calcite are not uncommon, while the lack of quartz of the same generation as the ore makes it an unusual one.

The sill of fine-grained porphyry, probably everywhere in the vicinity of the principal mine workings and lying 40 to 60 ft. below the main contact, is a variety not known elsewhere here. The shale underlying the ore and overlying this sill has undergone a hardening alteration, producing a dark gray rock lacking in shaly structure and carrying disseminated pyrite. Whether or not this alteration always or usually occurs under ore is not known. However, the effect of the sill is undetermined, although it could have constituted the source of the gold. The hardening alteration of the shale is an effect and not a cause of the mineralization.

Perry and Locke think that the reasons for localization of the ore at the main shale-sandstone contact are probably complex. Their most plausible hypothesis depends on the fact that, while the shale is somewhat plastic and does not readily maintain open spaces for the migration of solutions, the sandstone is brittle and does readily maintain such spaces. It depends, again, on the fact that the shale is more readily replaceable by ore minerals than the sandstone. According to this hypothesis, the sandstone would furnish the path for travel of gold, and the shale would furnish the matrix for its reception. Furthermore, the east-dipping faults break the sandstone and the shale. The extension of the orebodies in a direction parallel with that of the faults is plausibly attributed to the effect of the faults in cracking the brittle sandstone and crushing the more plastic shale. At the same time, the dip of the bedding in various parts of the workings indicates the possibility of an anticline with a northward trend, the axis of which coincides with the most productive part of the orebody. The apparent preference of the ore for the main contact cannot be explained by any particular abundance of soluble minerals in the shale in that locality, because no such abundance has been observed.

It seems to us that the granodiorite intrusive has been one of the controlling factors in ore derivation. The fact that this variety of porphyry is not known elsewhere (and, conversely, so far as known, the ore has not been found disassociated from it) is outstanding. Coupled with the presence of the sill everywhere in the shale, under the ore in the No. 4 level workings and over it in the Old Aztec workings, there is a good fracture zone at each of these main orebodies. In neither case does the sill join the orebody but the faults of small displacement do. They have crushed and opened the sandstone above the original contact, forming channels where solutions and gases could travel. The hardened shale, overlying the sill, was also fractured and cracked, maintaining small open spaces, however, to a much less degree.

Many areas of original contact were investigated by diamond drilling and mine exploration. The only mineralization found was below the Poñil level where the contact was slightly folded and where fracturing had resulted. Elsewhere the contact was tight and barren.

In all, there is evidence that mineralization has followed or been connected with fracturing and a little evidence that fracturing has followed folding.

Regarding Mr. Hershey's second group of orebodies, the veins accompanying faults Nos. 2 and 3 extending upward from the No. 4 level workings, he thinks the facts point to the inference that the mineralizing solutions were ascending along the fracture planes in the sandstone, having derived their gold from the adjacent and underlying bodies of shale ore. Thus he concludes that any fault veins which do not communicate with bodies of shale ore are unlikely to carry profitable ore; and conversely, any fault vein found to carry good ore, will, if followed downward, probably lead to a body of shale ore.

As to his third group, the copper mineralizations in sandstone adjacent to the principal faults and others of the same system, he argues that movement along the faults fractured the relatively brittle sandstone, forming little fissures in which the solutions circulated. The mineralized zones vary from a few inches to many feet in thickness. As the sandstone is usually over the fault the ore rests upon the gouge or occurs along seams over it, suggesting deposition by descending water; but when the sandstone is under the gouge, the same argument points to ascending solutions. The crystallized calcite and chalcopyrite are minerals not often formed by descending meteoric waters and would indicate solutions ascending.

PROSPECTING

Exploration for additional orebodies was conducted by drifting and diamond drilling, with particular attention to the regions of original shale-sandstone contact. Of drifting, slope sinking, and similar work, a distance of 9883 ft. was accomplished during the years 1918–1921, at a total cost of $15.89 per foot. During the same period, diamond drilling aggregated 18,276 ft. at an average total cost (including sampling, assaying, etc.) of $4.59 per foot. The average length of 77 holes was 237 ft., and many of the holes drilled from the surface encountered unusual difficulties in penetrating the talus deposit.

"Two large and high-grade orebodies were found at opposite ends of the deposit on No. 4 level, two small bodies were found on No. 1 level, an extensive but low-grade deposit was opened on the Ponil level; otherwise only scattering high assays near the lowest level reached rewarded the work."

MINING

The principal mine entries followed the Aztec fault and the main ore zone lay just southwest of No. 4 level and the fault, Fig. 2.

This zone, paralleling the Aztec fault a distance of 800 ft., had a maximum width of 80 ft. and lay 30 to 60 ft. above No. 4 level. Within it was a more or less continuous extent of ore. A generous provision of branch openings on the level and raises was made for the efficient movement of ore from stope faces to loading chutes on the haulage level.

The considerable length of the raises made them useful for ore storage at all times and, in the later days of the mine when painstaking mining around the margins of the ore was necessary, tonnages from narrowly limited areas were segregated in these raises and milled as separate lots. The practice was invaluable in avoiding the mistake of degrading good ore from one stope with ore from another place of lower grade. Hand sampling was the first guide to stoping but actual mill result governed; this practice materially extended the life of the mine.

The mine was mapped on coördinates approximately paralleling the Aztec fault with 100-ft. blocks, counting from the southeast end of the property. Abscissas were laid off from an arbitrary base. A factor of ten was used and block 170/50 was 1700 ft. northwesterly and 500 ft. southwesterly. Within that block a drift, crosscut, or raise took designation from the coördinates at its beginning; for example, raise 213/57. Samples were recorded in the same way. The working map was on a 10-ft. scale to accommodate the great amount of detail and in time the single large sheet gave way to maps of individual blocks, 100 ft. square, on which were entered the hand samples taken and the records of the mill lots extracted.

Timbering was by square sets, where the ground was heavy, or soft; and by posts or stulls, where it was hard. At times, waste from development served in large part to fill the sets, but frequently waste raises into the overlying sandstone were necessary; these served also to prospect the sandstone: they never found ore. Much of the ground was heavy, requiring careful timbering. Some of the best ore was overlaid by material that required spiling before the ore could be drilled and blasted. Timbering was by far the largest single item of expense in the extraction of the ore, though the cost of timber was reduced by the operation of a sawmill, a few miles from the mine.

MILLING

The mill is a ten-stamp amalgamation and concentration combination and by it a recovery of 85.81 per cent. was obtained in later years. As is frequently the case at small and out of the way mines, the provision of dependable and economical power, at moderate initial expense, was one of

the serious problems. In early days of the present operation a steam plant was in use, it having been handed down from a previous period of operation. Water for power was available during one month in the spring of the year. Upon development of a considerable supply of ore and needs for power expansion, the semi-Diesel type was selected and Muncie engines in 30- to 80-hp. units placed in the mill to furnish power for all purposes. A 35–40 gravity fuel oil gave better results than a heavier oil. Careful attention to lubrication, condition of the cylinders and piston rings, amount of water passing through the jackets, condition of fuel nozzle, and the amount of water used in the bypass is necessary for best results. The engines are simple, easy to operate, and satisfactory.

COSTS

On a basis of 30 tons, daily, mined and milled during the years 1918–1920, inclusive, the following costs prevailed.

Mining....................................	$4.91 per ton
Milling and marketing.........................	3.54 per ton
General....................................	1.62 per ton

The total cost of power was $0.0396 per hp.-hr. Fuel oil cost $0.151 per gal. and lubricating oil, $0.967. Fuel oil was used at the rate of 0.708 lb. per hp.-hr. and lubricating oil, 0.041. Trucking 16 mi. the round trip, with practically no back haul, cost $6.86 per ton of freight hauled. The mine is 2500 ft. higher than the railroad.

During the years 1918–1921, inclusive, the cost of mine development, including direct and indirect charges, was $15.89 per ft. over 9883 ft. of work. Of this amount, $7.59 was the cost of explosives, labor of breaking ground, mucking and placing timbers. Diamond drilling 18,276 ft. cost a total of $4.59 per ft., distributed as follows:

Direct charges:	
Drilling. only............................	$3.10
Pipe lines and stations...................	0.37
Power....................................	0.66
Water....................................	0.12
	$4.25
Indirect charges:	
Assaying, surveying, sampling and supervision..............	0.34
Total cost...........................	$4.59

The shortest hole was 40 ft. and the longest 748 ft. An average length of 237 ft. was obtained in 77 holes drilled.

Spies Open-stope System of Mining

BY S. R. ELLIOTT, ISHPEMING, MICH.*

(New York Meeting, February, 1922)

THE Spies mine is located in the eastern half of the northwest quarter of section 24–43–35, near the village of Iron River, Mich., and is operated by The Cleveland-Cliffs Iron Co. Speaking generally, the jasper hanging walls of most of the deposits in the Iron River district dip at a steep angle and are exceedingly strong, standing without caving, even after mining has been continued for years and after large excavations have been made. The ore is fairly hard and tough and contains no slippage faces. It occurs in large masses, which are often egg shape, with the smaller end down, and can be mined in open stopes without the use of timber. The common system employed in the district is the shrinkage stope, or some modification of it. The Spies system was developed to utilize the advantages of the open-stope method and to avoid the disadvantages of the shrinkage stope.

The Spies shaft was sunk to such a depth that the bottom, or third level, was in rock at a short distance below the bottom of the orebody, the shape and extent of which had been previously determined by diamond drilling. This is shown in plan and by the ore contours at the first, second, and third levels in Fig. 1, and in section in Figs. 2 to 7.

In order to provide the necessary mills and raises, three parallel crosscuts were driven on 25 ft. centers on the bottom level. Drifts on the second and first levels were driven to the orebody and followed the foot wall in a southerly direction to the end of the ore. In crosscut 3, Fig. 1, raises 30, 40, 50, and 60 were carried up on the foot wall and holed into the second level, and were ultimately carried up and holed into the first level. These foot-wall raises carry the pipes and ladders, provide a means of ventilation, and furnish safe communication to all working places in the stope.

From crosscuts 1, 2, and 3, raises were completed to a height of 32 ft., or the elevation of the first sublevel. They were then connected by east and west crosscuts to the main foot-wall raises 30, 40, 50, and 60, shown in Fig. 4. The crosscuts in raises 1 to 5 were then driven west and the

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*General Superintendent, The Cleveland-Cliffs Iron Co.

hanging wall located. While this work was in progress, raises 1 to 5, 10, 12, 14, 16, 18 were continued until they reached the hanging wall. At intervals of 22 ft., the raises were connected by east and west crosscuts and the hanging wall to the west was located at the elevation of each sublevel. This is shown in Fig. 2.
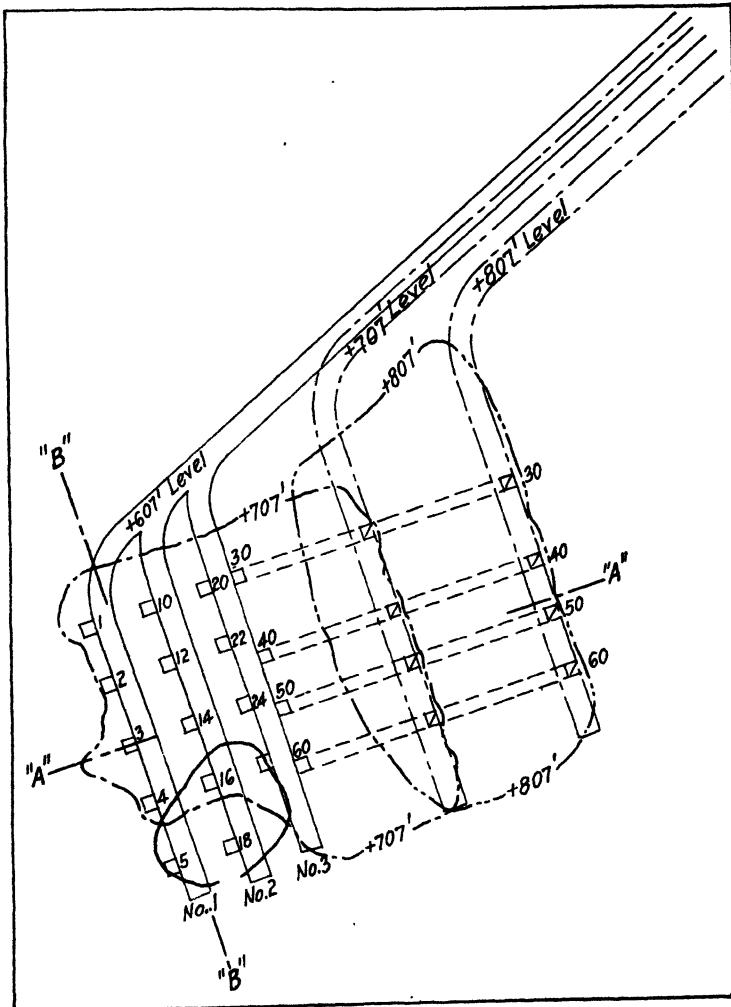
FIG. 1.—PLAN OF SPIES MINE, SHOWING MAIN LEVELS, CONTOURS OF OREBODY, RAISES AND MILLS.

Below the 639-ft. sublevel raises 1 to 5 were coned to an angle of about 45° to serve as mills. Miners then started in each crosscut on the 639-ft. sublevel, at points near the hanging wall and fanned holes toward the same. By repeating this operation on sublevels at higher

elevàtions, the ore was cut away from the hanging wall and a narrow opening existed over the mills in crosscut 1 for the length and height of the orebody. From this time on, to the completion of the stope, the work was straight mining. Miners set up their machines in the various crosscuts at a safe distance from the edge of the open stope and fanned holes toward the west, care being taken to pull back the east face in a vertical plane.

If one could stand against the hanging wall at a point opposite the center of the stope and look east, this face would resemble, in perspective, a number of pyramid-shaped funnels in a horizontal position, the larger end pointing toward the observer, the crosscuts being the spouts. This is shown in section and elevation in Figs. 5 and 6. While stoping is being continued, it is necessary to complete additional raises and to do crosscutting at higher elevations, or in parts of the orebody farther east.

DISADVANTAGES OF THE SHRINKAGE STOPE

1. In a shrinkage stope, after mills have been made, the orebody is undercut. By setting up the machines on the broken ore and working around the perimeter of the stope, its height is increased until the top of
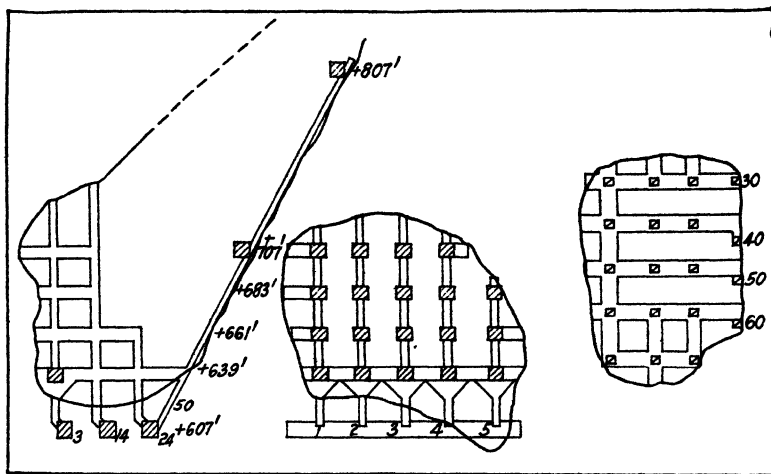
FIG. 2.   FIG. 3.   FIG. 4.

FIG. 2.—SECTION A-A BEFORE STOPING IS BEGUN.
FIG. 3.—SECTION B-B BEFORE STOPING IS BEGUN.
FIG. 4.—PLAN OF 639-FT. SUBLEVEL.

the deposit is reached. After each blast, only sufficient ore is drawn from the mills to make room for the miners to work. A safe rule to follow is to have the back above the point where the machines are set up close enough for it to be touched by a miner when his arm is extended. With this precaution, it can be readily trimmed. Further, miners should

not be permitted to go out toward the center of the stope, but should be kept close to a solid wall. Observing these rules and always working around the perimeter causes large bellies to remain in the center of the stope which either must be blasted, by the use of long drills, or if they cannot be reached they must be left until they become so heavy that they will fall because of their weight. In time, these large masses are drawn down to the mills, when they must be broken by men who enter the mills from the bottom to drill and blast them; this is slow, expensive, and exceedingly dangerous work.
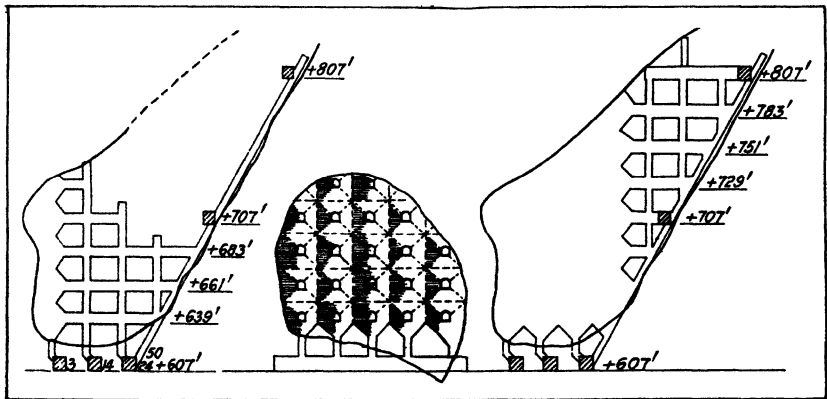
FIG. 5.   FIG. 6.   FIG. 7.
FIG. 5.—SECTION A-A AFTER STOPING IS BEGUN.
FIG. 6.—ELEVATION B-B.

2. As the ore is broken in a shrinkage stope, only about one-third of it can be drawn out of the mills. The pile in the stope must constantly be increased in height until the top of the deposit has been reached and the stope completed. A large investment, represented by two-thirds of the broken ore, is tied up until the stope is completed.

3. The miners spend a large proportion of their time trimming the back, trying to make it safe; their drilling efficiency is, therefore, low.

4. In drawing sufficient ore each day to keep the top of the pile at the proper distance from the back, there is constant danger that the ore will not settle properly and later on will surge and endanger the lives of the men in the stope.

5. If seams or bunches of rock are encountered, they must be broken with the ore; there is no means of separating the material.

6. Ventilation, as a rule, is poor and much time is lost after each blast.

ADVANTAGES OF THE SPIES SYSTEM

. These will be compared in the same order as the disadvantages of the shrinkage stope.

1. All men are provided with a safe and easy traveling road to their working places. There is every opportunity for miners to drill their holes with a proper burden on them, so that it is unnecessary to break the ore in large masses. It is not necessary to go up into mills to drill and blast. These are the chief factors that make it possible to obtain low cost by the use of this system.

2. All of the ore broken each day is drawn from the mills, only enough being left to protect the stopers from the shock of falling ore blasted in the stope. An accurate check can, therefore, be kept on the miners and high efficiency obtained.

3. No time is lost in trimming. The men are in no danger from falls of ground, so their time is spent in drilling and blasting ;their efficiency is, therefore, much higher than in the shrinkage stope.

4. There is no danger that the ore upon which the men are standing will be drawn from under their feet, as they work in crosscuts with solid ore below them. Their machines are set up on posts, which is preferable to using tripods on loose ground.

5. During the development work in the sublevels, all seams of rock in the ore are outlined and can be mined before the stope has reached that locality, the rock being dumped into raises that are not being used as mills. It is, therefore, possible to mine the ore and keep it clean.

6. On account of the large number of small raises and the connections with upper levels, no trouble is experienced with ventilation. Blasting can be done at any time during the shift and only a few minutes lost. The area of the open stope soon becomes so large that the gases are rapidly dissipated and pass off through the upper levels.

With the exception of a small thickness of ore left above the first level to support the overburden of sand, the stope in the Spies is completed. In this mine, on a comparatively narrow orebody, we have also worked the shrinkage-stope method, and, therefore, have had an opportunity to make a close study and comparison of the two systems. Our records show that the cost of breaking ore was about 40 per cent. higher in the shrinkage stope than where the Spies system was used. From every point of view, the Spies system has proved more satisfactory than the shrinkage stope.

Exploration Methods on the Gogebic Range

BY W. O. HOTCHKISS,* PH. D., MADISON, WIS.

AN ESSENTIAL mental equipment for planning exploration is the fullest possible knowledge of the way in which the orebodies occur in the region to be explored, also the realization that in no mining district is this knowledge complete, and that new facts of importance will be revealed to the explorer in every exploration if he is in the proper frame of mind to grasp them.

The geology of the ore deposits of the Gogebic Range for a long time was thought to be relatively simple, but work in recent years, by several men, has proved that this is not the case. We now realize that we are far from knowing all the factors that have resulted in the localization of ore deposits. For example, we have long known that the intersections of the dikes and the foot wall were favorable places to look for ore, but only within the last few years have we learned that there are several horizons in the various members, well north of the foot wall, that are likely to carry ore. The influence of the numerous cross faults on orebodies is not known or appreciated beyond a general realization that they are important. Just how they may be favorable or unfavorable to the localization of ore is not known. Such elementary information as the positions of these faults is just beginning to appear on the mine maps.

Such being the condition, one can hardly overestimate the importance to the various operators and fee owners of careful studies of conditions in their properties for the purpose of assuring themselves that they are not missing valuable ore—that they are doing all the exploration that their situation demands in order to secure maximum returns. Valuable ores have been missed in the past and will be missed in the future. Only with a complete knowledge of all the factors that have united to produce ore could anything else be expected. The purpose of this paper is to call attention to these facts briefly and to suggest a few of the principles that should guide effective exploration. The foremost principle is that the man who starts out with the idea that he knows all about the mine geology of the Gogebic Range is starting with a heavy handicap.

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BRIEF OUTLINE OF GOGEBIC RANGE GEOLOGY

The iron-bearing series on this range is similar to that in the other Lake Superior ranges. The important economic feature is the iron formation, on this range named the Ironwood formation, with which are associated the foot-wall quartzite and the slate hanging wall. These formations dip northward at an angle of approximately 60°. Numerous dikes cut these formations at nearly right angles to the bedding. Most of them pitch 20° to 30° from the horizontal downward toward the east, but some of them pitch at similar angles to the west. All of them make relatively impervious pitching troughs with the foot wall and with other impervious beds which occur within the iron formation. It is chiefly in these pitching troughs that orebodies occur.

The term iron formation in this paper is taken to include all the beds known to bear ore and so includes the basal part of the hanging-wall formation as well as the Ironwood formation.

In a recent description of the geology of this range,[1] I divided the Ironwood formation into two major divisions and five subdivisions, as follows:

Tyler formation (hanging wall). . . . . . Pabst member (ore-bearing)

Ironwood formation. . . . { Upper Ironwood. . . . { Anvil member / Pence member

Lower Ironwood. . . . { Norrie member / Yale member / Plymouth member

Palms quartz slate (foot wall)

For a detailed description of the various members of the iron formation and their characteristics the reader is referred to that article.

These six major members are recognizable throughout the main producing part of the range and are convenient guides for reference. However, when the exploration of a single mine is under study these divisions are not sufficient. The smaller the recognizable divisions that can be made, the more intelligently will the work be done. In one mine, the engineers have made over twenty divisions for correlating the geology of the various crosscuts and drill holes. So definitely can the successive beds be determined that the duplication or ellipsis of one or more beds by faulting is often evident in the drill core, and thus the point where the drill cut the fault and the direction and amount of throw are determined and mapped even though the core itself gives no suggestion of a fault.

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[1] Geology of the Gogebic Range and its Relation to Recent Mining Developments. *Eng. and Min. Jnl.* (1919) 108, 443.

The iron formation and the dikes are cut by a great fault, which is practically parallel to the beds. The northern part, or top of the formation, has been moved several hundred feet eastward in relation to the part below or south of the fault. There are also numerous faults at nearly right angles to the beds. Most of these are vertical, but some have about the same position as the eastward pitching dikes. A few faults also cut the formation at other angles, but it is not necessary to describe them here.

ORIGIN OF OREBODIES

The orebodies of the Gogebic Range, with a few unimportant exceptions, are caused by the leaching of the silica from the mixture of chert and iron oxide that makes up the iron formation. This leaching is done by meteoric water, which more or less slowly finds its way through the formation. Any condition that favors this water circulation is favorable to the formation of orebodies; any condition that hinders or prevents the circulation is unfavorable.

If a particular bed of the iron formation were originally high in iron and low in silica, it would require less leaching to produce ore. So if two beds were equally well situated with regard to water circulation and equally porous, the one that had the least silica would be first changed to ore. However, the bed having the least silica might be so extremely dense and non-porous that the bed that was more porous and contained a much larger percentage of silica might alter to ore while the first was not affected.

The ore-bearing horizons at present known are as follows, named in order from the foot wall north:

1. The base of the Plymouth member; this is where most of the ore has been mined.

2. Near the top of the Plymouth member.

3. In the bedding fault zone; where this fault has broken the ferruginous cherts it has favored the circulation of water and in a number of places ore has resulted. In most places the fault is in an impervious black slaty carbonate and no ore is found.

4. The top of the Norrie and base of the Pence members.

5. The base of the Anvil member.

6. About 100 ft. north of the base of the Anvil.

7. The Pabst member.

In a few of the largest orebodies, the ore is continuous from foot to hanging, but even in these bodies the rich porous beds of the formation are altered to ore farther from the dike than the other beds.

Among the known conditions favorable to localizing the leaching effect of the water, and hence favorable to the formation of ore, are: intersections of dikes and the foot wall or of dikes and impervious beds

of the formation; the more pervious beds of the iron formation; those parts of the formation that have been broken by faults.

EXPLORATION

In the order of time in a well-planned exploration there are three general things to be determined. (1) The positions of the various dikes both north and south of the great bedding fault must be known either from drilling or underground workings. (2) The particular beds known to bear ore should be located from some definite reference plane, such as the foot wall, so that the places where these beds are cut by the dikes can be determined with the necessary degree of accuracy. These intersections are the objectives for which exploration should aim. Every drill hole, exploration drift, or crosscut should be planned to cut a favorable part of the formation just above a dike. (3) The effect of faults must be considered.

Faults can be located in many cases by careful magnetic observations. This has been done successfully in both the eastern and the western ends of the range, and doubtless can be done in many places in the central part by working along the Pabst member of the formation. A careful examination of mine workings has served to identify many small cross faults that have not been observed before because their throw was too small to attract attention. These cross faults apparently die out toward the south so that the throw is often only a few feet at the foot wall, but toward the north side of the formation the throw increases and they are of greater importance in planning both exploration and mining work. These cross faults have, in many cases, made excellent opportunities for the water to get through and leach the formation, and oreshoots have been followed vertically hundreds of feet along them.

The nature of the influence of cross faults on the localization of orebodies is little known, except for the case where the fault zone itself is altered to ore. Between two faults a particular bed may be altered to ore on a dike. In the block next east, this same bed on the same dike may be almost wholly unleached and the ore may be found in a different bed, either nearer to or farther from the foot wall, or the entire block may be barren. The water has evidently gone through one fault block in one zone on a given dike. When it reached a cross fault, it dropped down that fault and found its way through the next block on some lower dike and perhaps in some other bed, which offered a more favorable channel. Sometimes it has gone through several fault blocks on the same dike and the cross faults are marked mainly by enlargements of the orebody due to the favorable situation created by the breaking up of the formation by the fault.

All these facts should be considered in planning exploration and

interpreting the results. Successful work demands close observation of the formation and careful study of all the possibilities of finding ore.

DRILLING

The iron formation of the Gogebic Range is especially difficult to drill. The leaching of the formation attacks favorable beds and leaves the less favorable ones more or less untouched. The result is an alternation of hard dense and more or less soft vuggy beds. There is consequently much difficulty in getting a proper return of samples, and also a great deal of trouble in operating the drill. The alternating hard and soft beds make it almost impossible to keep a drill hole straight for more than a few hundred feet. In order to keep the samples from contamination, it is necessary to keep the casing close to the bit as soon as the drill enters soft ground. If this is not done, an orebody may be passed through and the samples be so diluted by the soft leached chert worn off the walls of the hole that the analyses will show nothing of value.

The large amount of time and trouble necessary to handle caving holes and to fill open vugs makes diamond-drill operations difficult and expensive. The great expense and the inability to keep the drill straight on its course make most holes longer than 1000 to 1500 ft. of doubtful utility if there is any reasonable possibility of reaching the objective in any other way.

In consequence of the foregoing facts, the main function of the drill on this range is to determine general conditions rather than to find ore. Diamond drilling can be used to excellent advantage to show the nature and condition of the formation in advance of and for the purpose of guiding drifting and crosscutting and going to greater depth. But these extensions should not depend wholly on the actual finding of ore by the drill. If the drill shows a well-leached formation on a good-sized dike, the dike is well worth exploring by mine openings. Often a hole lost as the result of caving is an excellent indication of ore below. It is difficult and sometimes impossible for the drill to get through the broken formation which sometimes overlies an orebody that has been broken by the slump of the ore on account of the leaching out of the silica.

The core recovery from about 25,000 ft. (7620 m.) of diamond drilling, obtained from 26 holes of widely varying depths, was found to average 30.7 per cent. The core recovery from the individual holes varied from 6.4 to 58.1 per cent. The lowest recovery was from a short hole practically all in well-leached formation; the highest was from a hole that extended through hanging slates and the full thickness of the formation. This hole happened to go through an almost wholly unleached part of the formation. It probably represents about the maximum core recovery from average drilling.

The foregoing statements indicate strongly that drilling is not satisfactory as the sole method of exploration; but this does not imply that drilling is not a useful method. Drilling should be used by mine operators far more extensively than it has been in the past; even though the drill may not find ore, it may show a condition of the formation that warrants extending exploratory mine workings to determine what really exists in favorable situations that the drill may find.

EXPLORATORY DRIFTS AND CROSSCUTS

Exploratory drifts and crosscuts constitute the most satisfactory method of exploring, but such work is too expensive to carry on without having a definite objective, which should be shown in advance by drilling. In the past, before it was known as definitely as at present just what beds are the most likely to carry ore, many long crosscuts were driven north to the hanging wall. A few of them found ore but most of them did not. Most of them were driven with little or no knowledge of where the dikes lay north of the great bedding fault, and more often than not when they did penetrate dikes they were in beds that were unlikely to contain ore. Some of these old barren crosscuts could be used to great advantage, at present, by driving drifts or raises from them to explore the good ore-bearing horizons just above the dikes.

Every exploratory drift or crosscut should be planned to give the maximum possible information with regard to one or more of the seven known ore-bearing horizons where they are cut by fair-sized dikes. If the pitch of the dike is sufficiently great to make its strike cut the strike of the beds at a fairly large angle, the best results are obtainable by driving along the top of the dike—northeast with an east-pitching dike, and northwest with one pitching west.

The cross faults should also be given careful consideration in planning an exploratory crosscut or drift. A crosscut that strikes a favorable ore horizon on a good dike where the formation is faulted has probably the best chance of finding ore. These cross faults cut the formation into blocks that are seldom more than ¼ mile long east and west and are often only a few hundred feet long. In order to test the formation thoroughly, every fault block should have every favorable ore horizon tested on every fair-sized dike. This statement is made with a full realization that it is not always advisable to explore a property thoroughly. The object of exploration must always be to find ore—not to prove that no ore is there. General indications of the character of a certain block on a certain dike may make such a territory a good one to neglect. But if one bed on a dike is well leached or carries ore, the other favorable beds should also be tested on this dike. The failure to do this in one mine examined resulted in the loss of a very large tonnage of ore.

Exploration of the Gogebic Range offers great possibilities of reward. Even the foot wall has been only partly explored down to the depth of present operations. When to this unexplored foot wall are added the parts of the formation, little known but well worth exploring, north of the foot wall, the greater and unknown depths below to which ore may go, and the neglected parts of the range east and west of the producing mines it is obvious that this range has a long and profitable future before it.

Systems of Mining in Pocahontas Coal Field and Recoveries Obtained

By Thomas H. Clagett,* Bluefield, W. Va.

THE Pocahontas coal field comprises the area in Tazewell County, Va. and Mercer and McDowell counties, W. Va., in which Nos. 3 and 4, Pocahontas seams of bituminous coal are mined. It is a mountainous region with ridges rising 300 to 1000 ft. (91 to 305 m.) above the very narrow valleys and, except for some poorly kept farms along the tops of the ridges and the area occupied by the mining and coking plants and villages, is wooded throughout. First shipments were made from Pocahontas in 1883. There are now about 100 mines, 95 per cent. of which are drift mines. At the end of 1920, approximately 298,000,000 gross tons of coal had been produced and 35,000 acres of coal mined.

No. 3 seam is from 4 to 10 ft. (1.2 to 3 m.) thick, but the change is gradual and the thickness fairly uniform in any particular locality. It has a streak of bone, about 2 in. (5 cm.) thick, to which the coal adheres on both sides. In parts of the field this bone is replaced by hard slate, usually from 2 to 4 in. thick. The seam has a slate or fireclay bottom and a draw-slate top.

No. 4 seam is from 65 to 80 ft. (19.8 to 24 m.) above No. 3 seam, and of minable section only in the western part of the field. It is from 3 to 8 ft. thick and has two streaks of bone, each from 1 to 2 in. thick. It has a fireclay or slate bottom and is overlaid by from 4 to 24 in. of a laminated coal and slate, locally termed "black rash," which is separated from the coal seam by about 3 in. of slate. This "black rash" averages about 25 per cent. ash, but contains some clean coal.

The coal in both seams is quite soft. Cleavage sufficiently well defined to exert an influence on the mining is rare. Overlying strata arc slates, shales, and sandstones, the sandstones predominating, extending from 30 to 1000 ft. above No. 3 seam. The strata directly above the draw slate of No. 3 seam, or "black rash" of No. 4 seam, and up to the first sandstone are usually slates or shales, about 10 ft. thick over the

* Chief Engineer, Pocahontas Coal & Coke Co.

thicker coal, decreasing in thickness as the coal becomes thinner, and
sometimes disappearing altogether. The dip of the seams is usually
sufficient to provide natural drainage, although there are many undula-
tions, swags, and basins requiring pumping and ditching. Explosive
gas is rarely found except in the slope and shaft workings. The land as
a rule, including the surface, is owned by land-holding companies and
leased to operating companies on a royalty basis.

SYSTEMS OF MINING

The room-and-pillar system of mining has been used from the begin-
ning; the principal changes in the manner of its adaptation as mining
progressed are shown by the accompanying sketches. Fig. 1 is taken
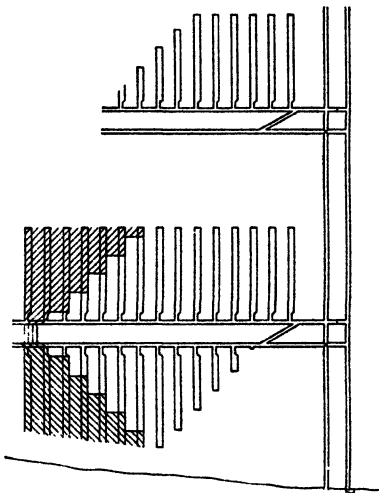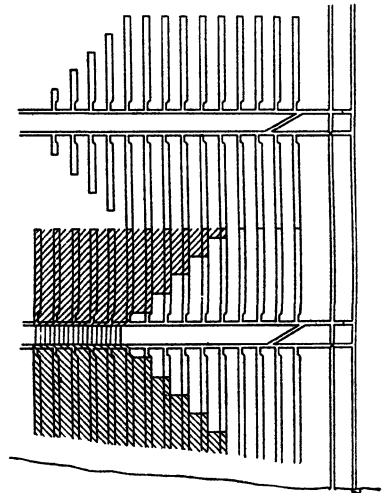
FIG. 1.   FIG. 2.

from the system of mining that was made a part of many of the earlier
leases. It provided that entries and break-throughs should be 10 ft.
(3 m.) wide, entry pillars 60 ft. (18 m.) wide, rooms 18 ft. (5.5 m.) wide,
and room pillars 42 ft. (12.8 m.) wide; that rooms from one entry of a
pair should be turned and worked out progressively on the advance; that
rooms from the other entry should be turned and worked out progres-
sively on the retreat; and that pillar mining should be deferred until
the entries had been driven the intended distance and should then begin
in the last pillars and be conducted on the retreat in both sets of
pillars, the chain pillar of the entry being mined at the same time. In
but few instances was this system carried out, apparently because of
the desire for immediate tonnage.

The first change was to work out rooms from both entries on the

advance (Fig. 2); with few exceptions this system was followed for a number of years. The first mines were in the thicker coal and it was considered good practice to leave from 18 in. to 3 ft. of top coal to protect the roof and reduce the amount of timbering otherwise necessary. It was expected that a large part of this top coal would be recovered when the pillars were mined. But rooms were driven wider than had been anticipated, room pillars were correspondingly thinner, and there were many falls and squeezes and few barriers, other than main entry pillars, to prevent the spread of squeezes. In many instances pillar mining was postponed for indefinite periods and much pillar coal and all of the top coal was lost. Marked improvement was soon noticeable in
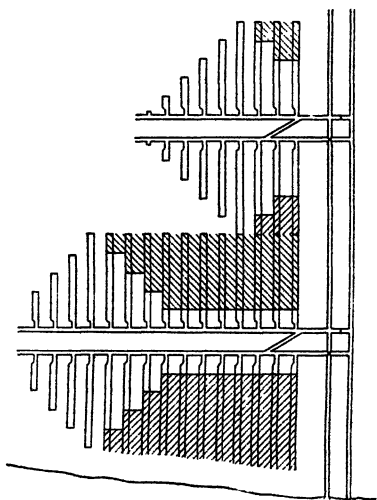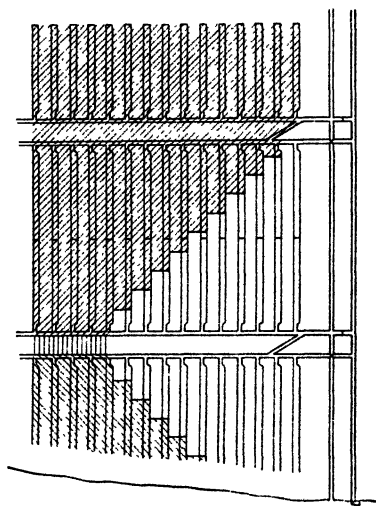
FIG. 3.   FIG. 4.

the spacing and widths of rooms, but top coal was left wherever the seam was thick enough to permit it. Only a small part of the coal so left has been recovered except over the roadways. When pillars were mined, the V-shaped break line caused additional weight upon the chain pillar at the apex of the V, which was aggravated by a tendency for the mining of the chain pillar to lag and frequently resulted in crushing portions of the chain pillar and the ends of room pillars. Usually but a small recovery was made of the entry stumps and chain pillar and in many instances they were not mined.

An effort was made to avoid the losses and inconvenience caused by the delay in pillar mining by mining pillars on the advance, as shown in Fig. 3; but, unless the rooms were unusually long and the cover light, the area opened was not sufficient to cause a break of the overlying sandstones and a squeeze usually followed. This caused a loss of the

entry stumps and chain pillar and the squeeze spread to adjacent areas when opened.

Longer and more uniform break lines were obtained by postponing pillar mining until it could continue in one direction over a larger area, as shown in Fig. 4, or, when the grades were favorable, as in Fig. 5. Frequent changes in the direction of the break line were thus avoided, and the extent and shape of the area mined were soon such as to cause a good break of the sandstones. But the orderly advance of pillar mining was often interrupted by falls, poor drainage, or squeezes in the long standing pillars, with consequent losses.

Rooms from both entries of a pair not only caused large areas to be
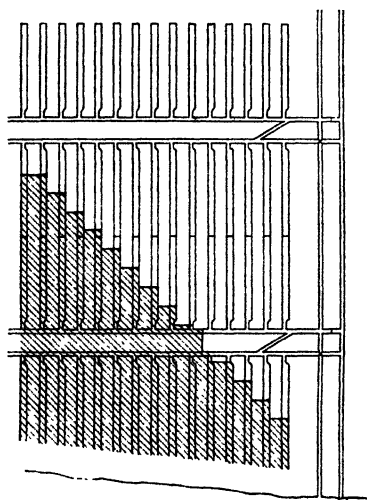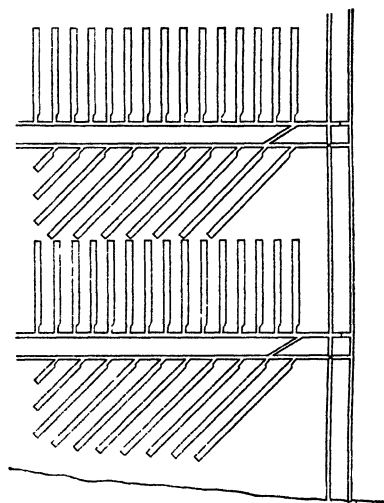
FIG. 5.   FIG. 6.

opened far in advance of pillar mining, with the probability of losses, as stated, but frequently required that one set of rooms be worked to the dip, or else at an acute angle with the entry, in an attempt to keep above the strike line, as shown in Fig. 6. Progress of dip rooms was usually delayed by water while changes in direction of rooms caused irregular break lines in pillar mining. The working of rooms from both entries of a pair was therefore discontinued, as a general practice, in favor of working rooms from one entry only (see Figs. 7 and 8), and although many mines continued to turn and work all rooms on the advance, others first drove room entries to completion and then turned and worked out the rooms progressively on the retreat, followed promptly by pillar mining as indicated.

There were other changes from time to time and a variety of combina-

tions as regards width, length, and spacing of rooms, but experience and a study of methods of other fields had shown that economical pillar mining and a maximum recovery of coal required that rooms be opened only fast enough to provide for the uninterrupted advance of pillar mining; that pillars be mined promptly upon completion of rooms; that the line of pillar mining be carried forward uniformly in one direction until sufficient area was completely mined out to cause a good break of the overlying sandstones; that the direction of the break line be varied as little as possible, within reasonable limits; and that barrier and other pillars be of sufficient size not only to protect the workings properly but to admit of their being satisfactorily mined later. Most new development since 1900 has been laid out with these principles in mind.
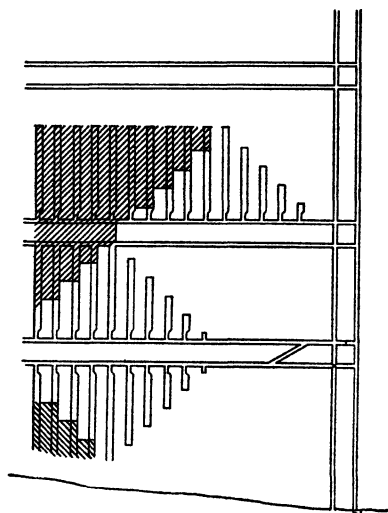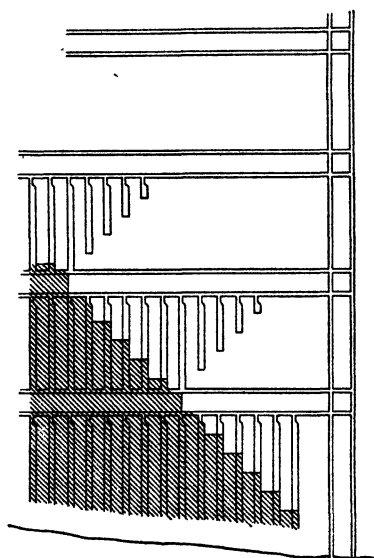
FIG. 7.   FIG. 8.

Fig. 9, which has been published several times,[1] was prepared in 1907 by one of the land-holding companies to show three prevailing adaptations of the panel system in general use in many mines, but usually modified as to width, length, and spacing of rooms and in various other ways

[1] H. H. Stoek: Pocahontas Region Mining Methods. *Mines and Minerals* (1909) **29**, 395.

Audley H. Stow: Mining in the Pocahontas Field. *Coal Age* (1913) **3**, 594.

W. H. Grady: Some Details of Mining Methods with Special Reference to the Maximum of Recovery, W. Va. Coal Min. Inst., Dec., 1913. *Coal Age* (1913) **5**, 156.

C. M. Young: Percentage of Extraction of Bituminous Coal with Special Reference to Illinois Conditions, Univ. of Ill. *Bull.* **42**, (1917).

to conform to local conditions, tonnage requirements, and individual preferences.

In panel No. 1, rooms are driven on the third cross entry as soon as their position is reached and pillar mining is begun as soon as the second room is completed. Pillars are mined on the advance on the second and third cross entries to within 100 ft. of the second cross entry. On the first cross entry, the last room is driven first and pillars are mined on the retreat, including the barrier pillar left on the second cross entry.
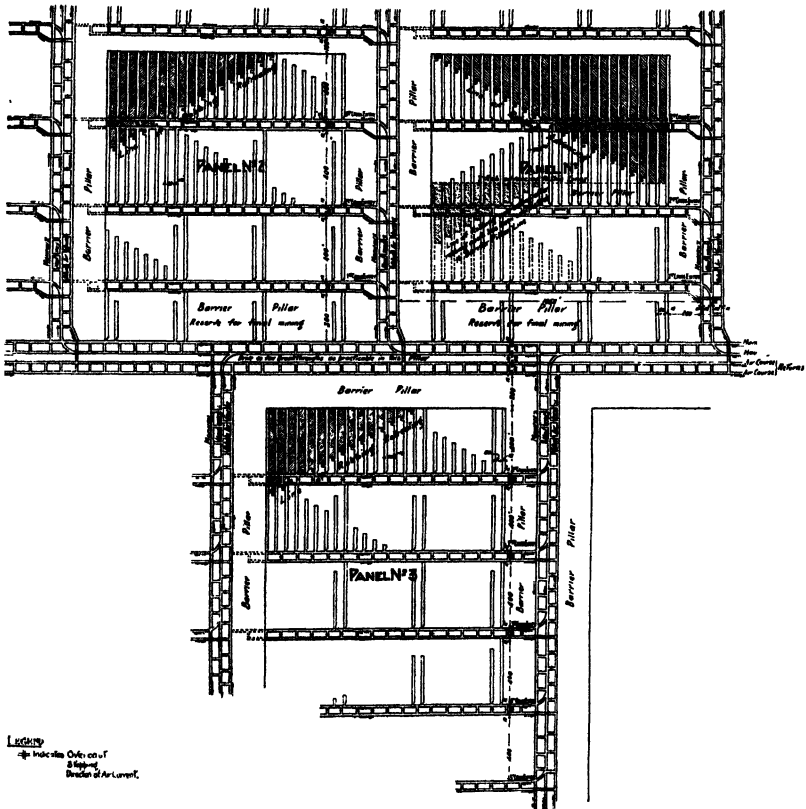
FIG. 9.—ADAPTATIONS OF THE PANEL SYSTEM.

In panel No. 2, the entries are driven to the limit before rooms are turned off, except an occasional pair of rooms as shown. The last room on the third cross entry is turned off first and pillar mining is begun at inside corner of the panel. Rooms are developed only fast enough to keep in advance of the pillar mining.

In panel No. 3, the entries are driven to the limit before rooms are turned off, except as shown. The last room on the first cross entry is turned off first and pillar mining is begun as soon as the second room is

completed. Rooms are developed only fast enough to keep in advance of the pillar mining.

Where pillar mining follows promptly, rooms from 15 to 24 ft. wide, (4.5 to 7.3 m.) depending on the character of the top, and spaced 60 ft. center to center provide pillars of sufficient size to permit successful pillar mining under any conditions of roof or cover yet encountered; hence these widths and spacing are commonly used, although there are many cases of greater spacing and some of greater widths. Satisfactory results have been obtained, where delay can be avoided, with rooms 36 ft. wide, spaced 90 ft. center to center, 800 ft. long, having two tracks in a room, but under good roof, well and systematically timbered, and with a high degree of supervision. Under present systems, from 40 to 55 per cent. of the coal area opened up by the rooms and room entries is taken in the first mining.

The procedure indicated by panel No. 1 has been enlarged upon by continuing the entries indefinitely, as shown in a general way by Fig. 10, and described by W. H. Grady.[2]

RECOVERIES OBTAINED

Several land-holding companies and some operating companies, some annually and others every two years, determine the quantity of coal recovered from each mine per acre of coal area exhausted, as taken from the mine maps, and calculate the percentage of recovery by comparing the tonnage mined with the theoretical tonnage contained in the same area, and also calculate the average recovery from beginning of operations to the end of the corresponding period. Theoretical tonnage is estimated on the basis of 1 gross ton per cubic yard of solid coal, or 1613.3 gross tons per acre-foot, corresponding to a specific gravity of 1.328. The specific gravity of Pocahontas coal varies from 1.298 to 1.330. Information shown by Figs. 11 and 12 is taken from the records of such companies, as prepared and carried forward from year to year since 1890.

Fig. 11 shows graphically the average thickness of coal, the average recovery, in gross tons per acre-foot and also in percentage, and the proportion of pillar mining, in 67 mines or groups of mines representing 94 per cent. of the total production of the Pocahontas field to the end of the year 1920. The arrangement of mines or groups of mines is about in the order in which shipments were begun. The combined average of the mines opened in each 10-year period and the combined average of all the mines are also shown. A summary of these averages, and of the coal area exhausted by the mines opened in each period, is as follows:

[2] Cost Factors in Coal Production. *Trans.* (1915) **51**, 138.

Mines Opened	Thick- ness of Coal, Feet	Gross Tons Recovered per Acre-foot	Percentage of Recovery	Proportion of Pillar Mining per Cent.	Coal Area Exhausted, Acres	Coal Mined Gross Tons
1883 to 1890..	7.18	1290	80.0	55	10,790	99,960,549
1891 to 1900..	5.56	1411	87.5	49	10,431	81,827,688
1901 to 1910..	5.30	1523	94.4	36	10,261	82,814,237
1911 to 1920..	6.24	1469	91.1	18	725	6,645,162
Average....	6.04	1394	86.4	46	32,207	271,247,636
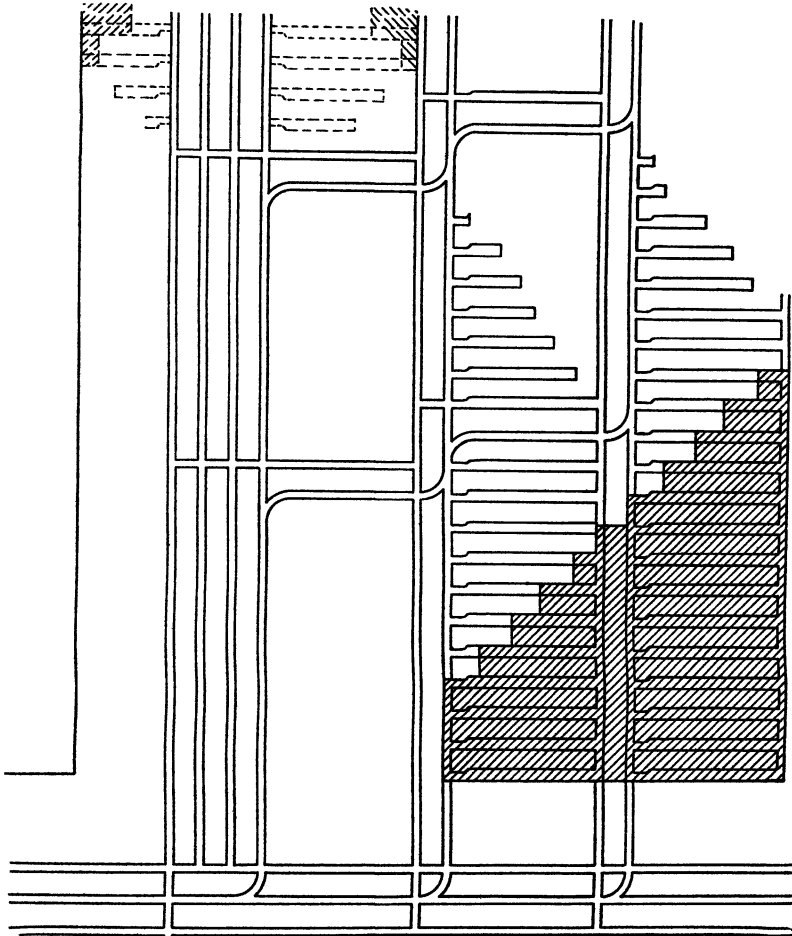
FIG. 10.

Thickness of coal shown, and on which recoveries are based, is the full thickness of clean coal in the seam, except that for a number of years top coal left in first mining was not included in the sections measured in mines 1, 2, 3, 4, 7, 8, 9, 13 and 55, Fig. 11. Thickness of coal shown for

these mines is, therefore, less than the actual thickness and recovery is
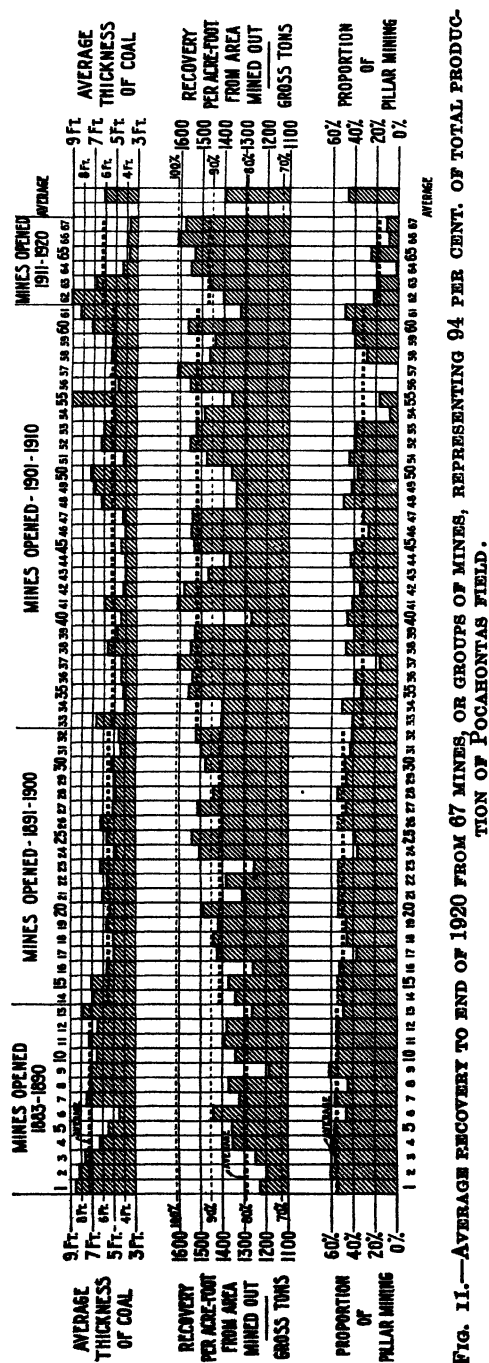
Fig. 11.—Average recovery to end of 1920 from 67 mines, or groups of mines, representing 94 per cent. of total production of Pocahontas field.

correspondingly higher than the actual recovery. On the other hand, coke was manufactured at mines 1 to 41, inclusive, and 48, 49, 50, 52, and 53. Coal used in making coke was estimated at the rate of 1.6 tons of coal per ton of coke shipped, but under the conditions of coke manufacture in the Pocahontas field 1.8 tons would have been a better factor. While this loss is not directly chargeable to recovery in mining, it is included in the recoveries shown.

It may be noted from the thicknesses that the first mines were opened in the thicker coal and that subsequent development, as a rule, extended to thinner coal. It may also be noted that recovery increases as thickness decreases, as indicated by the averages of mines opened in the first three periods, and that recovery decreases as thickness increases, as indicated by the averages of mines opened in the last period.

This increase in recovery may be attributed, in part, to decrease in thickness of coal; in part to improvement in top conditions as the coal becomes thinner, permitting a greater proportion of the coal area to be taken with safety in first mining, as indicated by the decrease in proportion of pillar mining; in

part to the fact that but few mines opened since the year 1900 have manufactured coke; and in part to the changes in systems of mining.

Decreases in recovery, as shown by the average of mines opened in the last period, is on account of the predominating tonnage from the thick-coal mine 62. Top coal has been left in most entries of this mine and has been included in the thickness, but the greater part of this top coal may be recovered. That recovery increases as thickness decreases is emphasized by comparing results at mines 1 to 8, there being little difference in their conditions.

Mine No. 9 is the only mine completed; it shows a recovery of 73.6 per cent. Much of the coal left in first mining was lost on account of very bad top, and about 30 per cent. of the coal produced was made into coke.
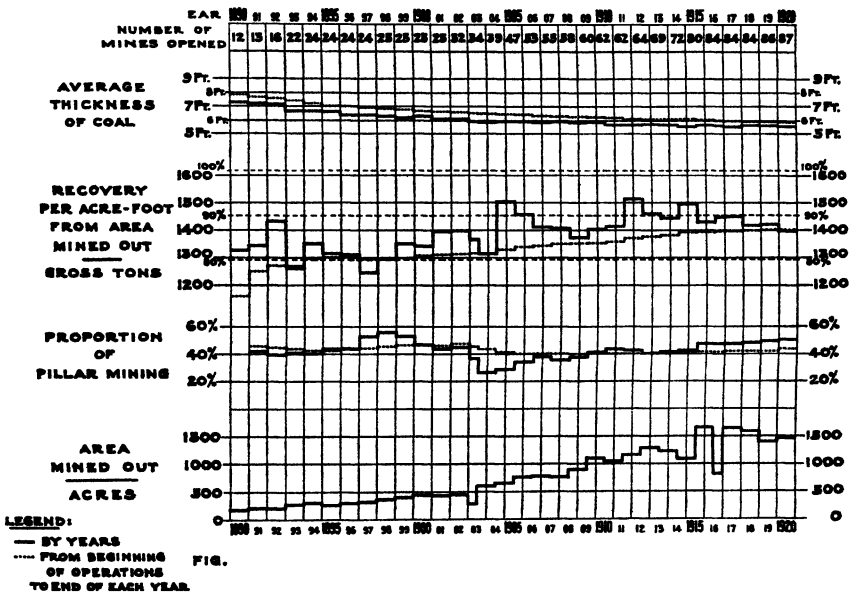
Fig. 12.—Average recovery, by years (1890 to 1920), and from beginning of operations (1883) to end of each year (1890 to 1920) from a group of mines which furnishes 70 per cent. of the total production of the Pocahontas field.

In mine No. 40, the coal was only 4.5 ft. thick and, in spite of excellent supervision and great care, there was a low recovery on account of unusually bad top.

Mines 41, 52, 53, and 60 have thick coal and an excellent recovery. Some of the clean coal in the black-rash top is loaded out in these mines, while only the clean coal of the seam is included in the thickness.

Mines 42, 43, and 44 have about the same thickness of coal and the same proportion of pillar mining but recover 97.9 per cent., 91.2 per cent., and 85.0 per cent., respectively, because of different top conditions.

Mines 50, 55, 62, and 63 are the only ones opened since 1900 in which top coal has been left in first mining. The practice has been discontinued in these mines, except as to entries, and is being gradually discontinued in all mines.

Mines 56 and 57 do not show any pillar mining. The workings are in No. 3 seam and are laid out with wider pillars than usual in order that pillar mining may be deferred until it can follow the pillar mining of No. 4 seam.

Figure 12 shows the average thickness of coal, the average recovery, in gross tons per acre-foot and also in percentage, and the average proportion of pillar mining by years (1890 to 1920), and from the beginning of operations (1883) to the end of each year (1890 to 1920), of all the mines on the property of one of the land-holding companies representing 70 per cent. of the total production of the Pocahontas field to the end of 1920, and including mines opened in other fields since 1915. It shows also the number of mines opened at the end of each year and the area mined each year.

The averages from beginning of operations in 1883 to the end of 1890, 1900, 1910, and 1920 are as follows:

YEAR	THICKNESS OF COAL, FEET	GROSS TONS RECOVERED PER FOOT-ACRE	PERCENTAGE OF RECOVERY	PROPORTION OF PILLAR MINING, PER CENT.	COAL AREA EXHAUSTED, ACRES	COAL MINED, GROSS TONS
1883 to 1890	7.89	1,161	72.0		899.00	8,237,734
1883 to 1900	6.74	1,310	81.2	47	4,054.20	35,786,555
1883 to 1910	6.26	1,358	84.2	41	10,894.95	92,640,961
1883 to 1920	5.90	1,404	87.0	44	25,431.59	210,547,609

It may be noted that, although the recovery, by years, varies greatly, due in part to economic and industrial conditions that prevailed in the field from time to time, and to other causes, the average recovery from the beginning shows a progressive increase.

As to the effect of the character of bottom on recovery, there have been occasional instances of pronounced squeezes in which fireclay bottom would heave in openings, causing serious interruptions to haulage, delay in removing pillars, and consequent losses. Otherwise, the effect of character of bottom on recovery has been negligible.

Some Considerations Affecting Percentage of Extraction in Bituminous Coal Mines in America

By H. H. Stoek,* Urbana, Ill.

A STUDY of American coal-mine practice shows two of its distinctive features to be: A greater number of accidents per thousand employees than in any of the other leading coal-producing countries; a much greater percentage of the coal lost than in any of the other leading coal-producing countries. Although our subject is mainly concerned with the second feature, any consideration of mining methods should include safety, efficiency, and economy or costs. If the argument could be offered that our wasteful mining was necessary on account of safety, and that it is carried out with less loss of life than in other countries, the plea for conservation of coal could not be so strongly urged. But when we are wasteful of both life and property, the subject merits more consideration than it has received.

Some coal operators will doubtless say that the unit of accidents should be the number per million tons produced rather than per thousand employees, for by adopting this unit America probably stands in the lead rather than the lowest among the countries. However, vital statistics are not usually so expressed by other governments, by life and accident insurance companies, or in other industries, and it is merely begging the question and attempting to bolster up a bad condition to adopt a peculiar unit for the statement of our accidents.

For many years the statement was made by various agencies, and was unchallenged, that "For every ton of bituminous coal mined in the United States a ton had been lost in the way of pillars left in the mines." This statement was undoubtedly true for many years, and although in the past decade some progress has been made toward obtaining a higher extraction, and in some districts very decided progress, the loss is still great; and, although no exact data can be given, it is probable that for the United States as a whole at least one ton is lost for every two mined.

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George S. Rice, chief mining engineer, U. S. Bureau of Mines, recently said:[1] "The bald statement that there is a loss of one ton of coal for every two recovered may seem to those not familiar with coal mining to imply inexcusable and wanton waste.   Perhaps a better way of expression is in terms of total recovery rather than in terms of losses.   Many mine superintendents and mine owners will readily concede that their total recovery may be only 60 or 75 per cent., but to state that they are wasting 25 to 40 per cent. arouses antagonism.   The suggestion is of wilful waste, whereas the majority of the operators try to the best of their ability to get a high recovery under the adverse and complex conditions that confront them.   These involve natural, commercial, and labor conditions, and the requirement, in many cases, of boundary pillars under railroads and buildings, which may be called for in the deed or lease, or in the absence of specific exemption in the deed or lease, the common-law requirement that a surface owner is entitled to the support of the surface."

In 1905, the writer compiled[2] a table of percentages of coal recovery secured by correspondence with coal operators in Alabama, Arkansas, Colorado, Illinois, Indian Territory, Iowa, Maryland, and West Virginia. The percentages given varied from 70 to 95, but in the light of later experience, as will be explained later, many of these percentages were probably estimated entirely too high.   In 1914, A. W. Hesse[3] gave percentages of extraction varying from 55 to 97 per cent. for some of the same states, also Michigan and Ohio.   In the bulletins of the Illinois Coal Mining Investigations, in 1915, the extraction for Illinois was given as varying from 41 to 96 per cent., and an average for thirty panel mines gave 55 per cent. recovery and an average of forty-eight pillar mines gave 54 per cent.

The writer is convinced that most of the figures published for extraction are too high, as they are estimates and not based on a close measurement of areas extracted compared with coal hoisted, or they apply only to small areas, as, for instance, a single panel or group of panels, and do not represent the average condition for the entire mine.

In an examination of a number of mines in western Pennsylvania, West Virginia, and Maryland, in 1918, wherever it was possible to secure percentages of extraction based on a measurement of areas excavated as compared with the output hoisted, the figures were found to be from 10 to 15 per cent. below those given by the operating officials at the mines. For instance, the consulting engineer of one large coal company, who has kept detailed figures for all of the mines of that company for a number of years, gave his best yearly figures for extraction as from 85 to 90 per cent.,

[1] Reports of Investigations, U. S. Bureau of Mines (April, 1921.)
[2] *Mines and Minerals* (1905) **26**, 107
[3] Paper before West Virginia Mining Institute.

but when the mines were visited the local superintendents and fore-men declared that "Every pound of coal is being taken out" or that "Undoubtedly the extraction is from 95 to 98 per cent.," etc.

In 1914–15, a committee reported that in a certain mining district about 65 per cent. of the coal was being recovered. A subsequent resur-vey of twelve of the largest mines in the same district made by a number of engineers gave percentages of extraction varying from 37.7 to 49.5 or an average of 41.4 per cent. The original amount of coal was obtained by carefully measuring a large number of sections of the coal bed and then measuring the area excavated, as shown by the mine maps with the plani-meter; this amount was then divided by the amount of coal hoisted and as the oldest mine had been operating only about ten years the total output was easily obtainable. One reason for the discrepancy between these figures and those previously reported was that in the earlier report only the thickness of the seam as mined was considered and no account was taken of the fact that in most of the mines 1½ ft. (0.5 m.) of top coal is left up to protect the shale roof. The average thickness of coal mined was 7.46 ft. for the twelve mines visited, which gives a loss of about 16 per cent. which was not included in the first estimates. In these mines, it was "hoped" when the above survey was made that much of the pillar coal would be subsequently recovered, but during a period of six years very little of it has been mined and the indications are that very little will be in the future, for the mines are all large producers and it will not be consid-ered profitable to reduce the daily output by interfering with the normal haulage system, as would be necessary if an attempt were to be made to haul the relatively small amount of coal that could be gotten from such pillars as may now be available for mining. Unless pillars are recovered quickly and pillar drawing forms a definite part of the initial planning, very little pillar coal will be recovered. In other words, many coal opera-tors are deceiving themselves by counting on a recovery of pillars at some indefinite date in the future.

American mining engineers and coal operators have shown that they can secure as good results, in the way of recovery, as are obtained in any foreign countries where the economic conditions permit. The reason this has not been done more generally has not been due to the technical difficulties, but to real or assumed commercial and labor conditions, such as very low cost of the raw material, which is not conducive to saving; easy operating conditions, which have made the number of mines to be greatly in excess of those needed for normal consumption and have thus given irregular work and too much competition; unfortunate agreements between operators and miners that have made changes in existing methods difficult to install for fear of strikes or a change in the wage scale.

· Where land containing coal 8 ft. thick, or about 13,000 tons per acre, has been bought for from $5 to $30 per acre, or even $100, the value

per ton in the ground is too often considered negligible and does not form a potent conservation argument. The greatest advances in coal extraction have been made where the end of a given deposit is in sight, or the value of the coal lands is quoted in thousands rather than hundreds of dollars per acre, as in the Connellsville district of western Pennsylvania.

Peele in his "Mining Engineers' Handbook" says "A mining method is rarely ideal from all standpoints. It is generally a compromise between conflicting factors." The factors that should be considered in any choice of a method may be classified under several heads:

First, safety.

Second, geological conditions, such as: Thickness, extent, inclination, and uniformity of the coal bed; hardness; cleat and partings of the coal; presence of dips, faults, gas, water, etc.; depth of cover; character of overlying and underlying rocks; surface topography, including the location of bodies of water, sand, etc.; climate and geographical location, etc.

Third, operating and financial conditions, such as: Laws requiring surface protection against subsidence and public opinion against subsidence; amount of timber available, or amount of material such as sand for filling; character of labor available, whether skilled or unskilled, organized or unorganized; if organized what the agreement is between operators and miners; character of management; amount of capital available and the necessity of obtaining a quick return or of working for the future as well as for the present; possibility of maintaining a haulage system for a given output that will keep the mine busy; disposition of the product: Is it necessary to keep the coal in lump form as much as possible for commercial purposes or is it to be used for coking so that crushing in the mine is immaterial?

Safer working conditions are steadily receiving more attention because of increased activity in inspection by state, company, and compensation insurance inspectors.

The geological conditions are generally favorable for systematic working and the natural conditions are much more favorable in nearly all of the principal American coal fields than in the European coal fields, where the accidents are much fewer and the extraction much greater. In spite of the opinion of many operating men to the contrary, the writer believes that there are few places in the chief producing coal fields of the United States where the natural conditions require the present wasteful methods. Pillars are now being successfully drawn under practically all geological conditions that are considered, in other regions, to be such as to preclude successful pillar drawing.

The financial condition that requires a quick return on the investment is undoubtedly one of the chief causes that leads to the use of tried methods giving certain results rather than to try methods having an element of uncertainty. This was well illustrated in southern Illinois.

At the beginning of the World War, a number of coal companies were either experimenting with pillar drawing or studying the problem preparatory to trying some method. The war demands for a maximum output naturally put an end to all experimentation and untried methods and the after-war period has offered no inducement to resume the experiments, because of financial and labor troubles.

The labor conditions in one section of the country are frequently not understood by residents in another section. An operator in a non-union Pennsylvania district can easily suggest how he would improve conditions if he were mining, say, in Illinois, but when he buys an Illinois mine he usually finds that the "agreement" interposes many unthought of obstacles to his plans for improvement. Many mines have been opened without adequate preliminary study of the local conditions and without any definite projection plans for the future operation of the mine. The only reason for the method adopted was that it was one with which the parties opening the mine were familiar when operating elsewhere.

Methods of mining have followed the migration of the miners from one district to another. As most of our pioneer miners were from England, Scotland, and Wales, the early methods were based on the methods used in those countries, too often with little regard for the local underground conditions. Then, as men moved from one mining region in the United States to a newer field, the methods of the older region were used in the new. Such a procedure is to be expected, because, owing to the unknown nature of underground conditions, men are naturally loath to experiment and prefer to go ahead along lines with which they are familiar.

A number of methods of drawing pillars have been successfully worked out within the past 10 or 15 years, particularly in the Appalachian coal fields where the increasing value of coal lands underlaid by the high-grade Pittsburgh seam has made it imperative that the remaining resources be husbanded. These methods have been described in detail in the several coal journals, in Bulletin No. 100 of the Engineering Experiment Station, University of Illinois, in the transactions of the several coal mining institutes, and particularly in a paper by A. W. Hesse read before the Coal Mining Institute of America in December, 1919.[4] Space will not permit

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[4] *Proc.* Coal Mining Institute of America (1919) 106 and *Coal Industry*, December (1919) **2, 543.** See also description of pillar drawing as follows:

Patrick Mullen: New Mining Methods as Practiced by the H. C. Frick Coke Co. *Proc.* Engrs. Soc. W. Pa. (1916) **32,** 714; *Coal Age* (1916) **10,** 700.

W. H. Howarth: Mining by Concentration Method. *Proc.* Coal Min. Inst. Amer. (Dec. 22, 1916); *Coal Age* (1916), **9,** 125.

F. W. Cunningham: Best Method of Removing Coal Pillars. *Proc.* Coal Min. Inst. Amer. (1910) 275; (1911) 35.

W. L. Affelder: Rib Drawing by Machinery. *Proc.* Coal Min. Inst. Amer. (1912) 232.

a detailed discussion of these methods, but a study of the literature on better coal recovery and a field study of successful methods will, I believe, confirm the following conclusions and suggestions:

1. In considering the applicability of a method of extraction all of the conditions must be considered. For instance, the concentration method of the H. C. Frick Coke Co. used in many parts of the Connellsville district, where the geological conditions are similar to those of Illinois in most respects, is carried on under the following operating conditions that are quite different from those that prevail in most of the Middle West coal fields: (a) The coal is intended for coking and hence the more it is crushed the better; (b) the labor employed is non-union; (c) the men are shifted from place to place in the mine and work thus concentrated at points where desired, but in the Middle West the practice prevails of one man for one room in which he expects to work and not to be shifted from place to place; (d) there is no yardage and narrow work and different gangs of men are employed for mining, shooting, timbering, and loading.

An essential consideration for high recovery is a mobile labor force, which permits concentration of miners; a rapid removal of the coal is also necessary.

2. While there is a similarity in the successful methods used for obtaining high percentages of recovery, no standard and universal method has been developed; there are, however, a number of methods that adapt the same general principles to a wide variation of local conditions.

3. The best results are obtained by companies that mine under a definite projection plan for the future and are not dependent on immediate financial returns. The best result will undoubtedly be obtained by driving entries to the boundries of the coal area and drawing the pillars full retreating. Very good results are being obtained by drawing pillars within panels by driving the panel entries to the limit of the panel, then beginning the rooms at the inbye end of the entry so that pillar drawing may begin as soon as the inbye room is finished. In many cases, the rooms are driven and pillar drawing started just inside the barrier pillar along the main or cross entry as the entries progress inbye. Many attempts to draw pillars have been unsystematic and have failed completely or have given unsatisfactory results. On such unsystematic attempts

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W. A. Stillman: Big Pillar System of Mining. *Proc. Coal Min. Inst. Amer.* (1911) 76.

H. V. Hesse: Maximum Recovery of Coal. *Proc. W. Va. Coal Min. Inst.* (1908) 75; and *Mines and Minerals* (1908) 29, 373.

H. H. Stoek: Pocahontas Region Mining Methods. *Mines and Minerals* (1905) 29, 395.

Audley H. Stow: Mining in the Pocahontas Field. *Coal Age* (1913) 3, 594.

W. H. Grady: Some Details of Mining Methods with Special Reference to the Maximum of Recovery. *Proc. W. Va. Coal Min. Inst.* (December, 1913); *Coal Age* (1913) 5, 156.

are based many of the prejudices regarding the practicability of pillar drawing. Pillar drawing should not be attempted unless a definite plan has been worked out and the company is willing to give the plan a fair trial, extending over a period possibly of several years.

4. The successful methods for obtaining a high recovery get most of the coal from the pillars and not from the rooms, as in ordinary room-and-pillar work. The development of the present practice has been by a gradual widening of the pillars, sometimes accompanied by a decrease in room width, but not always. Where best results are now being obtained rooms from 12 to 20 ft. (3.7 to 6 m.) wide are driven upon 90 to 100-ft. (27 to 30-m.) centers, thus leaving a pillar from 80 to 88 ft. wide. The width of room seems to depend more on whether or not yardage must be paid rather than on the nature of the roof. A number of companies are still using 60-ft. centers with narrow rooms with fairly good results, but the operating officials have frequently expressed the opinion that they would get better results if larger pillars were used.

5. For successful pillar work, strain on the roof above the rooms must not be great enough to cause the roof to fall before the pillar coal is removed or to make the cost of timbering such a roof prohibitive. The fall of roof in the rooms may be overcome by making narrow rooms, but the very narrow rooms used in some districts do not seem to be necessary; success in pillar drawing is quoted with rooms 18 to 22 ft. wide, which is above the ordinary yardage limit. Strain on the pillars must not be great enough to cause squeezing and the roof must be of such a character that it will break and not bring undue strain on the retreating pillar face.

6. A fundamental principle in pillar work, to prevent squeezing the coal back of the working face, is to take out as nearly as possible all of the coal so as to secure a roof fall as near as possible to the face of the pillar. Small stumps are worse than useless as they retard roof breaks and cause the pressure to ride back upon the pillars. Great care is taken, therefore, not to leave stumps of coal in the gob, even though the coal recovered may not pay for the work of removing the stump, or may have to be abandoned in the gob.

7. It has been found wise, generally, to protect the men driving through pillars by a narrow curtain pillar between the working place and the fallen gob, this curtain pillar being removed in great part as the track is taken up in the working place.

8. Subsidence of the surface should be considered a probable accompaniment of pillar drawing and the pillars should be removed systematically so that the resulting subsidence may occur uniformly and not in isolated spots, for, although there may be a temporary disturbance of the surface with systematic pillar drawing, the surface conditions after a short time may be as good or nearly as good as before the pillar drawing.

The question of leaving adequate support for the surface is being

seriously considered by the railroads in connection with the valuation of their rights-of-way where such are underlaid by coal which is owned by the railroads or has been leased to coal companies. In a circular in the report of the Sub-Committee of the President's Conference Committee upon the "Federal Valuation of Railroads in the United States" dated Nov. 1, 1921, one of the essential factors noted in connection with the valuation of the right-of-way underlaid by coal is, "The amount of coal (expressed in percentage) which is necessary to remain in the ground to support the surface." The following table is suggested in the circular:

Depth Below Surface, Feet	Amount Necessary to Support Surface Per Cent.
0 to 50	100
50 to 100	70
100 to 200	60
200 and deeper	50

This table is open to serious objections, as it takes no account of th character of the overlying material, and the results in Illinois, at least, show that 50 per cent. of coal left at a depth of 300 ft. and more may not prevent subsidence.

In the near future, the question of proper support for public highways may also arise in view of the great sums being spent for hard roads throughout the United States.

9. If the mine is at all gaseous, permissible explosives and electric safety lamps only should be used in the pillar work as the men necessarily work near the old gobs where falls are constantly occurring.

10. Although statistics are not available regarding the relative numbers of accidents in pillar drawing and room work, a number of inspectors connected with liability-insurance companies and operating officials have expressed the opinion that there are no more accidents in pillar work than in ordinary room-and-entry work and that, while such work requires careful workmen, often they are not unusually skilled. As pillar men usually make more money than other miners, such work is frequently preferred after it has been well tried by the miners.

11. Pillar coal does not necessarily mean more fine coal, as is generally assumed, although care must be taken to prevent the making of fines. The increased use of fine coal will remove one of the great obstacles to pillar drawing; viz., the fear that the percentage of slack will be increased. Some persons have expressed the opinion that pillar coal is inferior in quality to room coal, but it is doubtful if this can be substantiated, particularly in the case of large pillars and where the pillar drawing follows closely the first working.

12. It was formerly considered that pillar coal must be hand mined but in recent years, particularly with the large pillars now used in many

regions, the use of machines has proved practicable. Their use, of course, depends on the nature of the coal and of the roof; they are not universally applicable.

13. Undoubtedly one of the great drawbacks to more extensive experimentation in pillar drawing has been the fear of the operator in union districts that any improvement may mean a lengthy controversy, possibly a strike, and most surely a demand for an increase in the mining rate which may make a prohibitive differential against the proposed method.

14. It has been suggested that the Government might set up a regulation requiring certain methods to be used that would give a maximum production under existing conditions, but this has not been tried and most likely it will not be except possibly in connection with some of the federal leases in western lands.

DISCUSSION

(Includes also the paper by Thomas H. Clagett, pp. 294–304)

P. D. BROWNING,* Windber, Pa. (written discussion).—About 7 mi. (11 km.) southeast of Johnstown, Pa., the mining community of Windber lies on the border line of Cambria and Somerset counties at an approximate elevation of 1700 ft. (518 m.). The coal field is a part of the great basin embraced in the Appalachian plateau and the mining operations of this district are on the southeastern flank.

Paint Creek and its tributaries divide the Windber field into two parts. South of the creek, for a number of miles, erosion has left the coal seams in isolated areas varying in extent from a few to 2000 acres (809 ha.). North of Paint Creek and the community of Windber, the holdings of the company lie in a well-defined spoon-shaped trough, known as the Wilmore syncline. Its axis is about north 30° east and pitches approximately 2 per cent. to the north. In the isolated coal areas south of Paint Creek, the measures lie comparatively flat, except in occasional places where grades up to 5 or 6 per cent. are encountered. In the Wilmore syncline, the dip of the beds on the eastern limb or flank varies from 10 per cent., near the outcrop, to ½ per cent. in the trough, the transition being quite gradual as the basin is approached. On the western limb of the syncline, the mine workings disclose dips as great as 6 and 8 per cent. In driving parallel with the axis of the trough, grades up to 5 and 6 per cent. are encountered. The measures of the Wilmore syncline continue, over the Ebensburg anticline on the west and form the Johnstown syncline, in which large mining operations have been conducted in Cambria and Somerset counties for a number of years. ·

Mining Engineer, Berwind-White Coal Mining Co.

The only seam of coal that has been mined to any great extent by the Berwind-White operations is the Lower Kittanning, or B, the thickness of which varies from 3 to 4 ft. (0.9 to 1.2 m.), 3 ft. 8 in. being about the average. The coal is friable and has a characteristic columnar fracture but no well-defined cleats. Classified in Pool No. 1 and on the United States Navy acceptable list, it has an excellent reputation as a high-grade fuel.

Because of numerous rolls in the bottom and the ease with which the coal can be mined by pick, mining machines are not in general use, although shortwall chain machines are employed when conditions warrant. Clay veins are frequently met and several well-defined and rather extensive so-called faulted areas have been disclosed by drilling and mining operations.

The immediate roof of the seam is usually a shale, which varies from 3 to 15 ft. in thickness. Above this is a massive sandstone which frequently is found just over the coal. The floor is a hard fireclay several feet thick, which disintegrates upon exposure to air or water. Below the fireclay is a hard shale. Experience has shown that it is better to lift bottom than to take down roof in the entries, the overlying strata causing less trouble if it remains intact on the advance.

In general, little difficulty is experienced on account of roof conditions in that portion of the field lying south of Paint Creek, where the coal seams are in isolated areas. Here the maximum cover overlying the Lower Kittanning seam is approximately 275 ft. (84 m.); north of Windber, in the Wilmore syncline, the situation is similar until from 400 to 500 ft. (122 to 152 m.) of cover is attained. At present, many of the workings are under from 700 to 900 ft. of cover, and considerable difficulty is experienced because of unfavorable roof conditions. Entries driven for development frequently cave for great distances without any warning, the roof cutting or shearing off at the ribs to a height of 10 or 15 ft. (3 to 4.5 m.) above the coal. Unless ample pillars are left under this heavy cover and a well-defined break line is established in retreat work, the bottom is apt to heave, causing considerable annoyance and expense.

The room-and-pillar panel system of mining is employed. The mine openings are considered drifts although most of the main entries dip about 2 per cent. from the pit mouth to the face. The direction of the main entries depends on the territory to be mined, the grades encountered, and other factors. Panel entries are usually turned at right angles to the main entries at intervals of about 1800 ft. (549 m.), and room entries are broken off the panels every 350 ft. (107 m.). The rooms are turned on 60-ft. centers and driven from 18 to 24 ft. wide, depending on the cover and general roof conditions. Ample barrier pillars are left along the main and panel entries, from 30 to 35 per cent. of the coal being mined on the advance. The workings are projected

and driven in such a manner that the rooms hole through to gob, not more than five or six places being driven at one time. A well-defined break line is established in retreat work. Whenever possible, the panels are driven to the limit and the mining done on full retreat.

The question naturally arises as to what percentage of recovery is obtained under the conditions outlined. One of the first mines developed in the field was situated south of Paint Creek and embraced a merchantable coal area of 550 acres (223 ha.) of the Lower Kittanning seam. Based on the entire life of the mine, which was approximately 14 years, the recovery is shown to be 81 per cent. A similar operation in the same part of the field, but having 770 acres of merchantable coal, shows a recovery of 85 per cent. Both mines were opened at the same time but the second mine was operated 7 years longer than the first. The maximum cover overlying the smaller coal area was approximately 125 ft. (38 m.), the character of the roof, the drainage and other conditions being almost ideal for efficient mining. The strata overlying the larger area attained a maximum thickness of approximately 175 ft. The roof conditions were not nearly so good as in the other mine, the drainage, etc., also were less favorable. That an increased percentage of recovery was obtained under these circumstances is attributed to more efficient methods of mining in the later years of its operation, when the other mine had been exhausted and abandoned.

Based on a period covering the last 12 years, three of the largest mines in the northern part of the field show recoveries of 85, 87, and 89 per cent., respectively. As would be expected, the highest recovery was obtained in the mine where the most favorable conditions exist. During the time mentioned, the mine workings have been under cover which gradually increased from approximately 400 to 700 and even 900 ft. It is thought that the recoveries mentioned, which were obtained under conditions much less favorable than those encountered in the southern part of the field, speak well for the system of mining employed and the supervision given to the work.

Because of careful adherence to the system of mining outlined, the mine that had a recovery of 87 per cent. during the 12-year period has improved its recovery in the last year or two. During 18 months, approximately 100 acres (41 ha.) of coal were mined, 57 per cent. being from advance workings and 43 per cent. from pillars. The total recovery during that time is shown to be from 92 to 93 per cent. Assuming that a loss of 2 or 3 per cent. occurs in the coal mined on the advance, a recovery of approximately 85 per cent. is indicated in the pillar work.

While several of the coal fields in the United States exceed the recoveries here given, the results of the Windber operations will compare quite favorably with those of other mines working the same seam of coal and encountering similar conditions. The factors that enter into the

problem are so many and so varied that it is difficult to establish any fixed standard of comparison or goal of achievement.

G. L. Cox,* Huntington, W. Va. (written discussion).—The methods used and the recoveries obtained by the Solvay Collieries Co. are as follows:

At the Kingston colliery, located on Paint Creek, in Fayette County, W. Va., the Eagle seam has an average thickness of 60 in., but a soft slate parting from 1 to 12 in. thick occurs 21 in. from the top. The maximum cover is 1000 ft. The top is soft scaly draw slate; posts and caps are required. The bottom is hard slate. A recovery of 95.6 per cent. is obtained on first mining, which is done by the room-and-pillar method. The rooms are driven 16 ft. wide and 300 ft. long on 95-ft. centers, this wide spacing being necessary to protect a seam 120 ft. above. A 150-ft. barrier pillar protects the main entry.

At the Springton colliery, located on Wide Mouth Creek, in Mercer County, W. Va., the No. 3 Pocahontas seam has an average thickness of 4 ft. 3 in. of clean coal. One-third of the mine has a sandstone roof; where the roof is slate, from 4 to 16 in. must be taken down. The bottom is hard slate, which does not heave and makes a good shoveling surface. Mining is done by the three-entry panel, room-and-pillar system, twenty rooms to an entry. The rooms are 20 ft. wide and 300 ft. long, spaced on 60-ft. centers. For 1920, with 65 per cent. retreating and 35 per cent. advance mining, the recovery was 89.7 per cent.

At the Westerly colliery, located on Paint Creek, the Eagle seam has an average thickness of 6 ft. 8 in. At the top of the seam, there are 21 in. of clean coal, which is very much softer than the remaining portion of the seam. The roof is soft scaly slate, which requires posts and caps for its support; the bottom is hard slate, from which the coal easily parts. The maximum cover is 1000 ft. A recovery of 96.6 per cent. is obtained on first mining; pillar mining has not begun. Mining is done by the room-and-pillar system, 16-ft. rooms being driven 300 ft. with 80-ft. centers. Entries, 14 ft. wide, are driven on 50-ft. centers and 150-ft. barrier pillars protect the main entry.

At the Marytown colliery, located on Tug River, in McDowell County, W. Va., the Sewall, or Davy, seam has an average thickness of 3 ft. 5 in. of clean coal. The roof and bottom are hard slate and little timbering is required; the maximum cover is 800 ft. Mining is done by the three-entry panel, pillar-and-room system, with twenty rooms to the entry. Rooms are turned off both sides of the entry on 150-ft. centers and 24 ft. wide and 150 ft. long. With 65 per cent. advance work and 35 per cent. pillar mining, the recovery for 1920 was 92.4 per cent.

The Tolland colliery is on Pond Creek, Pike County, Ky., 4 mi. from Williamson. The Freeburn, or Pond Creek, seam has an average thick-

* Chief Engineer, Solvay Collieries Co.

ness of 4 ft. 8 in. of clean coal, except for 7 in. of laminated coal parting 6 in. from the top. The cover varies from 0 to 800 ft. but averages 500 ft. The roof consists of from 4 to 9 in. of draw slate, then hard slate, and requires timbering; the bottom is hard slate. Mining is done on the three-entry panel, room-and-pillar system, the rooms being driven 20 ft. wide and 300 ft. long on 60-ft. centers, twenty-four rooms to each panel entry. Entries are driven 14 ft. long on 60 ft. centers. The mine was opened in 1917 and 94 per cent. has been recovered on the advance work; pillar mining has not begun.

ERSKINE RAMSAY,* Birmingham, Ala. (written discussion).—Seven companies furnished the following information concerning mining meth-
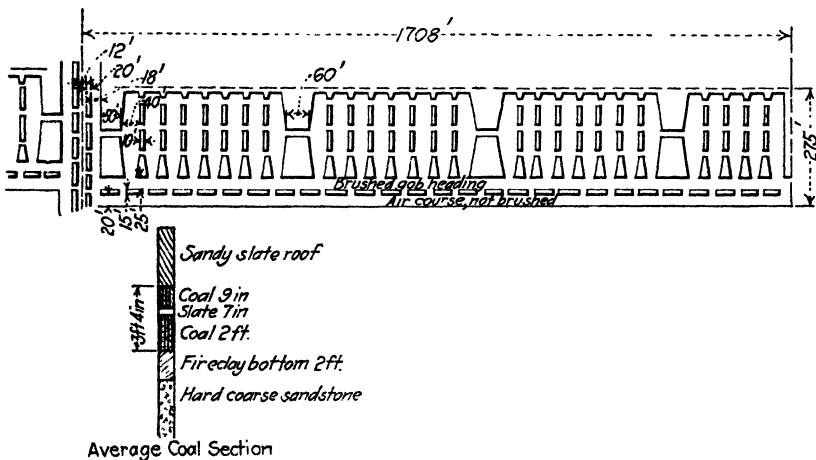
FIG. 13.—ROOM-AND-PILLAR SYSTEM OF MINING.

Where pitch of seam does not exceed 2 per cent., rooms are also turned on air course. Slate, sandy slate, and sandstone cover 200 to 450 ft. Recovery, 65.4 per cent., exclusive of barrier pillars which can be partly recovered at end of life of section.

ods and recoveries in the Alabama mineral district. These reports may be considered representative of the results obtained in this district.

Company A has hard coal with well-defined face and butts. The average pitch is 1 to 2 per cent. southwest and the seam is broken by displacement faults varying from 7 to 50 ft. throw. Intervals of fault are irregular. Seam is of workable thickness in a basin averaging 1¼ mi. wide and averages 3 ft. 4 in. thick including a parting of 7 in. of slate. The cover varies from 35 to 450 ft. A section of seam is shown in Fig. 13.

Mining is done by the room-and-pillar system, the coal being undercut with electric chain machines. Electricity is also used for haulage and pumping. Main headings are brushed; 9 ft. wide and 5 ft. 2 in. above

* Chief Engineer, Pratt Consolidated Coal Co.

rail. The main air course is not brushed; it is 20 ft. wide. The main pillars between heading and air course are 20 ft. wide. The main barrier pillars protecting heading are 50 ft. wide. Cross headings are brushed and gobbed; they are 25 ft. wide and 4 ft. 6 in. above rail. Cross-heading air courses are not brushed; they are 20 ft. wide. Cross-heading pillars are 15 ft. wide. A 60-ft. barrier pillar is left between the eighth and ninth rooms where cover exceeds 150 ft. On first mining, 65.4 per cent. is recovered; 70 to 75 per cent. will be the ultimate recovery.

Company B reports that the coal has well-defined face and butts. The average pitch is about 3 per cent. to the southeast. The seam is not broken with faults and the workable thickness in the basin is continuous for several miles from the outcrop. It has an average thickness of 10 ft., including two partings. Cover varies from 150 to 650 ft. Mining is done by the room-and-pillar system. Coal is undercut with electrical chain machines; haulage on the main slope is by electric motor but on the cross entries by mules. Electricity is used for pumping, and rooms are driven one way up the pitch. Main headings and air courses are driven about 20 ft. wide, the motor roadway taking up 8 ft. of the width and the gob the balance. The main pillar between the heading and the air course averages 36 ft. in width. Main barrier pillars protecting the heading are 175 ft. and room pillars average 36 ft. The main slope is driven about 18 ft. wide and all of the gob is removed; it is double tracked. Chain pillars between the road and the left air course are 50 ft. wide. No pillars have been drawn and about 50 per cent. of the coal is taken.

Company C uses the room-and-pillar method when mining the Blue Creek seam. Rooms are turned quartering the pitch and are driven 27 ft. wide with 15-ft. pillars. The initial recovery is about 65 per cent. Cover begins with 75 ft. and progresses with the depth of the mine. The seam is 8 ft. thick and has two partings 4 to 6 in. each, there being 7 ft. of coal on an average. The top has 8 to 2 in. of bone and the coal is overlaid with 3 to 5 ft. of slate; above that is hard sandstone.

Company D operates three mines. Mine A works the American seam in the basin with about 600 ft. of cover; there is a good slate top overlain with heavy sandstone; the bottom is fireclay. Rooms and pillars are each 40 ft. wide and every seventh room is omitted. Pillars will not be mined until the entries have reached their limit, so the final total yield cannot be given; first working yields 47 per cent. Formerly the mine worked 40-ft. rooms and 30-ft. pillars, but several years ago there was an extended heaving of the fireclay bottom, possibly superinduced by infiltration of water. A chain pillar was left around the squeeze, and the size of the pillars increased. Early in the mine some pillars were taken, producing squeeze; since then none has been drawn. The burden being largely sandstones has not broken, to our knowledge, anywhere in the

mine. Seam is 61 in. thick with a small parting near top and a 10-in. parting near middle. The top coal is hand mined and the bottom bench is shot up.

In mine B, the coal is the top bench of the Mary Lee seam, which is 52 in. thick with a small parting near the top and a 6-in. parting in the middle. Coal is undercut with shortwall machines and shot down. Rooms are generally 40 ft. wide with 10 to 20-ft. pillars, which are drawn when the room is driven up to the pillar. As entries have not reached their limit, the ultimate yield cannot be stated as entry stumps and pillars have not been pulled. First working yields probably 85 per cent. in the rooms. The cover is from 100 to 200 ft. of good slate top and the slate bottom forms the parting between the upper and lower benches.

Mine C is a drift in the top of a long narrow ridge, probably Black Creek coal. The maximum cover is probably 100 ft. The top is a rotten draw slate, which is taken down in all roadways. The bottom has a 5-in. rash with fireclay underneath. The coal is 30 in. thick and clean. Rooms are driven about 20 ft. wide with 8 to 10-ft. pillars which are taken when the room is driven up. The yield is variable as the coal gets thin and is generally not worked under 24 in., also the draw slate sometimes thickens beyond the economic point. The territory also has numerous small faults. These considerations, together with outcrops on two to three sides of mines on account of ravines cutting in, hold the general yield down to about 40 per cent., which cannot be bettered. Where conditions are good probably 85 per cent. is obtained.

Company E operates two mines. In one mine, where the coal is small and hard, electric machines undercut the coal, which is carried to the entries by conveyors. The mine is worked by the longwall system. Another mine is worked by the room-and-pillar system, taking out 80 ft. in advancing, by driving two 30-ft. rooms and then drawing back the 20-ft. pillar between them. An 80-ft. pillar is being left which will be gotten in retreating by driving a 40-ft. room up the middle and drawing back a 20-ft. pillar on each side.

There is a slate and sandstone cover of 500 to 600 ft. at the deepest part of our present working. It is impossible to figure the recovery for the mines are young, but in each operation it should be exceptionally good, and at least 90 per cent. is expected.

All development work of company F is done on the room-and-pillar system, 30-ft. rooms being worked on 60-ft. centers until rooms are driven up 200 ft. deep. The pillar is then pulled back to the first crosscut, which is about 60 ft. from the entry, when roof conditions will permit, and when an entry is finished, entry pillars and room stumps are pulled. The recovery is about 70 per cent. Thickness of seam averages about 33 in. and the depth of overlying cover will run from 700 to 1000 ft.; the pitch of seam varies from 3 to 25 per cent. Rooms are worked with

two tracks, one up each rib, and pulleys and ropes are used in rooms where the grades are sufficient for the loaded car to pull the empty up to face of the room as it is being lowered to the entry. Overlying the coal seam is about 4 in. of draw slate, then either a sandy shale or sandstone several feet thick. The bottom is soft shale or fireclay about 4 in. thick. All coal is shot from the solid, permissible powder being used.

All operations of company G are in the Cahaba Field on pitching seams. Where the pitch is not too great, entries are turned on 250 to 300-ft. centers on either side of the slope; an air course is driven on the lower side of the entries with 12 to 16-ft. pillars between the entries and the air course. Rooms are turned on the upper side of the entries and a track is laid to the face of the coal; mules are used for pulling the cars to the face. The rooms are from 35 to 40 ft. wide, with 16 to 20-ft. pillars. Rooms are worked in blocks of eight or ten with a 100-ft. pillar between the sets. All room pillars are then drawn as soon as a set of rooms is completed. The large pillar is left until the entry is practically worked out, then rooms are turned in the 100-ft. block. By this means the mine is not subjected to any creeps or squeezes.

The deepest operation, the slopes, are in 1¼ mi. with a maximum cover of 800 ft. The slopes on the heavy pitches, the entries, and the air courses are driven similar to the slopes described. The rooms are turned about the same width and same pillars and worked in a similar manner except that chutes are used in place of track. All of the mines generate gas and are ventilated on the split system, the air current being carried to the working face of each room by means of curtains and brattices; the entry is the intake and the air course the return. The coal from all of the operations is dirty and requires preparation by being sized; the larger sizes are hand picked on traveling belts and the smaller sizes are washed.

FRANK HAAS,* Fairmont, W. Va. (written discussion).—In coal mining, as in other industrial lines, a combination of labor and capital properly applied is required for success. Probably, however, no business comparable in magnitude with coal mining requires so large a percentage of labor, as indicated by the total cost, and fluctuations in quantity and price of labor affect the industry more than they do any other business. Should an equation be constructed by which the economic equilibrium of coal mining could be estimated, it will be found to contain two basic variables—the value of money and the price of labor. There is no constant that will establish the relation between these variables for any considerable period of time and no tendency to create such a constant relation except the elusive law of supply and demand. Keeping in mind the fact that the price of labor is the predominating factor, it is interesting

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* Consulting Engineer, The Consoliidation Coal Co.

to note that the records of one portion of West Virginia show that from 1913 to the present time general inside labor of coal mining has increased 294 per cent., while the value of money has increased from 25 per cent. to 33 per cent. Should conditions indicated by these increases prevail for a long period, and the fact be definitely known, we would at once scrap our present methods of mining and plans of operation.

Within the last few years there have appeared several machines designed for loading coal mechanically, some even cutting, mining, and loading. At first, these machines showed various defects in design and operation, which were removed and the machines were improved until they could be considered practical mechanically. The machines demonstrated that they could load coal with astonishing rapidity, but they can only operate successfully by a system that involves nearly every operative feature within the mine; and at most they can handle only 60 per cent. of the total coal. It was suggested that, to give the machine a fair trial, a system be evolved which would give it every possible advantage. But when we consider that it requires about 10 years to try out a method of mining in all of its phases, it requires more than a moment's thought before starting a new method. While the machine would replace a large amount of high-priced labor, such savings would be largely absorbed by interest and depreciation. What may have been a narrow gain in favor of the loading machine during normal times, under present relative values of labor and money would be a large gain in favor of the loading machine.

Several years ago, in times of chronic car shortage, a railroad incorporated in its method of mine ratings the number of working places a mine had. Irrespective of what other conditions existed, it was necessary to get cars to transport the coal, so miles and miles of headings were driven and rooms turned, tracks laid and all expenditures made to qualify for a high rating of working places. After 10 years the ruinous effects of this plan were still noticeable. The small pillars laid out could not be fully recovered, which represented an actual loss.

To say that a certain district will mine 85 per cent. of the calculated tonnage might be construed to infer that 15 per cent. was lost; I should be rather proud of an 85 per cent. recovery. Because we recover 85 per cent. of the coal does not mean that we have lost 15 per cent. and before we can understand each other it is necessary to know what is meant by lost coal.

Conservation of coal is not the principal thing for which we are working; we are working for dividends for the company. Shall we spend 50 cents for timber to recover 5 cents worth of coal? If part of a seam carries inferior coal and cannot be sold should this be considered as lost coal? If laws prevent mining coal within certain boundaries should this be charged as loss in mining? In certain localities gas and oil wells

penetrate the workable coal seams; it is considered necessary to leave a certain area of coal around such gas and oil wells for safety to both life and property. If we are going to talk about conservation and recovery of coal, all such features over which the mining man has no control should be eliminated. Unless this is done, comparisons between various fields would be misleading and broad generalities would be worthless. However, to get back to the original question, I believe that 85 per cent. recovery, after eliminating all features beyond the control of the mining man, would represent the best practice in bituminous mining today. I might add that this is in excess of what our company is doing, which represents about ten million tons annual output.

GEORGE S. RICE,* Washington, D. C.—I quite agree with Mr. Haas that conservation does not necessarily mean getting out 99 per cent. I question the 100 per cent. recovery. Let us suppose that 85 per cent. is the best that is practicable in many fields, still there is a great difference between that and the recoveries in some sections.

The effect of first mining upon the impurity of the coal as loaded was strikingly brought to my attention some years ago when the U. S. Bureau of Mines had occasion to examine and sample all of the mines then producing naval fuel in West Virginia. Analyses did not show as good coal in one mine in the Pocahontas field as in the other mines in the neighborhood; this we found was because of the large amount of pillar coal that had been standing for many years, so that the gradual weighting had so developed the columnar structure passing through the impurities as well as the coal, that in mining it all dropped down together and there was no practical method of separating the impurity.

Another point is the importance, for high recovery and other reasons, of not turning rooms in any panel until the pair of entries has been driven to the boundary of the panel, and then working back. Particularly commendable, it seems to me, is the Frick system where only narrow work is driven to the property boundary, then retreat is begun and the line is brought back by what is practically a retreating longwall system.

Professor Stoek says that sometimes the rooms are unnecessarily wide. I question that simply as an abstract proposition. It may be necessary to have the rooms wide in order to use machines or to conform, in the Middle West to certain agreements with the miners. It seems to me that the maximum recovery may be obtained, generally speaking, with narrower rooms. The greatest fault that I have observed in the Middle West has been that insufficient pillars have been left adjacent to the more permanent entries. There are conditions there where strain is caused in the roof and although what we term "squeeze" may not develop, these roof strains lead to falls of roof.

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* Chief Mining Engineer, U. S. Bureau of Mines.

In the experimental mine, near Pittsburgh, few rooms have been opened, and in about 12 or 13 years we have had practically no falls of roof in the main entry. Yet in the old entries of other mines in that gas-coal district the immediate roof has usually given way. That is largely because the room working has been carried so close to the main entry that the bending and shearing strains I speak of have resulted.

RUSH N. HOSLER,* Harrisburg, Pa.—Professor Stoek says that in the opinion of many operating officials and inspectors connected with liability-insurance companies, the accidents are no greater in pillar work than in advanced work. By statute, all compensation-insurance rates on coal mines in Pennsylvania are made by a rating bureau approved by the Insurance Commissioner. This bureau maintains competent coal, mine inspectors to examine and report the condition of the mines, as reflected by the standard upon which the mine is rated.

In the year 1920, in about 1500 inspections of bituminous mines in Pennsylvania, made by the rating bureau, we found that 20 per cent. of the men engaged on mining coal were employed in pillar mining. (Pillar mining includes heading and entry stumps, as well as room stumps and pillars. Pillar work includes drawing stumps, loose-end pillar mining, slabbing, cutting crosscuts or break-throughs in pillars, pillar splitting and every other species or description of coal recovery from pillars.) Men on coal includes miners, loaders, machine runners, and scrapers.

For the five year period, from Jan. 1, 1916, to Dec. 31, 1920, there were in the Pennsylvania bituminous mines 1243 fatalities resulting from falls of roof and coal. Of these, 163 were in the entries some distance back from the face and should not be counted as occurring in either advanced working places or pillar work; 424 were in pillar work, as defined above; 644 were in advanced working places. (An advanced working place is defined as follows: (a) Rooms, from the entry rib to the face of room; (b) headings, from the outside of the last crosscut turned to the face of entry; (c) crosscuts or break-throughs which are being driven in entries; (d) all crosscuts or break-throughs in or between rooms.)

It will, therefore, be seen that 20 per cent. of the men engaged on coal in the year 1920 in about 1500 bituminous mines in Pennsylvania were employed in pillar work, where between 34 and 35 per cent. of the fatalities for five years occurred.

H. N. EAVENSON, Pittsburgh, Pa.—The fourth conclusion is that to obtain a high recovery the coal should be gotten from the pillars and not from the rooms, so that the gradual widening of the pillars is the best

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* * Coal Mines Section of the Pennsylvania Compensation Rating and Inspection Bureaus.

method of recovering of coal. As a general proposition, that statement is correct, but in the Connellsville region within the last year there has been developed a system of mining for workings under about 250 ft. of cover. Rooms 11 to 12 ft. wide are driven on 32-ft. centers; but as soon as they are driven 250 ft. long a 6-ft. slab is taken off one side of the rib on the return and two slabs, 6 ft. each, on the other side. The rooms are driven at an angle of 45° from each butt so that the break line is kept practically in a straight line. It is really a longwall-retreating system. The face is now about 600 ft. long and about 900 tons a day are being mined by that section. From 80 to 125 per cent. more coal, per foot of face, is being obtained than was possible by the concentrated system with narrow rooms and wide ribs.

The reason this plan is successful, and it is not claimed to be adapted to all cases, is that the rooms are driven only 250 ft. long and the coal is taken out before the roof has a chance to settle. There are about twelve places on each heading, five or six of which are mining pillars and the rest driving up.

The system has the advantage that it is all machine work, and in pillar mining five cars can be loaded at once. There is no double shoveling and the work nowhere is more than 12 ft. from the rib; it is also possible to use as many men as is necessary to clean up a place in a hurry.

While the greatest thickness of the cover was about 250 ft., a section 400 to 450 ft. deep has been laid out and started.

GEORGE S. RICE, Washington, D. C.—Do they do any cribbing or filling in the gob?

H. N. EAVENSON.—No; they use their regular systematic timbering and draw as much of the timber as they can.

H. H. STOEK.—Is not the labor non-union so that there is no yardage claim? Do they not use the men in the same concentrated way as in the Mullin system, i.e., different gangs for drilling, loading, etc.?

H. N. EAVENSON.—Yes, just as they do in the other system, the concentration method.

W. H. GRADY, Duncott, Pa.—Do they carry it as a continuous panel?

H. N. EAVENSON.—They are planning on getting a continuous section of 1200 ft., bringing each 600 ft. from each set of butts down as panels.

About two weeks ago, I visited a mine in the Pocahontas field, where a loading machine is used in two sections of a mine, where they were mining pillars. In one section, the rooms had been driven 350 ft. and the first rib pulled back the full length, and they were just crossing the chain pillar into the rooms from the second heading below. In the other section, they had been cleaning up some old ribs that had been

standing for years. The ribs were 45 to 50 ft. thick and they had been split as in hand mining. I did not think that any machine would be able to work in this section but they have found less difficulty with the loader than when loading by hand; the timbering is less because the machine can load out as much coal in an hour as two or three men would load in two or three days at the least; consequently the machine will get out the coal before the roof has time to settle. In the case of hand loading, the place would probably fall in at least once and would have to be cleaned up and retimbered before the coal could be gotten out.

H. H. STOEK.—What kind of a machine was used?

H. N. EAVENSON.—One that was developed there, but I think any machine would do. This shows that sometimes the conditions govern these things entirely and that the rapidity of removing the coal is an important factor. They can get out eight or ten times as much coal with the machine as with the largest force of men they could put in the place.

GEORGE S. RICE, Washington, D. C.—Did the timber show much sign of stress before the machines had finished cleaning up? Was there sufficient warning to withdraw the machines in case of trouble?

H. N. EAVENSON.—This section had been worked by hand for some years and an area of, probably, 1500 or 1800 ft. by 1200 or 1500 ft. had been cleaned out. They were just cleaning up the last eight or ten rooms, which formed a big panel section. The places were timbered with posts spaced from 3 to 6 ft. with cross-bars and about one-half of the places were lagged between the cross-bars. The roof would generally hold until the machine was through loading. Sometimes, though, when the men would clean a place they would set temporary timbers which would be removed when the machine went in, then they just let the roof drop in.

S. A. TAYLOR, Pittsburgh, Pa.—I put in the first shortwall machine in the Pittsburgh district and had a lot of labor trouble as a result. In that mine we had only a little over 100 ft. cover, so I mapped out and tried a new scheme for the section. Our centers were 35 ft. but I widened the room to 27 ft., leaving an 8-ft. rib. Those familiar with the Pittsburgh district know that the track is on one side of the room and that the gob is piled up from 3 to 4 ft., depending on the thickness of the slate.

We drove those rooms up 250 ft. and then turned around and cut back on the rib with the machine which had a 7-ft. cutter bar. That left a little prism about 1 ft. wide at the bottom running up to practically nothing. We cut from 35 to 40 ft. of the rib on the return, which coal two men would load out in a day; the result was that we got our ribs out in about 10 days, with good recovery. We decided that with so light a cover the 8-ft. rib would be ample, which proved to be the case.

In the Clarksburg field, in one case where the cover was comparatively light, the work was laid out on the panel system with ten to twelve rooms on a panel, laying off an entry that would keep the work from that panel separate from the two main headings, and 30-ft. rooms were driven on 50-ft. centers. When we reached the limit of that room, we turned around and slabbed back 10 ft. on each side. The advantage of this method was that it was not necessary to wait until a room got up to the line before we could start to retreat. We could start the moment we got a room up and we had 10 ft. of rib in the other room to hold up the roof. Our recovery was remarkable but we sold the mine before we had time to determine the exact amount; we had worked out one panel of ten or twelve rooms, however. I have frequently found, especially in the Pittsburgh district, that the recovery given for the first work is usually greatly in excess of what is recovered later. In checking this up, I have found several times that the reason is that instead of the rooms being, say, 20 to 25 ft. wide, they are frequently 27 or 28 ft. wide; so that the area actually mined out is more than what is assumed in the calculation.

In many districts there is considerable impurity in the coal; so if the full thickness of the vein is considered, the recovery will be very low unless allowance is made for the amount of material that must be thrown out. That is especially true in Freeport, where the bone comes in the center of the seam. During the past few years, when trade was good and the miners were more independent, they loaded out a great deal of this refuse, so that the tonnage recovery from a given acreage was greatly in excess of what it is when business is dull and competition keen. So the tonnage recovery was much greater then than later.

The same thing applies in the Clarksburg region, where in many places there is a large amount of pyrites in the bottom. Figuring the full thickness of the seam and not eliminating any of the waste material, the estimated recovery on the veins is very much in excess of the actual recovery.

THOMAS H. CLAGETT.—In order that comparisons may be properly made it is necessary that the basis of determining recoveries be uniform; and as it is only intended that the clean coal shall be shipped the logical recovery is the ratio of tonnage shipped during any period with the theoretical tonnage of clean coal in the area mined during the same period. The value of the figures obtained depends, of course, on the accuracy of the information on which they are based. Inaccurate or incomplete mapping can frequently be detected in determining recoveries. Errors on account of inaccurate mapping of first workings are balanced when the pillars are mined. On that account the recoveries from beginning of operations to the end of any period are of more value, as showing the true situation, than the recoveries by years. When ship-

ments exceed the theoretical tonnage, it is apparent that one or more of the factors used in estimating the theoretical tonnage is in error or else shipments are not confined to clean coal.

S. A. TAYLOR.—In the first, second, and third pools of Pittsburgh the entire seam is usually figured from the top of the roof coal clear to the bottom, though in the early days all that was taken out was that between the draw slate and what is called the brick coal, leaving in the bottom about 12 to 15 in. of coal, while none of the roof coal was taken. As a result, the percentage of recovery was a very small proportion of the vein.

EDWARD W. PARKER, Philadelphia, Pa.—How much roof coal was left?

S. A. TAYLOR.—It varied from 2 to 5 ft. It was full of laminations and was not marketable, although tests made by the Bureau of Mines showed that it contained just as many thermal units as the other coal. It could be run through gas producers, of course producing a lot of ash, and a great deal of gas secured; but when the entire thickness of the seam, especially in those three pools, is considered, the recovery is very small. In addition, for a while the Government and some others were buying their coal on the thermal-unit method and their specifications were such that only a small portion, probably 3 ft., of that entire 6 ft. could be used. So that in a discussion of the recovery of coal per acre all of those things should be taken into consideration.

GEORGE S. RICE.—We should have a more definite understanding as to what is included in the calculation of recovery. In the operation of longwall mines in northern Illinois, some years ago, I was never able to make the actual coal weights check within about 4 per cent. of the quantity of coal in place, calculated from measured thickness. We were mining in a bed 3 ft. 2 in. to 3 ft. 6 in. thick. I was puzzled at first to find that we recovered only 96 per cent. of the theoretical quantity, but found that here and there points of the longwall face were buried in the gob; also, there were pyrite balls and lenses of other impurities that were discarded with considerable adhering coal. Therefore, as one would expect longwall would give the most complete extraction, I have always been rather skeptical of the high recoveries claimed, such as 90 or 95 per cent., from ordinary room-and-pillar work, especially from operations where in pillar withdrawal "curtain" walls are left, adjacent to the gob, and also where coal is left up for roof in advance workings, if this had been figured as part of the bed, as the complete recovery of this is almost impossible.

H. H. STOEK.—I had the opportunity, a few years ago, of examining a mine that had been operating not over 10 or 12 years and in which the

circle of the face was not very large. It had been surveyed each year and at the end of the year the face was platted. By measuring the areas with a planimeter it was found that only 93 per cent. of the coal had been gotten out and not 100 per cent. as is usually claimed for longwall work.

GEORGE S. RICE.—Did you have accurate measurements of the thickness?

H. H. STOEK.—I crawled all around the face to measure the thickness in many places and to obtain other data. We should emphasize the need of defining what is meant by "recovery." The only way to calculate recovery is to take the worked-out area, put the planimeter on it, get a large number of cross-sections and calculate the amount of coal that was originally in the ground; then knowing the output the per cent. extracted can be calculated. Any bone or refuse that is thick enough to be separated by hand and thrown out should be deducted. In the investigation mentioned, that was done but it made a difference of only 1 to 2 per cent.

R. V. NORRIS, Wilkes-Barre, Pa.—Another matter that should be considered in a discussion of recovery is the recovery from second mining; the recovery from first mining is usually nearly complete, the loss being in second mining. There is a great tendency among opreating men, particularly the younger engineers, to work out the pillars as a solid and then apply the original percentage of recovery; this results in a preposterous expectation of recovery. For instance, suppose the ultimate recovery is 85 per cent. and 60 per cent. is recovered in the first mining, leaving 40 per cent. for second mining. Of that 40 per cent. only 25 per cent. of the original coal, or 62½ per cent. of the pillar, can be removed to get the 85 per cent. result expected; while his estimate using 85 per cent. of the 40 per cent. would give 94 per cent. ultimate removal.

That error is continually made in estimates and reports and attention should be called to it, because it is a serious misconception of mathematics caused by the fact that most people fail to realize that the loss is in second mining. If conditions are such that you can get an ultimate removal of 90 per cent., you cannot get an ultimate removal of 96 per cent. by removing it in two slices of 50 and 46 per cent., but that is precisely what is often estimated.

W. H. GRADY, Duncott, Pa.—In the case of those Illinois mines that recovered 94 per cent. by the longwall method, is the percentage of recovery figured on shipments or production?

H. H. STOEK.—The recovery is figured not on the amount of coal sold but on the amount for which the miner was paid; that means the total amount of coal coming up the shaft. An estimate was made of

the amount of coal in place and that amount then divided into the amount that had been hoisted up the shaft.

H. N. Eavenson.—That is the coal produced, not the amount shipped.

R. V. Norris.—Your removal is what you can take out, your recovery is what you ship.

H. N. Eavenson.—We all agree that the figures on recovery should not include coal that cannot be used; that is, slate and such material that cannot be used at all. Any coal left in the mine to protect the top should be charged to mining. In the case of a Pocahontas mine having the usual streak of bone, which is about 2 in. thick, this thickness is not included in Mr. Clagett's figures because it cannot be shipped.

The worst feature of that streak of bone is that the coal is frozen to each side of the bone so that when you pick out the bone, you pick out about 4 in. of coal and 2 in. of bone. Much of that is picked out inside, some is picked out in the tipple, and some is shipped. I do not think nearly as much is shipped as is taken out. In some mines working the No. 4 and the No. 3 seam, exactly on the same system, we would always exceed the estimated recovery in the No. 4 and would be under it in the No. 3. In the No. 4, the top contained some good coal but we could not ship it because it ran 25 to 30 per cent. bone. We insisted that all of it be thrown away, but the miners, when nobody was watching, would pick it out and load it. This made the percentage of recovery higher than in the No. 3, where there was no refuse except this one streak of dirt. Coal left around gas wells, under roads, etc., should not be charged against the operators; it should be deducted from the total area.

As to accidents, in the past 11 years, our experience has been entirely different from the figures reported to us this morning. We never found any greater liability in pillar work (certainly not under our conditions) than in the first mining. In fact, if anything, the conditions were reversed because most of our accidents from falling top were due to kettle bottoms, which usually fell without any warning, often when there was timber under them. These had practically all been discovered before the pillars were mined, or if they were in the ribs, the coal was usually crushed enough so that they were down before the coal was removed.

H. H. Stoek.—I understand that is the experience of the mines in western Pennsylvania and northern West Virginia.

H. N. Eavenson.—More of their accidents occur in the first mining than in the pillar mining; I can easily confirm that because they keep careful records of their accidents.

· George S. Rice.—Do you not think that the reason for fewer accidents in pillar mining is that the men are alert, picked men, and that

they are constantly expecting a fall, whereas that is not true in advance mining?

H. N. EAVENSON.—I think that is true and for that reason do not think that there are as many accidents as in the first mining.

R. D. HALL, New York.—The draw slate over the pillars in most cases is more or less broken by the time the pillar is ready to be withdrawn. It has in any event three loose ends, one at the end of the pillar and one on each side. Consequently the miner is sure that it is ready to fall and must be propped up or wedged down, if it does not come down with the coal. When a room is first driven, however, the draw slate often appears to be solid and for a while may adhere to the roof; it is, however, liable to come down at any time if not watched. For these reasons there are risks in room driving that are not so apparent in pillar extraction.

Relative to the stray measures above the coal and those of inferior quality, such as the "rooster" coal above the Pittsburgh bed, I have heard that frequently when they come down with the draw slate they are loaded out by the miners, sometimes contrary to instructions. As the possible recovery is based on the main seam and not on the riders, the percentage of recoveries is sometimes figured at 103 or 104 per cent.

GEORGE S. RICE.—Has anybody analyzed the Pennsylvania accident figures given here?

H. N. EAVENSON.—I have not, they were new to me.

H. H. STOEK.—I have not had time to analyze them. They were sent to me, but I have not had time to go into them. The output per man in the pillar work would undoubtedly be much greater than in advance mining; and while I believe that accident statistics should be taken on the basis of the employees, they should also be considered in connection with tonnage. The two should be studied together because it is not fair to take either item alone.

H. N. EAVENSON.—By whom were they published?

H. H. STOEK.—The Coal Mines Section of the Pennsylvania Compensation Rating and Inspection Bureau.

GEORGE S. RICE.—This matter has a most important financial bearing, because the next stage will tend to be to penalize mines for drawing pillars, so that these figures should be investigated to determine if their factors are correct. My experience has been that pillar withdrawing was not so likely to cause accidents as work at the face, at least where experienced miners carry on the work, and usually only experienced men are assigned to such task. The condition of the roof, after the first

break from systematic pillar extraction, plainly shows its own condition of danger; and if proper propping is done, due warning is given by sounds and by dribbling of loose pieces; whereas, in solid working often a "pot" or intersecting "slips" are concealed and fall with no warning.

CHARLES ENZIAN, Philadelphia, Pa.—Perhaps that statement includes the accidents from certain districts of anthracite pillar mining, especially those of the heavy pitch workings in the middle and Schuylkill or southern field, and especially in the areas where the seams are inverted—that is, where the bottom rock becomes the roof. Under such conditions we have the paradox that more injuries are caused by falls of bottom rock than by falls of top rock, because the true top rock has become the bottom, or floor of the seam.

S. A. TAYLOR.—I think that there is probably some mistake about the figures given. A much better class of men, usually, are employed on the pillar work. For a long time in the Pittsburgh district our agreement with the miners prohibited the use of machinery, so that we had a much better class of men working on pillar drawing than in the room workings.

H. N. EAVENSON (written discussion).—The results of the investigation of the facts stated by Mr. Hosler that the writer was directed to make have been rather unsatisfactory, partly because the information that was necessary, in order to get the exact data, was not in the possession of many companies in available form, and the time at which this information was requested, because of the impending strike, made it impossible for most the companies to give the request any attention. Of the numerous requests sent out, only four companies that replied had actual data and for three of these the figures bore out exactly the statements made by Mr. Hosler; for the other, the difference was too small to change his conclusions. A representative of one of the largest companies in the state, while not having the figures at hand, said that its experience bore out Mr. Hosler's statement, although this company was not represented in his figures. After a long talk and a thorough discussion of the method by which Mr. Hosler collected his data, there is no doubt in the writer's opinion that the conclusions he expressed, as regrettable as they may be, are correct so far at least as Pennsylvania is concerned.'

Can Anthracite Mines be Operated Profitably on More than One Shift?

BY DEVER C. ASHMEAD,* KINGSTON, PA.

FROM time to time metal-mine engineers have inquired why anthracite mines and their preparators are rarely operated on the two or three-shift basis. The subject may be approached as affecting: labor, development, the underground workers, the preparator, the outside workers, capital, the supply account, power consumption, the storage of coal over various shifts, railroad car supply, and the market.

LABOR

From the start, the adoption of a plan of three full-shift operations in a day would cause difficulty with the colliery employees, for they have never worked on that plan and would show their resentment by suspending work. Even if they could be induced to work according to this schedule they would not be as efficient, for they would be working more or less unwillingly at hours at which they are accustomed to rest.

Operation on a three-shift basis would necessitate triple manning of each working place, and dissensions would arise between the men, for each shift would claim that the preceding shift had taken undue advantage and had not left as favorable conditions as it had found. To obviate these disputes it might be necessary to let out the work to a contractor, who would hire his helpers to mine out the chamber, or breast. The state mining law requires that a miner be put in charge of each working place. To conform with this provision he would have to hire three helpers and two miners to mine the breast. The labor unions have been trying to abolish the subcontractor in mine work and any method of working that would cause his return, however innocent in purpose, would result in labor disturbances.

It might be possible to form partnerships of three miners working a place together, but these men would be quite likely to have frequent dissensions as to the proportion of work performed by each shift. Only

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' Mining Engineer, Anthracite Editor, *Coal Age*.

one other method of working remains—operation by company men, but this is ordinarily too expensive to be considered.

In addition, the state mining law requires that before he can become a miner a man shall have two years' experience and then pass an examination. This limits the number of miners available, and if all of the anthracite collieries were to go on a three-shift basis there would not be enough men to fill the working places.

DEVELOPMENT

The development work in those parts of the mine in which there is first mining is often on the three-shift basis. It is only with difficulty that this development work is kept sufficiently far in advance of the coal-mining forces to keep them supplied with working faces.

SECOND MINING

It is difficult to maintain the output from those mines or portions of mines where production is from second mining; first, because it is frequently necessary to reopen closed places, to re-lay tracks, clean falls, and in places to penetrate squeezed areas; second, because it is necessary to allow time at frequent intervals for the settlement of disturbed measures so that the danger to mine workers at the face may be minimized; third, because special conditions, such as overlying workings or the necessity for partial or complete support of the surface and surface buildings, often limit the location and extent of second-mining areas.

UNDERGROUND FORCES

If it were possible to divide the miners into three shifts and work three parts of the mine integrally, one in each shift, the objection to contract work would be done away with, for then every miner would still have his own place. But it would not be possible to get the present efficiency, as two-thirds of the men would be working on off shifts and their willingness to work would be decreased.

But if it were possible to divide the miners among the three shifts, there would not be enough company men, for the miners and their helpers constitute only 30 to 40 per cent. of the men employed. The problem would be to make this force of men handle the duties of all three shifts. In order to deal intelligently with this subject, a careful analysis of the types of underground workers at collieries should be made.

Development

It may be just as well to leave out the men at work at development for their number will hardly be increased, as even at present the advance work is often conducted during three shifts of each day. So, omitting development men, who are included among the miners and their helpers,

the number of underground workmen at four collieries is as shown in Table 1.

TABLE 1.—*Underground Force at Four Anthracite Collieries*

	Mine No. 1			Mine No. 2	Mine No. 3				Mine No. 4
	Opening No. 1	Opening No. 2	Total	Total	Opening No. 1	Opening No. 2	Opening No. 3	Total	Total
Cutting and loading									
Contract miners......	106	133	239	147	54	89	60	203	321
Contract laborers.....	5	3	8	27	15	12	8	35	194
Company miners. ...	11	22	33		3			3	9
Company laborers....	12	10	22		3			3	33
Total............	134	168	302	174	75	101	68	244	557
Ventilation									
Door tenders........	2	5	7	11	7	4	5	16	
Brattiecemen.........	1	5	6	5	3	2	2	7	9
Masons.............				3	2		2	4	7
Safety-lamp men.....		1	1						2
Total............	3	11	14	19	12	6	9	27	18
Transportation									
Stationary engineers..					4	4	2	10	3
Shaft, slope and plane attendants.......	3	2	5	14	11	9	5	25	21
Locomotive engineers.	6		6		2			2	
Drivers.............	8	15	23	27	18	18	15	51	35
Runners and spraggers		3	3	30	20	13	8	41	29
Electric motormen....									14
Motormen's helpers...									16
Road cleaners.......	2	2	4	5					10
Trackmen.	3	7	10	14	6	6		12	13
Stablemen...........	1	1	2	1	1		4	5	
Shoers.............				1	1		1	2	
Pulleymen..........				2					2
Total.............	23	30	53	94	63	50	35	148	143
Pumpmen.............	2		2	4	5	2	1	8	7
Miscellaneous									
Rockmen............							3	3	12
Rock stowers........					4	4	5	13	
Timbermen..........		3	3	9	7	7	1	15	
Machinists..........				2	2		1	3	5
Electricians.........									1
Siltmen.............				9	6			6	21
Others..............									6
Total.............	0	3	3	20	19	11	10	40	45
General									
Foreman............	1	1	2	1	1	1	2	4	8
Assistant foremen....	6	10	16	10	6	3	5	14	10
Driver bosses........									2
Timekeepers........									2
Total............	7	11	18	11	7	4	7	18	22
Total..........	169	223	392	322	181	174	130	485	792

Pumping

With the introduction of the triple shift, no increase would have to be made in the number of pumpmen. This, however, is a small item for in Table 1 mine No. 1 has only two men, mine No. 2 only four men, mine No. 3 only eight men, and mine No. 4 only seven.

Supervision

Ordinarily, with three shifts a somewhat greater number of foremen would have to be engaged. Where, of course, there is but one opening the number of foremen must be increased for there must be one man in charge of each shift; but where there is more than one opening there is usually a foreman for each. Consequently, if each opening or some combination of openings were put on different shifts the number of foremen would not be increased. This is clear from the table.

Ventilation

Men employed in maintaining the ventilation of the mines are more numerous; thus in Table 1, mine No. 1 has fourteen; mine No. 2, nineteen; mine No. 3, twenty-seven; and mine No. 4, eighteen. If the mine were placed on three shifts, the number of men would have to be increased somewhat, for men would have to work on the ventilation in every shift, so that promptness in making repairs would be assured. More door boys would be required, for even if the mine worked in sections there would be parts of the haulage systems included in more than one section, and boys would have to be supplied at the doors in these roadways during two or even possibly three shifts. It is not unlikely that to obtain proper ventilation the number of these employees would be increased 50 per cent. by the tripling of shifts.

Transportation

It is, however, in the solution of the transportation problems incident to the triple shift that the biggest difficulty would be encountered. The simplest condition would be where there are several independent openings. These might be so grouped that it would be possible to use the same number of men with three shifts as are now employed on one. This, however, would be an unusual condition. In most cases there would be a mine with a long roadway along which all, or a large part, of the coal, no matter in what section it is produced, must be hauled. Some of the workings in the mine, consequently, must be allotted to one shift and some to another.

It might be necessary to put the workings in one or more beds on one shift, and the workings in other beds in a second shift, and to operate the rest of the beds in the third. Here the transportation force would be greatly increased. In both cases it might be necessary to increase the

number of transportation employees considerably. Only the number of the men taking coal from the batteries or the face and hauling it to the partings that the sections served and the door tenders in those sections would remain unchanged. The three-shift operation might well involve an increase of 150 per cent. in the transportation force without any compensation derivable from larger tonnage. The trouble today is that sections of the transportation system cannot be worked to capacity; therefore any dilution of the traffic by the introduction of the three-shift system would merely intensify a condition that even now is sufficiently crippling.

Silting

In those mines where silting is practiced, the three-shift plan would make it necessary to silt on all three shifts, provided that the preparation equipment as well as the mine was run with equal continuity. This would mean increasing the force almost 200 per cent. and yet obtain only the same output and do the same amount of silting. The building of batteries to retain the silt and the laying of silting lines would not involve any more work for three shifts than for one; therefore the cost would not be increased proportionately to the number of shifts.

Working Force

In order to show the effect of the output on the number of men employed, nineteen curves have been compiled from data from ninety collieries having an annual output of 44,205,896 tons. The data from each colliery are plotted separately and an average made for each 50,000 tons increase; these points are indicated by a circle. The curves give due weight to these subaverages, considering the number of observations involved in each. The high or low points apparently neglected are usually from meager, often single, observations or from known abnormal conditions. The curves are intended to represent general conditions, not individual cases.

In Fig. 1 the curve shows the number of men employed per 10,000 tons output for total and inside employees; in Fig. 2 the curves show the number of men employed per 10,000 tons output for outside and preparation employees. These curves show that a decreased force of men is required per unit as the output increases; also that the greatest decrease in men is in the preparation forces.

Figs. 3 and 4, giving the output per man employed, show how the output per man increases as the collieries increase in size. Figs. 5 and 6 show the number of men employed per start of different capacity collieries and Figs. 7 and 8 show the number of men employed by collieries of different yearly outputs. Fig. 9 shows average curves of the total number of employees on one shift constructed from the preceding,

compared with curves showing the estimated number of men required on two and three shifts for the same total output.

These charts furnish the data necessary for a reasonably accurate analysis of the subject, dividing the output in three and then producing the full output in three shifts. Taking first the operation of a mine having a capacity of 900,000 tons per year, or approximately 3200 tons per shift, assuming 280 starts per annum, which is the average number of starts of the 90 collieries considered, if this mine is to be operated at
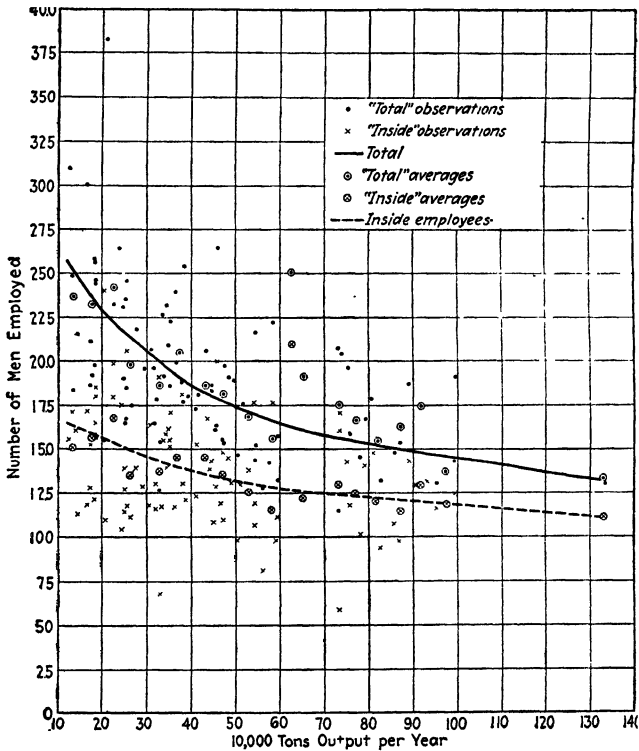
FIG. 1.—NUMBER OF INSIDE EMPLOYEES AND TOTAL NUMBER OF EMPLOYEES, IN 1920, PER 10,000 TONS OUTPUT.

the same tonnage but with an output of only 1070 tons in each shift, the curves in Fig. 1 show that the one-shift operation requires 12.1 inside employees per 10,000 tons annual output, but that the three-shift operation will require 14.7 inside employees. Multiplying by 90, it will be found that the mine that would have produced 900,000 tons with 1089 inside men will require, when operated on three shifts, 1323 men to attain the same production. Thus the output, after the division of the work, will require the employment of 234 more inside men than before, pro-

vided, of course, the efficiency of the men is unaffected by the change, which is not likely to be the case.

The curve in Fig. 3 shows the output per inside man for mines of
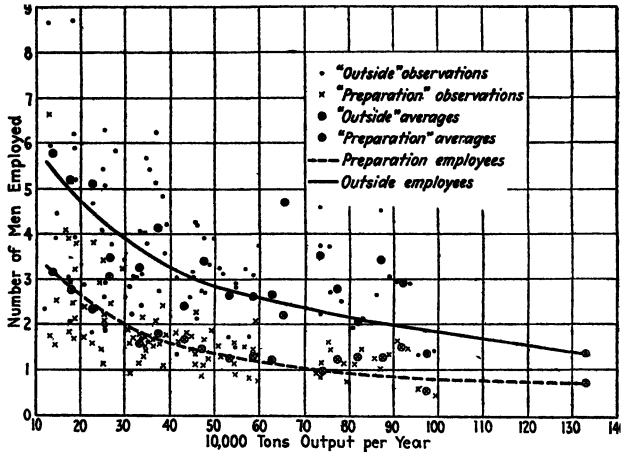
FIG. 2.—NUMBER OF OUTSIDE AND PREPARATION EMPLOYEES, IN 1920, PER 10,000 TONS OUTPUT.

different sizes.   A mine producing 900,000 tons per annum would have an output per man of 815 tons per year.   In a mine one-third the size, the output would be 710 tons.   The difference, 105 tons per man, is the loss in
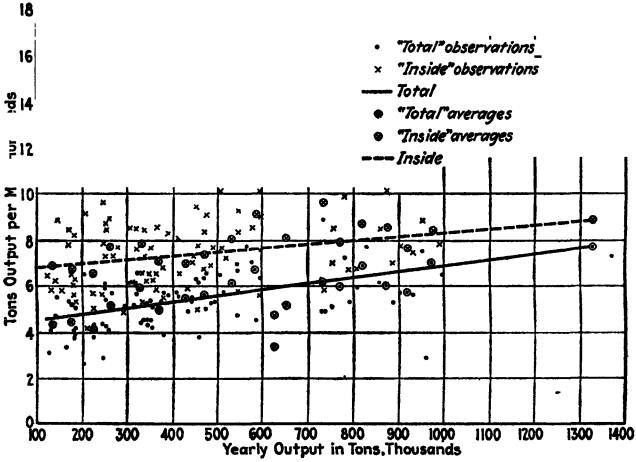
FIG. 3.—OUTPUT, IN TONS, PER INSIDE AND TOTAL NUMBER OF EMPLOYEES, IN 1920.

output sustained where the larger mine is operated on a three-shift basis. Thus, at the larger mine, working one shift per day, there would be 1105 inside men; and at the smaller mine, working three shifts a day, 1268 inside men.

From Fig. 5, it is seen that the larger mine would produce its tonnage at the rate of 3200 tons per start with 1120 inside men, whereas the smaller mine, running at the rate of 1070 tons per start, would have 410 men.
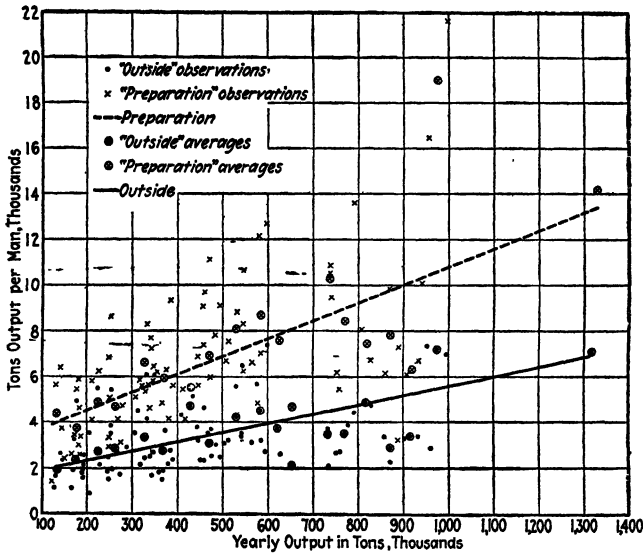
Fig. 4.—OUTPUT, IN TONS, PER OUTSIDE AND PREPARATION EMPLOYEE IN 1920.
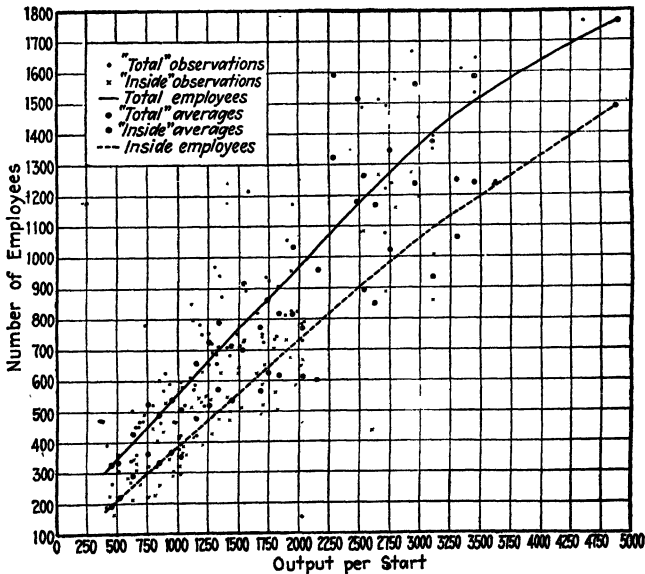
Fig. 5.—NUMBER OF INSIDE EMPLOYEES AND TOTAL NUMBER OF EMPLOYEES, PER START, IN 1920.

Runnin the smaller mine three times as many shifts to get three times the tonnage would require not 1120 but 1230 inside men.
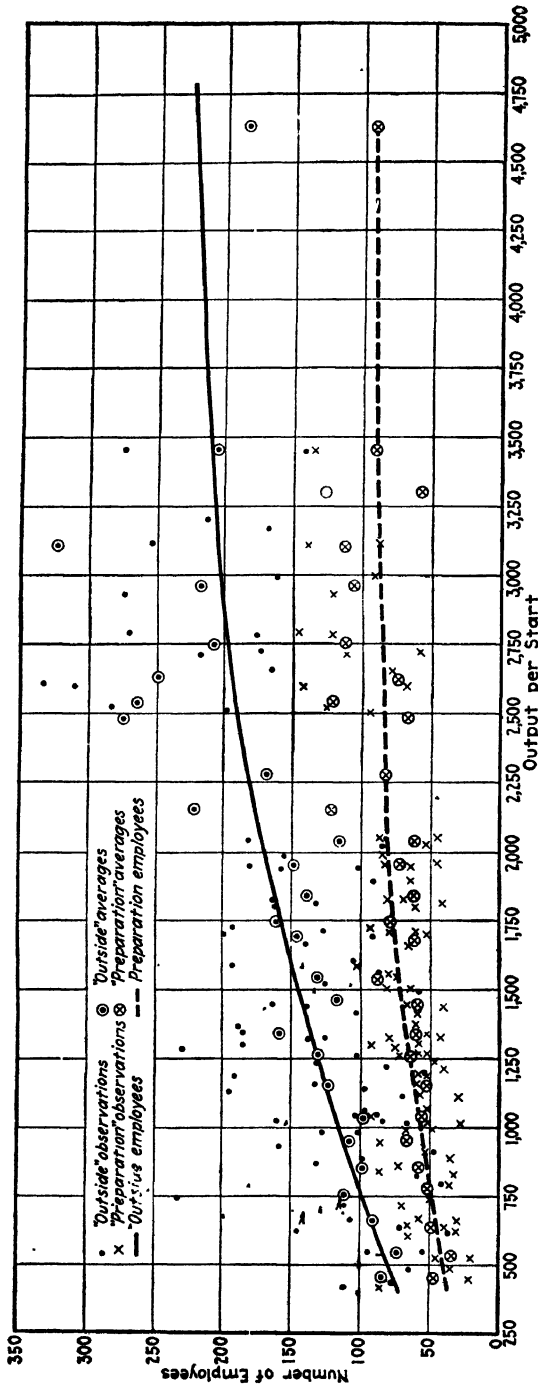
FIG. 6.—NUMBER OF OUTSIDE AND PREPARATION MEN EMPLOYED, IN 1920, PER START.

Fig. 7 shows the number of men employed in direct relation to the number of tons produced, not taking into consideration the number of starts. Here the mine with 900,000 tons of annual output would employ 1118 inside men whereas the mine with 300,000 tons of annual output per shift would employ 416 men, or 1248 inside men to produce 900,000 tons per annum from three-shift operation.

These figures show a slight discrepancy which is caused by the different methods of compilation. In some cases, the time feature is
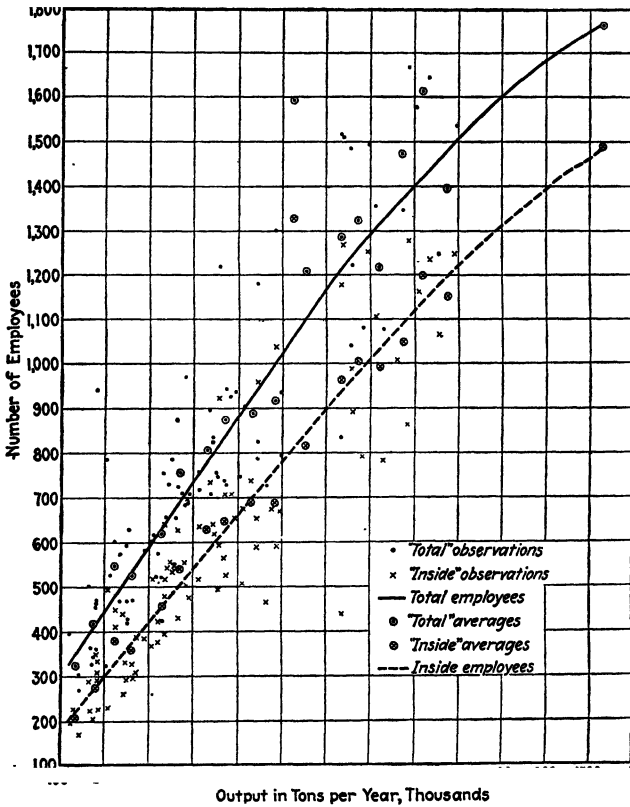
FIG. 7.—NUMBER OF INSIDE EMPLOYEES AND TOTAL NUMBER OF EMPLOYEES, IN 1920, AT ANTHRACITE COLLIERIES OF VARIOUS OUTPUTS.

considered and in others it is not, so the probable result will be an average of these figures, which is 1108 for the one-shift operation and 1267 for the three-shift mine, an addition of 159 men to the force.

THE PREPARATOR

Coal preparators are not designed to work on the three-shift system. The anthracite preparator involves a complete and complicated flow sheet

Without undue and unnecessary duplication of machinery, as many of the machines are of large capacity, few single machines can be stopped for repairs without disarranging, and usually stopping, the entire process. In this, it differs from the mills of metal practice, which contain a large
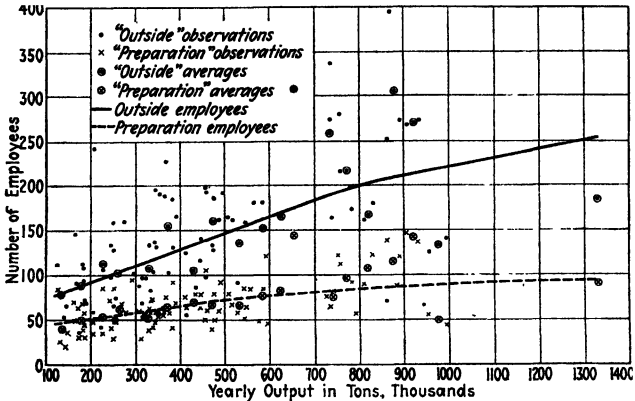
FIG. 8.—NUMBER OF OUTSIDE AND PREPARATION EMPLOYEES, IN 1920, AT ANTHRACITE COLLIERIES OF VARIOUS OUTPUTS.

number of similar machines, that can be stopped and repaired without materially reducing the output.   Under the present system of preparator operation, it is possible to examine all the equipment every night and make the necessary repairs in time for the next morning's start.
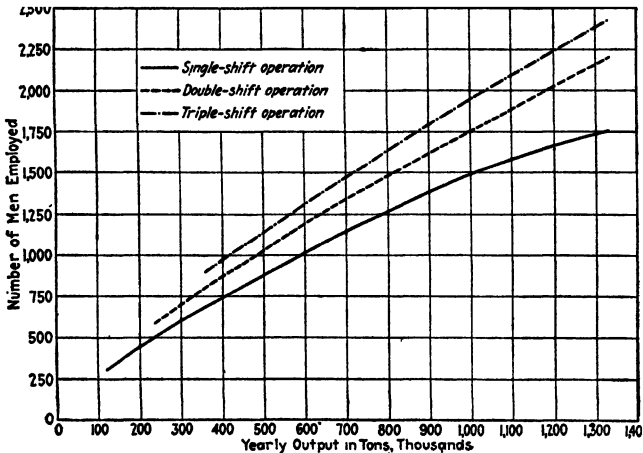
FIG. 9.—AVERAGE TOTAL NUMBER OF MEN EMPLOYED COMPARED WITH NUMBER ESTI-MATED FOR TWO AND THREE SHIFTS.

If the preparator is already built, the proposition that it be run on three shifts instead of one, without correspondingly increasing the output, is, of course, unreasonable.   In one mining operation, the output has fallen

from over 1,000,000 tons to about 350,000 tons a year. When the preparator was cleaning and sizing the larger output 93 men were required; for the smaller output 56 men are required. Thus, though the output has decreased 65 per cent., the number of men needed has dropped only 40 per cent.

It is apparent, therefore, that the number of men employed in preparator operation does not decrease in direct proportion to the output of the preparator, and it is probable that the ordinary rate of decrease is much smaller than in the example quoted, for improvements have been made in this preparator which have decreased the number of men required to operate it irrespective of the output. For this reason, were the output of the mine put back to 1,000,000 tons, the preparator would size and clean it for market with less than 93 men. Nevertheless, using this case as an example, 168 men would be required if the preparator were run on a three-shift system and only 93 if run only one shift; or an addition of 75 men.

Assume that the preparator is to be built of a size suitable for three-shift operation, and that it will be as economical as a plant to prepare and size an output one-third as great on the one-shift basis. As Fig. 2 shows that two men are required in the preparator for each 10,000 tons per year of preparator output where 300,000 tons are sized and cleaned per annum 60 men would be employed on each 8-hr. shift; therefore, if the plant were run three shifts in every 24 hr., it would be necessary to employ 180 men, to obtain an output of 900,000 tons per annum. But, Fig. 2 also shows that this tonnage could be cleaned and sized by a force averaging 0.88 man per 10,000 tons per annum in a preparator having sufficient capacity to clean and size the coal in 8 hr.; that is, 79 men would be required for an output of 900,000 tons per annum. Such a preparator, therefore, would require 79 men. Thus a big preparator working 8 hr. will do the work with 101 fewer men than the smaller preparator working three shifts. From Fig. 4, it is possible to deduce that 170 men would be needed for three-shift operation as against 90 for a plant that could prepare the coal in an 8-hr. day. The curves in Fig. 6 show that whereas 174 men would be required for three-shift operation 88 men would suffice with the larger plant. The coördinates of the curves in Fig. 8 show that 174 men would be required for the small plant working 24 hr. and 87 men for the large plant operating 8 hr. Averaging all the results, the conclusion is reached that whereas 86 men would be sufficient to work a single-shift preparator producing 900,000 tons per annum, 175 men would be needed to effect the same result in one working continuously. Thus the larger preparator reduces the number of men employed by eighty-nine.

Having seen from the curves the effect of three-shift output of the preparator, let these be compared with the actual force accounts of four anthracite preparators shown in Table 2.

Where only one foreman is employed on single-shift operation, there

TABLE 2.—*Outside Force at Four Anthracite Collieries*

	Mine No. 1	Mine No. 2	Mine No. 3	Mine No 4
Transportation				
Stationary-hoist engineers	9	9	10	13
Locomotive engineers and helpers	6	4	4	4
Headmen, footmen, and plane tenders	5	6	9	11
Drivers, runners, and spraggers	5	1	4	5
Car oilers	1		1	2
Track repairmen	2	3	1	6
Total	28	23	29	41
Preparation				
Breaker bosses	1	1	1	3
Ticket takers	1	1	1	
Dumpers	1	1	1	2
Platemen and tablemen	9	1	11	6
Pickers and slatemen	5	8	6	28
Jig and spiral attendants	9	7	6	3
Machinery attendants	2			6
Oilers	1	1	1	1
Ropemen				
Breaker cleaners	1	1	2	
Engineers	1		1	1
Breaker pumpman	1			
Miscellaneous	1	2	2	7
Box-car loaders			1	
Coal-car loaders	1	4	3	7
Car runners	5	1	3	
Car cleaners and patchers		1	1	
Total	39	29	40	64
Distribution of refuse				
Locomotive engineers and helpers				2
Machine attendants	2	1	1	
Laborers	6	3		3
Pulveriser and bore-hole attendants				1
Total	8	4	1	6
Repairs and maintenance				
Blacksmiths	5	2	4	3
Carpenters	15	4	4	9
Mine-car repairmen		6	4	
Machinists	2	2	1	2
Masons	1		1	
Electricians		1		
Machine attendants		1	1	
House-repairmen		1	1	
Laborers				6
Total	23	17	16	20
General				
Watchmen	2	2	2	3
Court house (coal inspection)		1	1	2
Stablemen	1	1	1	3
Teamsters	1	1	2	2
Fan engineers		1	1	4
Fresh-water supply	2			
Laborers	10	4	10	
Prop men	6	2	3	8
Crane engineer				1
Total	22	12	20	23
Steam				
Firemen	6	6	13	6
Fuelmen	1	1	1	2
Ashmen	2	1		5
Boiler repairman	1			
Water tender			1	
Total	10	9	14	13
Supervision				
Foremen and assistants	1	1	1	2
Clerks	3	3	3	4
Messengers	1		1	
Total	5	4	5	6
Total	135	98	125	173

is no way in which the tripling of this expense can be avoided, when three-shift operation is introduced, except by an increase in tonnage. On every shift there must be one foreman. The same is true of the ticket taker. The number of dumpers must be tripled. The number of plate or table men and pickers, or slate men, could probably be decreased per shift, though not in proportion to the number of shifts; it would take at least twice as many for three shifts as for one.

The number of spiral, jig, and machinery attendants on any shift probably could not be reduced, even though the output per shift were reduced two-thirds. The number of all other employees would probably have to be tripled, with the exception of the car loaders and runners, the number of whom would only increase about 100 per cent. Allowing for an equal number of men on each shift for each operation and making due allowance, in some cases, for abnormal conditions, it will be found that for the same output in 24 hr. by using the three-shift plan, mine No. 1 would need 141 per cent. more men for preparation; mine No. 2 would use 147 per cent. more men; mine No. 3, 120 per cent. more; and mine No. 4, 119 per cent. more. These figures may be compared with the increase from a minimum of 80 per cent. to a maximum of 140 per cent. in the number of men employed in the preparator, as shown by calculations made from the curves.

OUTSIDE FORCE ACCOUNT

Considering the total outside forces, except the preparation employees, Fig. 2 shows that a colliery having an output of 900,000 tons on one shift, or 300,000 tons on three shifts, will require 351 men for the three-shift operation and 180 men for the single-shift operation. Fig. 4 shows that the output per outside man in the three-shift operation is 2720 tons, whereas for the single-shift operation it is 5200 tons. Thus 331 men will be required for three-shift operation and 173 for single-shift. From Fig. 6, assuming 280 starts per year, it is seen that 360 men are needed for the three-shift operation and only 206 for the single-shift. Lastly, from Fig. 8, 330 men are needed for the three-shift and 212 for the one-shift operation. Averaging these figures, it is found that 343 men are required for a three-shift operation and only 193 men for a single-shift operation with the same output, or a saving of 150 men operating for the single shift.

SUMMARY OF FORCE ACCOUNTS

The previous results, which have been obtained from the curves, show that with three-shift operation 1794 men will be required, whereas with the single-shift operation, which now obtains, only 1387 men are needed; thus the present method saves the employment of 407 men. Fig. 9 is a summary curve showing the average number of men employed and also the number that would be required for two- and three-shift operation

of the same sized colliery but with the output divided into two or three as the case may be, depending on the number of shifts. In the example taken, the force of men required would have to be increased 29.4 per cent. if the mine were to be placed upon the three-shift basis without increasing the output.

These figures are, of course, only approximations but it is believed that they err on the side of conservatism. While it is true that some classes of employees, as pumpmen, would not be affected by the number of shifts worked, these variations are amply compensated for in the conservative allowances for other classes of employees.

CAPITAL

Unfortunately, no figures are available that will show the effect on the cost of development by operating the mine on the three-shift instead of a single-shift basis. It is possible, however, to show in the case of a few preparators the effect of this on the cost of equipment. Table 3 gives data for five collieries, from which the effect of three-shift operation on the equipment cost can be approximated.

TABLE 3.—*Cost of Construction and Other Preparator Operating Data as of* 1920

	Mine No. 1	Mine No. 2	Mine No. 3	Mine No. 4	Mine No. 5
Maximum capacity........	1,800	1,500	1,800	4,800	6,000
Cost of construction.......	$358,835	$307,865	$360,346	$495,976	$1,078,000
Cost per ton of capacity....	$199	$205	$202	$103	$180
Total number of employees.	39	28	39	59	56
Tons prepared per day per employee...............	46	54	46	81	107
Tons coal prepared in 1920.	299,883	222,792	326,569	515,169	997,563
Average tons prepared per day...................	1,082	883	1,320	1,812	3,420
Per cent. of maximum capacity.	60	59	73	38	57

.If the preparator was operated upon three full shifts or, in other words, if the output was tripled, full use of the equipment investment would be had, but the investment in the preparator would have to be increased to provide machinery to take care of breakdowns and permit continuous operation.

If the total output is to remain the same and the present preparator is to run on three shifts, no advantage will be obtained as the investment remains the same, provided no new preparator is constructed. Although operated continuously, only one-third of the capacity of the equipment is being used each shift, so there is practically no change in the use of the cost of the equipment in three-shift operation over one-shift operation.

Another assumption is the use of a smaller preparator. Mine No. 5, in Table 3, has a daily capacity of 6,000 tons and its preparator cost $1,078,000. This is a one-shift operation. Suppose that the preparator is destroyed and that it is proposed to build a plant to prepare coal on three shifts, so as to reduce the capital invested, as the new preparator will only have one-third of the capacity. Mines Nos. 1 and 3, which have approximately one-third the capacity of mine No. 5, are modern steel buildings of the same general type as that at mine No. 5. The average cost per ton capacity of No. 1 is $199, whereas the preparator for mine No. 5 costs only $180 for the same unit output, therefore the actual investment for a preparator of small capacity is greater per ton than for a larger one.

Mine No. 2 has approximately one-third the capacity of mine No. 4, but the cost of the preparator per ton capacity is $205 as against $103 for the larger operation. This latter relation is more typical of the average condition in the anthracite field than is the preceding, for the preparator for mine No. 5 cost more than is usual because of certain conditions peculiar to its construction. Therefore the cost of a preparator one-third the size of another will be approximately 100 per cent. more per ton of output.

From this it is seen that in no case is the saving in cost directly proportional to the change in the tonnage of the preparator. In the case of the erected preparator, the cost per ton of product will not be one-third that with one-shift operation although the output is increased three times; it will be one-third plus such an amount as will provide machinery to permit continuous operation. In the second case, the amount of cost of equipment will remain roughly the same no matter whether the preparator is on one shift or on three shifts as the output remains constant. In the third case, the cost of the preparator per ton of output increases approximately 100 per cent. if the capacity per shift is decreased to one-third. Therefore, the three following conclusions are formed on the investment per ton of output: First, that the triple shifting of an already erected preparator will decrease the amount of the cost of investment per ton of daily output; second, such a triple shifting of a preparator, if not accomplished by enlarging the capacity, will not make any change in the amount of the investment per ton of output; third, the construction of a preparator of one-third the capacity will not proportionately reduce the cost per ton of capacity.

SUPPLY ACCOUNT

With triple shifting, a larger supply account would have to be carried in the regular mining operations as more equipment would have to be held in reserve in case of breakdowns, for as the equipment would be in

continual service it would not be possible to repair it while it was running and hence the maintenance could not be as good.

POWER CONSUMPTION

A saving would be effected by triple shifting, because the demand for power would be steady and hence stand-by losses would be avoided. All the power, no matter when produced, would directly promote the production of coal.

STORAGE OF COAL

If the mine is planned to operate on three shifts and the preparator on one shift, the storage of coal mined while the preparator is idle becomes important. There are two places for the storage of this coal—on the surface and underground, and there are two methods that can be used in each place.

Either storage pockets must be provided for the coal or it must be dumped in piles on the ground or retained in the mine cars. It would be necessary to construct storage pockets large enough to hold the output from two shifts and so designed that degradation will be small and the coal can be loaded out of them economically. If the coal were to be stored in stock piles a suitable method must be provided to handle the coal from the mine cars to the stock piles and from the piles to the preparator. This system must be so designed that the degradation of the coal will be reduced to a minimum, and the coal must be recovered cheaply. Also some method for keeping the coal from freezing in cold weather will be needed.

If the storage is to be underground, it must be either in mine cars or, if the measures are steeply pitching, in the breasts themselves. If the first is chosen, enough cars should be provided to handle the output for a period of 24 hr. Some, however, would probably be enough to hold the output for two full shifts and for part of the shift that is being worked while the preparator is running. Although it has been said that this storage is to be underground, in warm weather all or part of the cars could be held on a siding on the surface and in cold weather a thaw-shed could be used. If the cars are to be left underground, ample additional siding space must be provided, and this is extremely expensive, especially underground.

If the measures are generally steeply pitching, the coal can be stored in the breasts during two of the shifts and only drawn out while the preparator is operating. This would mean that a somewhat smaller number of mine cars would be needed. Underground haulage force would be lessened in this case as no haulage would be done during two of the shifts, except in those parts of the mine where the measures are so level that the coal cannot be stored in the breasts. Fewer men also would be required for hoisting as this operation would go on during only one shift.

This would reduce the number of headmen and footmen, but would not entirely eliminate them during the two shifts during which no coal was being hoisted, as supplies would be handled and men raised and lowered.

Of the two methods of storage that in mine cars is the better, as the coal must be handled only once, so that there is no degradation; but a greater investment in mine cars is required, also a greater cost for sidings. If stored in piles on the surface, second handling of the coal cannot be avoided and the coal is broken and subject to attrition. The cost of providing storage room is excessive. The extra handling of the coal increases the operating expenses.

The converse of this case is that in which the preparator is to be operated on three shifts and the mine on one shift. Here it is again necessary to store coal. Two-thirds of the mine output in the one shift must be handled or stored to supply the three-shift operation of the preparator. In this case, however, underground storage would be unreasonable, for if the coal were stored in the breasts, it would be necessary to have three-shift operation of all underground work except cutting the coal. If the coal was stored in mine cars, the hoist would run steadily in all three shifts. Either of these methods would make the operating costs too excessive to consider. If the storage were on the surface there would be excessive degradation and the cost of constructing sufficiently large pockets and unloading and reloading machinery, besides the cost of rehandling this coal. Therefore the conclusion must be reached that the storage of coal from the mine, because of the single-shift operation of either mine or preparator, is not feasible and would increase mining costs unreasonably.

RAILROAD CAR SUPPLY

If an anthracite mine was to be operated on the three-shift plan, it might mean that greater railroad-car empty and loaded tracks would have to be provided in order that on the off shifts a sufficient supply of cars might be on hand if the railroad would not make provision to furnish them when they were needed.

EFFECT ON MARKET

If a considerable number of anthracite collieries were put on the full three shifts and their outputs tripled, there would be a tendency to over-produce and the markets would be glutted. This would particularly affect the steam sizes which, under present conditions, are sometimes difficult to move.

SPECIFIC EXAMPLES OF TRIPLE SHIFTING

It would be well to tabulate the conditions under which triple shifting might be arranged, noting in each case the results favorable or unfavor-

TABLE 4.—Summary of Probable Changes Due to Operating in Multiple Shifts as Compared with Normal Operation

No.	Description	Mine	Preparator	Disturbances	Necessity for Contract Work	Mine Efficiency	Outside Efficiency	Miners	Pump Men	Foremen	Ventilation	Transportation	Hoisting	Silting	Development	Second Mining	Preparation, Number of Men
1	A mine of normal production operating under present methods.	1	1	None	None	Nor.	Nor.	Nor.	Nor.	Nor.	Nor.	Nor.	Nor.	Nor.	Suf.	Suf.	Nor.
2	An old mine with preparator in good condition; desire to place on full three shifts.	3	3	Yes	Yes	Dec.	Dec.	Inc.	Nor.	Nor.	Nor.	Nor.	Nor.	Nor.	Insuf.	Insuf.	Inc.
3a	Old mine with normal output, mining different beds.	3	3	None	None	Dec.	Dec.	Inc.	Nor.	Inc.	Inc.	Inc.	Inc.	Inc.	Suf.	Suf.	Inc.
3b	Same as 3a, except mining different openings instead of different beds.	3	3	None	None	Dec.	Dec.	Inc.	Nor.	Nor.	Nor.	Nor.	Inc.	Inc.	Suf.	Suf.	Inc.
4	Old mine with normal output.	1	3	None	None	Nor.	Dec.	Nor.	Nor.	Nor.	Nor.	Inc.	Nor.	Inc.	Suf.	Suf.	Inc.
5	New mine maximum output each shift.	3	3	Yes	Yes	Dec.	Dec.	Inc.	Dec.	Inc.	Nor.	Nor.	Nor.	Nor.	Insuf.	Insuf.[b]	Inc.
6a	New mine with output same as if operating under present methods.	3	1	None	None	Nor.	Nor.	Inc.	Nor.	Inc.	Inc.	Inc.	Inc.	Nor.	Suf.	Suf.[b]	Nor.
6b	New mine same as 6a but having maximum output on each shift.	3	1	Yes	Yes	Dec.	Nor.	Inc.	Dec.	Nor.	Nor.	Nor.	Nor.	Dec.	Insuf.	Insuf.[b]	Nor.
7	New mine with normal output.	1	3	None	None	Nor.	Dec.	Nor.	Nor.	Nor.	Nor.	Inc.	Nor.	Inc.	Suf.	Suf.[b]	Inc.
8	Consolidation of three mines, each operating on different shifts.	3	3	Yes	None	Dec.	Dec.	Inc.	Nor.	Nor.	Nor.	Nor.	Nor.	Nor.	Suf.	Suf.	Inc.

9	Consolidation of three mines, each operating on different shifts.a	3	1	Yes	None	Dec.	Nor.	Inc.	Nor.	Nor.	Nor.	Nor.	Nor.	Nor.	Suf.	Suf.	Nor.
10	Consolidation of three mines all on one shift.	1	3	None	None	Nor.	Dec.	Nor.	Nor.	Nor.	Nor.	Nor.	Nor.	Inc.	Suf.	Suf.	Inc.
11	Consolidation of three mines; abandon two and triple output of third.b	3	3	Yes	Yes	Dec.	Dec.	Inc.	Dec.	Nor.	Nor.	Nor.	Nor.	Nor.	Insuf.	Insuf.	Inc.

a In this case, the capital invested in the two other mines would stand idle and the upkeep cost of pumping, repairs, and watchmen would continue.

b In case of new mines that were just starting and were not reopened old ones this would not apply.

NOTE.—Inc. means increase; Dec. means decrease; Insuf. means insufficient; Suf. means sufficient; Nor. means normal as compared to assumption No. 1; Dif. means that difficulties might arise.

TABLE 4.—Continued

Assumptions		Out-side, Number of Men	Cost of Mine Development	Cost of Preparator Equipment	Supply Account	Cost of Power	Coal Storage	Effect on Market	R.R. Car Supply	Number of Items			Conclusions
No.	Description									Favorable to Operation	Normal	Unfavorable to Operation	
1	A mine of normal production operating under present methods.	Nor.	Nor.	Nor.	Nor.	Nor.	Nor.	Nor.	Nor.	0	22	0	This is present normal method of operating an anthracite mine to which all of other assumptions are compared.
2	An old mine with preparator in good condition; desire to place on full three shifts.	Dec.	Dec.	Dec.	Dec.	Dec.	No	Bad	Dif.	6	6	10	Both inside and outside operating costs would decrease but preparation cost would increase. Serious labor troubles would arise and efficiency of men would be reduced. It would also be impossible to get sufficient development and market could not absorb coal.
3a	Old mine with normal output, mining different beds.	Inc.	Nor.	Nor.	Inc.	Inc.	No	Nor.	Dif.	0	9	13	Does not show any items in favor of multiple-shift operation; therefore it would be less favorable than one-shift operation.
3b	Same as 3a, except mining different openings instead of different beds.	Inc.	Nor.	Nor.	Inc.	Inc.	No	Nor.	Dif.	0	13	9	Labor efficiency would decrease, outside and preparation costs would increase, as would power and supply accounts; not one item is in favor of this assumption.
4	Old mine with normal output.	Inc.	Nor.	Nor.	Inc.	Inc.	Yes	Nor.	Dif.	0	13	9	Mine operation is practically normal; increased costs of preparation, together with storage of coal, make this assumption unfavorable.
5	New mine maximum output each shift.	Dec.	Dec.	Dec.	Inc.	Dec.	No	Bad	Dif.	5	5	12	Labor troubles together with increased cost of preparation and flooding of market make up for any advantage gained by increased use of capital and decreased cost of power; there would be insufficient development.
6a	New mine with output same as if operating under present methods.	Inc.	Inc.	Nor.	Inc.	Inc.	Yes	Nor.	Nor.	0	11	11	No advantage will be gained for operating under this assumption, operating costs are heavier and there would not be enough development.

6b	New mine same as 6a but having maximum output on each shift.	Dec.	Dec.	Dec.	Nor.	Dec.	Yes	Bad	Nor.	6	8	8	Labor troubles and decreased efficiency together with securing sufficient working places and storage of coal offset any gain in reduction of operating expenses or more continuous use of capital.
7	New mine with normal output.	Nor.*	Nor.	Dec.	Inc.	Dec.	Yes	Nor.	Dif.	2	13	7	Decrease in cost of preparator and power would not make up for storage of coal and increased operating expenses.
8	Consolidation of three mines, each operating on different shifts.	Inc.	Nor.*	Dec.	Dec.	Dec.	No	Nor.	Dif.	3	12	7	Decreased efficiency of men, together with labor disturbances, increased outside and preparation costs, are not counterbalanced by decreased amount of capital, supply account and power costs; therefore assumption is not feasible.
9	Consolidation of three mines, each operating on different shifts.	Inc.	Nor.	Nor.	Nor.	Nor.	Yes	Nor.	Nor.	0	17	5	Nearest of any assumption to present normal methods, but five items in this method are not as favorable as in one shift operation.
10	Consolidation of three mines all on one shift.*	Nor.	Inc.	Dec.	Inc.	Dec.	Yes	Nor.	Dif.	2	13	7	Increase in cost of preparation, storage of coal, and cost of mine development are not counterbalanced by decreased preparator equipment and power costs.
11	Consolidation of three mines; abandon two and triple output of third.*	Nor.	Dec.	Dec.	Nor.	Nor.	No	Nor.	Dif.	3	10	9	Three items of decrease in number of pumpmen, cost of mine development, and preparator equipment, do not counterbalance losses caused by labor troubles, decreased efficiency of men, impracticability of getting sufficient working places; so this assumption is unfavorable.

* Or Increase.

NOTE.—*Inc.* means increase; *Dec.* means decrease; *Insuf.* means insufficient; *Suf.* means sufficient; *Nor.* means normal as compared to assumption No. 1; *Dif.* means that difficulties might arise.

able, deviating from the time-honored plan of single shifting mine and preparator.

Assumption 1.—A mine of normal production operating under present methods.

Assumption 2.—An old mine with the preparator in good condition and operated with first and second mining; it is desired to place the mine on full three-shift operation to triple the output.

Assumption 3.—An old mine that can be operated by dividing the present force into three parts, each part working an integral portion of the mine, and operating the preparator on a three-shift basis. (*a*) By working different beds in different shifts; (*b*) by working different shafts, slopes, or drifts in different shifts.

Assumption 4.—The whole mine worked on a one-shift basis and the preparator on a three-shift basis.

Assumption 5.—A new mine, a preparator to be so planned as to operate on three shifts.

Assumption 6.—A new mine to be laid out to operate on three shifts and the preparator on one shift. (*a*) The output to be the same as operating under present methods; (*b*) the output to be the maximum for each shift.

Assumption 7.—A new mine to be designed to operate on one shift and a new preparator on three shifts,

Assumption 8.—A consolidation of two or three collieries, each to be operated on a separate shift, and the preparator to be operated on a three-shift basis.

Assumption 9.—A consolidation of two or three collieries each to be operated on three shifts and the preparator on a single shift.

Assumption 10.—A consolidation of two or three collieries, all to be operated on the same shift and the preparator on three shifts.

Assumption 11.—Two of three collieries to be abandoned and work to be concentrated in one colliery producing the same tonnage as obtained from the three; both colliery and preparator to be operated on three shifts.

CONCLUSION

The ten main conditions that must be considered in the multiple shifting of anthracite collieries are shown above and in Table 4; besides these others might arise in the case of any particular colliery.

There may be individual cases where multiple shifting of anthracite mines might be advisable due to peculiar conditions that are not general throughout the fields. It might be advisable, in case of accident to one operation, to give employment and maintain output, or if for any cause there is a serious shortage of coal, or on account of the pending expiration of a lease on a property. Such an operation would not be multiple

shifted for any saving in operating expenses or continous use of capital but on account of special conditions.

Double shifting instead of triple has not been mentioned but the conditions and the items affecting this would be similar but less in degree.

From a study of the preceding, the effect of the different assumptions on the operation is shown and it appears that not one of the assumptions shows as economical and practicable a method of operation as the usual—a mine that operates on one shift and preparator that also operates on the same shift.

DISCUSSION

R. V. Norris, Wilkes Barre, Pa.—The author has made a careful study of the results of the operation of nearly half of the anthracite region and has shown the economies of production in large units. The diagrams should be on a much larger scale as they will be widely used for reference in studies of future installations, and in the analyses of costs. Necessarily, the upper ends of the curves for very large unit output are based on relatively few observations and it is reasonable to suppose that some changes might be necessary in these were a larger number of observations available.

It is interesting to find, in Fig. 9, that up to 1,000,000 tons output, taking the single-shift operation as a base, the double shift roughly requires 16½ per cent., and treble shift 30 per cent. additional employees; this regularity in the final result tends to confirm the accuracy of the figures and the methods employed.

L. M. Kniffen.—Two assumptions that should receive further consideration are numbers eleven and five. The first covers the case of an existing group of collieries where one preparator is operating at a much smaller cost per ton of output than the others. Under these conditions it might be possible to make large savings by operating the best equipment three shifts per day and dismantling two other structures of like capacity. This would not require large capital expenditure and the improved working conditions might overcome the disadvantage of requiring the men to change shifts.

Similar consideration might also be applied to the mining conditions. In an existing group of properties considerable savings might be made by concentrating work in one mine with resulting savings in the operation of haulageways, slopes, etc. The first mine in this case would be exhausted and abandoned at an earlier date so that maintenance cost could be cut off.

The consolidation of collieries is receiving close engineering attention at present and three-shift operation offers a means that should be given consideration in the estimates. It will be necessary to take account of all

local conditions and the general comparisons given in the paper would not apply to this special condition.

In assumption 5, the problem is less complicated, for the transportation, hoisting and preparation can be done on a less expensive scale. Under these conditions the conclusions arrived at from a study of the curves would not apply, for new equipment intended for three-shift operation would be built to avoid the difficulties pointed out. The paper shows that the output per man decreases as the output decreases; the reason why the change is so marked is that the preparators shown are

COMPARISON OF OPERATING COSTS, 300 DAYS PER YEAR, STRUCTURAL STEEL PREPARATOR

PREPARATOR	ONE UNIT 1000 TONS PER 8 HR. OR 3000 TONS PER 24 HR.		THREE UNITS 3000 TONS PER 8 HR. OR 3000 TONS PER 24 HR.	
	TOTAL PER YEAR	COST PER TON	TOTAL PER YEAR	COST PER TON
Interest, at 8 per cent......	$14,400.00	$0.016	$ 43,200.00	$0.048
Amortization, at 5 per cent..	9,000.00	0.010	27,000.00	0.030
Taxes, at 2 per cent.........	3,600.00	0.004	10,800.00	0.012
Fire insurance, at 0.2 per cent.	360.00	0.000	1,080.00	0.001
Depreciation at 1.5 per cent.	2,700.00	0.003	8,100.00	0.009
Power....................	18,000.00[a]	0.020[a]	28,800.00[b]	0.032[b]
Total..................	$48,060.00	$0.053	$118,980.00	$0.132

Difference $0.079. Interest and amortization at 12 per cent. Leased property royalty basis.

[a]1.0c. per kw.-hr.   [b]at 1.6c. per kw.-hr.

	AMOUNT	AMOUNT PER YEAR	COST PER TON	AMOUNT	AMOUNT PER YEAR	COST PER TON
Underground development and equipment....	$250,000	$30,000	$0.033	$400,000	$48,000	$0.053
Hoist and misc. equipment................	75,000	9,000	0.010	100,000	12,000	0.013
R. R. yards and sidings..	60,000	7,200	0.008	60,000	7,200	0.008
			0.051 ·			0.074
Difference,			$0.023			

operating at but little more than half capacity. If a colliery designed for three-shift operation was found to be over capacity some of the shifts might be discontinued.

It is stated that the cost of a preparator one-third the size of another will· be approximately 100 per cent. more per ton of output. This applies to most existing conditions, but in some recent structures of large capacity the unit principle has been adopted in order to simplify construction and operation; this is regular metal-mining practice. This condition affords a means of comparing the two systems of operations under the same conditions. As each unit is complete in itself, the

construction cost of each should be about the same. A reduction in the mining capacity would therefore mean a reduction in the number of units. The cost of a one-unit plant to operate three shifts per day would be just about one-third of the cost of a three-unit plant to turn out the same quantity in 8 hours.

The same condition applies to the force of men required. It is general practice for each unit to have its complete crew so that outside of superintendence a one-unit plant should require about one-third as many men as a three-unit plant.

On this assumption a table has been prepared which shows the effect on the various items of operating cost that depend on the amount of capital invested in the equipment. This is based on an assumed case of a new property which is to be equipped to yield 3000 tons by the two methods of operation.

It will be noted that the possible profits that would otherwise be obtained would be an item of some consideration.

The amount of development work per ton of coal mined would remain approximately the same whether the mine were operated on one shift or three. The faces in coal would be kept open a shorter length of time and a smaller number would be required for a given colliery output.

It has been stated that there is difficulty in keeping development work sufficiently far in advance of the coal-mining forces. It is possible that a more general intensive operation of the collieries would bring about faster and cheaper driving of development forces, as has been the case in metal-mining practice.

As the day time is the natural time for men to work we should hesitate to change the long-established custom of the coal-mining regions, which allows nearly all the employees to spend the natural recreation and rest part of the day with their families. The small margin of saving that three-shift operation presents might well be considered as an investment in the welfare of the workers.

WM. H. GRADY, Duncott, Pa.—Many of the men here know that the face of a breast today is not working 8 hr.; 20 or 40 years ago the face of a breast was worked very much more than 8 hr.; today 4 hr. is the estimated time worked. I believe that is the gist of the matter; whether or not a breast can be worked continuously. I believe it can.

DOUGLAS BUNTING, Wilkes-Barre, Pa.—In the paragraph entitled Underground Forces, the statement "The miners and their helpers constitute only 30 to 40 per cent. of the men employed" appears to be in error. Taking the four operations cited and totaling up the working forces gives 1174 miners and laborers, 817 company men, and 53 outside company men, or 46 per cent. miners and laborers, 33 per cent. inside

company men, and 21 per cent. outside force.   If the underground forces only are considered, the percentage of these forces would be 59 per cent. miners and laborers and 41 per cent. company men.   From my personal observation, the statement should be 40 to 50 per cent. in place of 30 to 50 per cent.; the four operations cited gives 46 per cent.

One matter in connection with this two-shift or three-shift operation that was not mentioned is the hoisting and lowering of men and supplies. The curves given will be referred to by many engineers and will prove of value.   It may be that they have not been compiled from as many operations as they should have been; that the figures from which they have been compiled have not been compiled with as great accuracy as they might have been; that the forces involved as reported do not include all of the men employed at the operation, as for illustration "floating gangs;" and that they have simply taken the force as appearing upon the colliery pay rolls.   A number of the anthracite operations that carry "floating gangs" may not have been included in this report.   To get at a true record of the force employed every employee should be counted. But, nevertheless, the curves are good and of value.

W. S. AYRES, Hazleton, Pa.—Although metal-mine engineers have characterized the methods used in the anthracite region as old-fashioned and antiquated and entirely out of date, I have always reasoned that the anthracite engineers were operating according to plans developed by experience.   Cost sheets and much other data have been secured for their guidance.   This applies to mining and transportation under and above ground as well as preparation.   The problems to be met are not only numerous but varied and complicated.   From the data given in this paper, a fair forecast can be made as to what a new operation can be made to do, and also what returns the remodeling of an old plant will yield.

To establish these average curves was no small task, and the great variations of the individual results show too that improvements can possibly be made in those operations below the average efficiency, and that the engineer must be constantly on the lookout for local advantages. This presentation of the problem opens up many fields for investigation in individual operations and furnishes a standard average efficiency below which any deviation should receive careful analysis.

In Table No. 3, the item "Tons prepared per day per employee" should read "Tons per capacity."   It does not give the average tons prepared per man, which I find to be: for No. 1, 27; for No. 2, 31; for No. 3, 34; for No. 4, 31; and for No. 5, 61.

It is almost an impossibility to acquire a three-shift operation with the men we now have, whether the machinery was designed for it or not. It is almost impossible to get three shifts into the one mine.

ELI T. CONNER, Scranton, Pa.—The author appears to prove conclusively the negative side of the question; there must be another side to it, however.

J. B. WARRINER, Lansford, Pa. (written discussion).—I agree with Mr. Conner that the author has proved conclusively the negative side of the question; in fact, I think that he has taken the negative side too much for granted. I do not believe that the difficulties in the way of establishing a double-shift or triple-shift system of operation that he mentions are at all insurmountable. I do question, however, the practical value of double or triple shifts under present conditions.

An Inventory of Results of Accident Prevention

C. A. ALLEN,* SALT LAKE CITY, UTAH

FOR over three years we have been endeavoring to reduce the number of fatalities and minor accidents in Utah. The physical condition of the
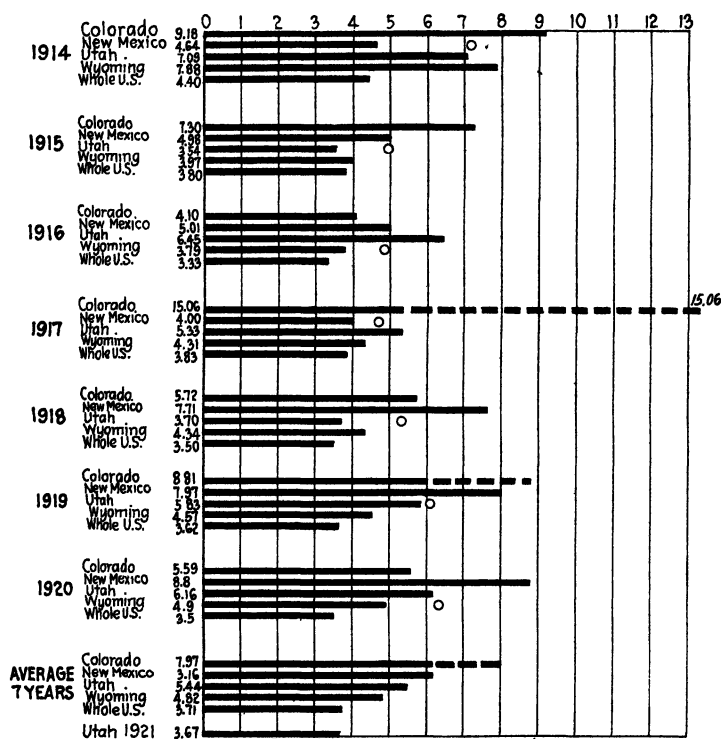
NOTE: Dotted part of line represents catastrophes in Colorado.
o-Represents the average of 4 States mentioned excluding the catastrophes indicated.

FIG. 1.—NUMBER KILLED PER MILLION SHORT TONS OF COAL PRODUCED IN COLORADO, NEW MEXICO, UTAH, WYOMING, AND BITUMINOUS MINES OF THE UNITED STATES, 1914 TO 1920.

mines, concentrators, and smelters is good, yet the number of accidents,

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* Mining Engineer, U. S. Bureau of Mines; Chief Mine Inspector for Industrial Commission of Utah.

especially in the coal mines, has been much higher than it should be; though compared with all the mines of the United States, the metal-mine accident rate has been good. This has been a source of wonder to us because we are proud of the physical condition of our coal mines, just as much as of our metal mines; in fact, because we have a number of large new coal mines developed along the most modern lines, we believe that, taking the state as a whole, we have just as good coal mines as any state in the Union.
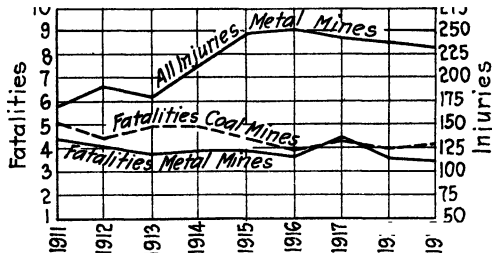
FIG. 2.—FATALITIES AND INJURIES IN METAL MINES AND FATALITIES IN BITUMINOUS COAL MINES PER 1000, 300-DAY WORKERS, 1911 TO 1920. WHOLE UNITED STATES.

The fatality rates in the coal mines of Colorado, New Mexico, Utah, and Wyoming, as compared with those in the United States, for the years 1914–1920, are given in Fig. 1, which shows that something is wrong with the coal mines not only in Utah but in those of at least three other western states.

To determine whether the lack of improvement in lowering accident rates was peculiar to Utah and the West, or was common to the whole United States, Figs. 2 and 3, showing the number of men killed in coal

(a) - One-third of fatalities due to snow slides
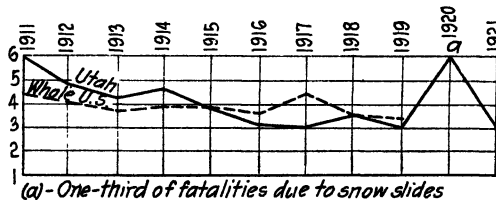
FIG. 3.- -FATALITIES IN METAL MINES OF UTAH AND WHOLE UNITED STATES PER 1000, 300-DAY WORKERS, 1911 TO 1920.

and metal mining since 1910, were prepared. Considering the efforts in safety work during these years, conditions do not show the improvement they should. Every fatality in Utah is investigated by a mine inspector, and their reports were carefully gone over in an effort to place the responsibility for the deaths. Out of 57 fatalities in the mines, quarries, mills, and smelters, in 1919 and 1920, the responsibility could be placed about as follows:

	OPEN-PIT MINES AND QUARRIES, PER CENT.	UNDER-GROUND MINES, PER CENT.	MILLS AND SMELTERS, PER CENT.
Fatalities due to conditions that could have been avoided by companies..............	30	6½	30
Laxness of foremen or bosses, in some cases shared by man killed...................		16	
Fault of fellow employees................	40	6½	
Carelessness of man killed................	10	29	20
Purely accidental........................	20	42	50

Out of 63 fatalities in the coal mines in 1919 and 1920, we placed the responsibility as follows:

PER CENT.

Fault of man killed.. 39
Due to conditions that could have been avoided by company...... 9
Fault of fellow employee...................................... 11
Purely accidental... 41

Fig. 4 shows the total accident rate at the concentrating mills in Utah, the plants being arranged in the order of the efficiency of their mechanical safeguards. Fig. 5 shows the accident rates at a number of
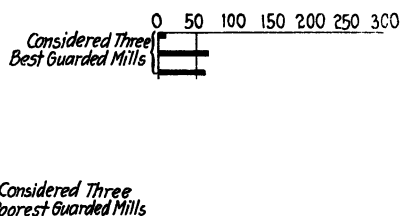
FIG. 4.—ACCIDENTS PER 100,000 SHIFTS IN ORE MILLS OF UTAH, 1920.

the large metal mines, which are arranged in the order of the number of men employed; and Fig. 6 shows the same data for the larger coal mines of the state. We realize that some of the data shown do not cover sufficient time to make conclusive the deductions to be drawn. Also in 1920, on which Figs. 4, 5, and 6 are based, the accident rates were the highest in both metal and coal mines. Probably a year of high accident rates is the best to study but, at any rate, it was the only year for which complete figures were available.

Fig. 4 shows that some of the mills that were considered to be the best equipped with mechanical safeguards, such as guards for gears, belts, and pulleys, and railings for stairways and runways, had unusually high accident rates; whereas poorly guarded mills showed low accident rates.

In Fig. 5, the accident rates at eleven of the larger underground metal mines are shown, the mines being arranged in the order of the number of men employed, which ranges from 75 to nearly 500 men. It will

be noted that six of the mines had a higher accident rate than the state total, whereas only four had a lower rate, which proves that the smaller mines not shown in the figure had still lower accident rates. The two mines with the most accidents were both mines in good physical condition
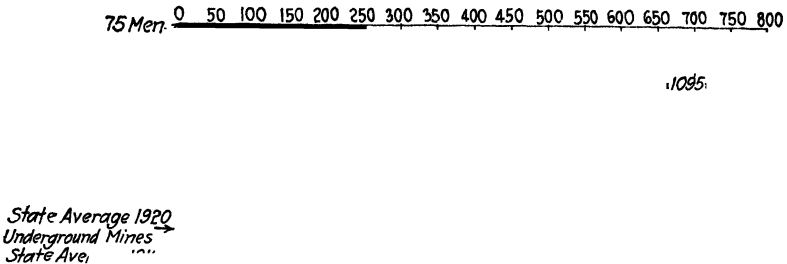
FIG. 5.—ACCIDENTS PER 300,000 SHIFTS IN LARGER UNDERGROUND METAL MINES ARRANGED IN ORDER OF NUMBER OF MEN EMPLOYED. UTAH, 1920.

but, because of their location, had difficulty in maintaining steady employees, thus giving them a high rate of labor turnover each month.

Fig. 6, shows the accidents at the larger coal mines. Knowing the conditions at the mines given, it can be positively stated that, other things being equal, the smaller coal mines have the lowest accident rates.

NOTE: Where a company has more than one mine, the average employees per mine governs their position in the above list. Accidents are not tabulated for the separate mines of the same company.
Broken part of line indicates accidents causing time loss of less than 14 days.
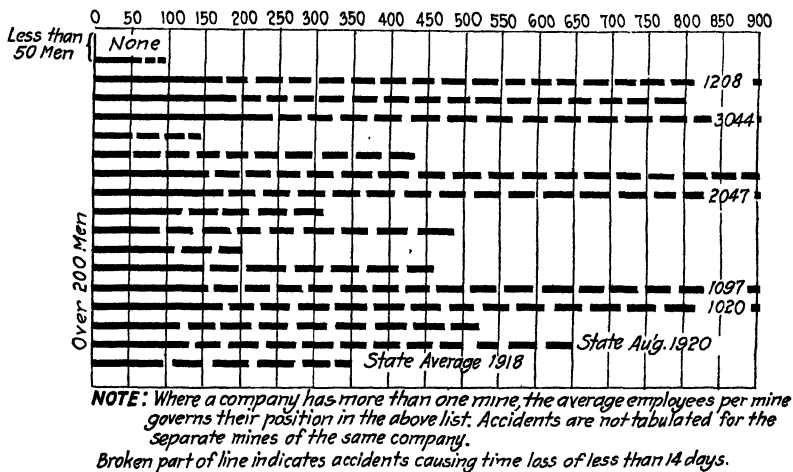
FIG. 6.- –ACCIDENTS AT THE LARGER COAL MINES OF UTAH, 1920. ARRANGED IN ORDER OF NUMBER OF MEN EMPLOYED. BASED ON 300,000 SHIFTS.

One of the two mines employing less than 50 men did not have an accident during the year; the other had the next lowest accident rate. Many of the employees of both of these mines had worked at the properties for years and the percentage of "floaters" was very low. The three mines showing the highest accident rates were not the largest mines, but were

mines having the largest labor turnover during the year; two of them also had several changes of management. Some of the largest mines had comparatively low accident rates, but most of them had more efficient safety inspection and instruction and had steady employees, which outweighed the advantages of the smaller mines.

As has been stated, during 1920, accidents were more numerous than in the years preceding or following. For most of the year, men were not plentiful, there was a large demand for coal, and the men were making extremely high wages.

A careful study of the results shown, taken in conjunction with other information, led to the following conclusions:

1. The men are responsible for most of the accidents, and efforts to reduce the number of accidents must take this fact into consideration. While there is a large proportion of good, careful, efficient, conscientious workmen, unfortunately, too large a percentage of the men are not of this type and they are the ones toward whom our efforts must be directed.

2. A mine, or more especially a plant, having the best physical condition may have the highest accident rate. When a plant is well provided with safeguards, there is a tendency for some men to become careless and to consider the safeguards as an excuse for less care and sometimes as an object of ridicule.

3. Other things being equal, small mines have proportionately much fewer accidents than large mines.

4. The efficiency and ability of the mine management is one of the most decisive factors in the prevention of accidents. When a good foreman can maintain proper discipline regarding safety measures, accidents will be few.

5. Since the passage of compensation laws by the various states, there has been a marked increase in the number of minor accidents reported, as is shown in Fig. 2. While the great benefit of these laws has been most thoroughly proved, men of a certain type have taken advantage of the laws to lay off from work when they would not have done so if they were not to receive compensation. It is also not unreasonable to assume that the knowledge that they will receive a monetary consideration if they are injured has a tendency to make some men more careless.

6. When labor is scarce and wages are high too many men will refuse to recognize safety regulations, and accidents increase.

When we found that one of our best guarded concentrating mills had the highest accident rate in the state, and one of our poorest guarded mills had one of the lowest accident rates; also when we saw that the coal mines had a much higher accident rate than the metal mines, while there had been a careful state inspection of the former, and no

inspection of the latter prior to 1917, we naturally questioned the value of safeguards and inspections.

A few other facts however, and a little careful reflection, will show that we must abandon neither our mechanical safeguards nor our inspection service. In Utah, there are four large smelters that are excellently equipped with mechanical safeguards and have well-defined safety organizations for the education of the men. When these safety organizations were inaugurated, at least two of the plants reduced their accident
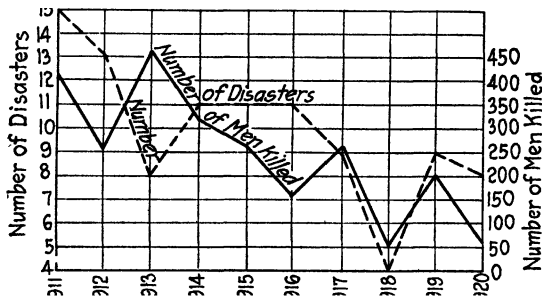
FIG. 7.—NUMBER OF DISASTERS KILLING FIVE OR MORE MEN AND THE NUMBER OF MEN KILLED BY SUCH DISASTERS, 1911 TO 1920. WHOLE UNITED STATES.

rate one-half the first year, and reduced that one-half the next year; this shows that mechanical safeguards, with proper education of the men at the same time, will produce results. In 1919, Utah required that safeguards be installed in the metal mines and, in 1920, in the coal mines; a number of instances have come to the attention of the inspectors where lives have been saved by these safeguards.

Then, too, in accident prevention the operating company must take the lead and show its interest; therefore, if we discard the safeguards referred to we would be placing all the responsibility on the men.
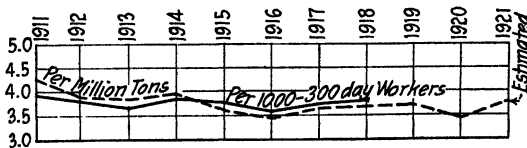
FIG. 8.—NUMBER OF MEN KILLED IN COAL MINES BY COMMON ACCIDENTS ONLY, 1911 TO 1920. WHOLE UNITED STATES. INCLUDES ANTHRACITE MINES.

Another thing that must be considered is that a number of safeguards that can be applied in mining protect the men against conditions over which they themselves have no control; for instance, men on a swiftly moving cage cannot save themselves if there is an accident caused by a defective hoist or the lack of an overwind device. Also, if an explosion occurs, the careful man suffers with the careless. Fig. 7 shows the number of disasters and the number of men killed by such disasters in

the coal mines of the United States from 1911 to 1920, and Fig. 8 shows the fatalities, eliminating those due to disasters. The curves in Fig. 7 show a decided downward trend; and when we remember that, in 1911, in fifteen major disasters in coal mines 413 men were killed, which was 15½ per cent. of all the men killed in the coal mines that year, whereas in 1920 there were only eight major disasters in which 61 men were killed, or 2.7 per cent. of all the fatalities, there is reason for congratulation. Credit for this remarkable showing can be given to the operators who have incurred considerable expense to make their mines less liable to explosions, to the state inspectors, who have insisted on strict precautions in this regard, and to the United States Bureau of Mines, which has been the leader in explosion prevention.

In Utah we saw that, while we had made considerable progress in providing safeguards and in reducing the number of accidents caused by conditions over which the men had no control, it would be necessary to do more work with the men themselves. We, therefore, inaugurated a safety campaign, in which the mine operators, the mine inspectors, and the crew of the U. S. Bureau of Mines Car No. 11 coöperated. This campaign consisted of foremen's meetings, rallies for the men, and work in the schools and other organizations. The methods by which this campaign was carried out have been described; the accompanying table shows the number of persons who attended the various meetings.

TABLE 1.—*Safety Meetings and Their Attendance*

	Smelters	Metal Mines	Coal Mines
Foremen's meetings held..................	3	6	16
Number attending.......................	209	176	184
Safety rallies held......................	3	6	19
Number attending.......................	345	1455	3962
Men trained in first aid..................	209	210	487
Children enrolled as safety and health scouts.	0	1079	1405

The number of fatalities in mining in Utah during 1921 was much lower per 1000 employed than the average of the last seven years, as shown in Figs. 1 and 3 and, no doubt, some credit must be given to this safety campaign. It enlisted the efforts of the foremen and the good miners, and due to the plentiful supply of men, the foremen were enabled to enforce strict discipline. This was also the first year in which the coal mines were brought up to the standards of the new safety regulations accepted by the operators and the state in 1920. The results of a campaign of this kind, however, are but transitory, and something

more permanent is necessary. Real training of a large percentage of our miners is imperative before accident prevention will attain its ultimate success; the methods for doing this have not been fully developed. Large machine shops and factories of the East, I understand, have solved the problem satisfactorily, but mining presents a somewhat different problem. The following methods seem the most likely to result in success.

1. The principles of safety and efficiency should be inculcated in the minds of the school children in the mining camps; also the spirit of taking pride in their work.

2. The apprentice system should be adopted wherever the conditions of the work will permit. By paying the apprentice less money, the skilled man may receive more without increasing the total cost of the products. In the coal mines, a man just arrived from foreign lands frequently gets the same wages as an experienced miner; and boys 17 years old get the same wages as their fathers, which is not only an injustice but also gives the boys more money than they know how to spend properly. Contrast this with the conditions surrounding the entry of the Scottish youths into the coal mines. They receive a mere pittance for helping their fathers or older brothers, but into their minds is inculcated every principle of coal mining, including how to take care of themselves.

3. In British Columbia, a man must pass an examination before he can work as a miner and receive first-class miners' wages; this is essentially the apprentice system applied underground.

4. Probably the most effective way to train men and get immediate results is through the foreman and any way of increasing his knowledge or his ability to impart it is a step in the right direction. The requirements of examining boards have undoubtedly given us a better class of men in these positions but it would be still better if applicants for mine foremen were required to spend a certain time in a school of instruction. In metal mines there are no requirements for foreman, at least in most states, with the result that many of the smaller mines are handled by men who know little about metal mining.

The writer is familiar with three instances of foreman training in mines that, apparently, were very successful. Two of these, one at the mines of the North Butte Mining Co., and the other at the Copper Queen branch of the Phelps Dodge Corpn. at Bisbee, have been described; the third was conducted at one of the coal mines of Utah. The methods at these mines differed largely but they produced more efficient foremen and other bosses.

One objection to any method whereby the state demands a stricter examination of a foreman or of a miner is that most of those in the operating end of the industry are claiming that the state already exerts too paternalistic an influence and there should be less interference rather than more, but if they desire to avoid state interference, they must them-

selves take the necessary steps to overcome the present conditions, and there is no doubt they can do it, as well as the state, if they will.

Labor unions also have their part to play. They must not demand that because a man holds a certain card, he must receive the same wages as all others holding the same card. A man's wages must be based on his training and ability and that training should include the proper knowledge of how to protect himself from injury. Self protection is the first law of nature, and it should not require a special campaign to make men obey that law.

In conclusion, I might reiterate that while we have made some progress in accident prevention, the results have not been what they should, and will never be until we more properly train the workmen; and this training should take the form of making men more thorough, more efficient, more proud of their work; then they will be more careful. I have tried to indicate some of the ways in which men can be trained, but whatever the method, the outstanding fact is, we must have more trained men in and around our mines.

Coal-mine Ventilation

By Joseph J. Walsh,* Nanticoke, Pa.

VENTILATION within a coal mine is essential to the welfare of those employed therein, from the standpoint of health, safety, and efficiency. While the saving of life and the preserving of health are the chief reasons for ventilation, efficiency and production are increased.

In spite of the fact that we speak of air as having a certain composition, the proportions of oxygen, nitrogen, and carbon dioxide are not fixed, but vary between small limits. The different elements are not chemically united, they are merely mixed and each is free and can be separated with little energy. For this reason air in the mine workings is seldom found in a pure state.

Factors that necessitate ventilation in coal mines are those that change the composition of pure air. Contamination of mine air results from the burning of oil-fed flames, the use of explosives, the oxidation of coal, and the liberation of gases by the coal. The extent to which these factors exist governs the size and general plan of the ventilating appliances required. An efficient ventilating system is one that will furnish air at the working faces in sufficient volume to dilute and render harmless noxious gases and at the same time maintain an oxygen content closely approaching that of pure air. In a well-ventilated mine, the oxygen seldom falls more than 0.5 per cent. below normal.

The carbon-dioxide content of the air in the average coal mine will vary from 0.1 to about 0.8 per cent., which amounts are not harmful. Three per cent. of carbon dioxide in air doubles the frequency and the depth of breathing; it is plainly noticeable and if breathed continuously for several hours its effect would probably be harmful. Slight reductions in the oxygen content of the air, such as occur in well-ventilated mines, have no debilitating effect on the worker. A person not exerting himself would fail to observe any unusual physical effect in an atmosphere containing 17 per cent. of oxygen. This condition, however, or even a condition showing an oxygen content below 20 per cent., would indicate poor ventilation and that the elements which tend to reduce the efficiency of the worker are present, namely water vapor and a high wet-bulb temperature.

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The wet-bulb temperature of the mine air controls almost entirely the physical condition of the human body and is the principal determining factor of the efficiency of the workman.

The temperature of the human body is about 98° F., and to be in a proper physical condition this temperature must be kept constant within small limits. This bodily state is controlled by certain organs, by means of which the bodily heat is expelled, aided principally by radiation and evaporation. At high temperatures, the cooling effect produced by radiation is not sufficient for the needs of the body, in which case the burden is shifted to evaporation. If the relative humidity is 100 per cent., or nearly so, there will be little or no evaporation; consequently no cooling effect will be obtained from this source and the physical condition and the efficiency of the worker will be reduced. The only practical remedy for a condition of this kind is found in increasing the velocity of the air.

For the purpose of determining the extent to which the efficiency of a man is reduced by reason of a high wet-bulb temperature, the data given in Table 1 were collected by the writer. Table 2 shows the extent to which production was increased by increasing the velocity of the air in certain mines. Before the ventilation system was improved, the velocity of the air current was not sufficient to turn the anemometer; after the changes were made, the velocity was from 100 to 150 ft. per min. The principal retarding element in the way of efficiency appears to be the high wet-bulb temperature.

BEFORE IMPROVING VENTILATION   AFTER IMPROVING VENTILATION

Mine No. 1

Carbon dioxide, per cent.....	0.50	Carbon dioxide, per cent. ..	0.32
Oxygen, per cent............	20.23	Oxygen, per cent.....	20.46
Nitrogen, per cent..........	79.27	Nitrogen, per cent.	79.22
Dry-bulb temperature.......	68.0° F.	Dry-bulb temperature. ...	69.0° F.
Wet-bulb temperature.......	65.0° F.	Wet-bulb temperature.	60.0° F.
Velocity...........	poor	Velocity................. .	fair
Humidity, per cent.........	86.0	Humidity, per cent..... ...	60.0
Pounds of vapor per 1,000 cu.		Pounds of vapor per 1,000 cu.	
ft. of mixture.............	0.926	ft. of mixture	0.667
Dew point.................	63.5° F.	Dew point....	54.3° F.

Mine No. 2

Carbon dioxide, per cent.....	0.34	Carbon dioxide, per cent ..	0.15
Oxygen, per cent............	20.42	Oxygen, per cent	20.60
Nitrogen, per cent..........	79.24	Nitrogen, per cent.	79.25
Dry-bulb temperature.......	68.0° F.	Dry-bulb temperature.... ..	59.0° F.
Wet-bulb temperature.......	67.0° F.	Wet-bulb temperature.......	57.0° F.
Velocity.................	poor	Velocity...............	good
Humidity, per cent.........	95.0	Humidity, per cent.	89.0
Pounds of vapor per 1,000 cu.		Pounds of vapor per 1,000 cu.	
ft. of mixture.............	1.023	ft. of mixture.	0.713
Dew point.................	66.6° F.	Dew point.................	55.7° F.

Mine No. 3

Carbon dioxide, per cent.....	0.37	Carbon dioxide, per cent....	0.22
Oxygen, per cent............	20.36	Oxygen, per cent...........	20.54
Nitrogen, per cent..........	79.27	Nitrogen, per cent..........	79.24
Dry-bulb temperature.......	69.0° F.	Dry-bulb temperature.......	68.0° F.
Wet-bulb temperature.......	66.0° F.	Wet-bulb temperature.......	61.0° F.
Velocity..................	poor	Velocity..................	fair
Humidity, per cent..........	85.0	Humidity, per cent	67.0
Pounds of vapor per 1,000 cu.		Pounds of vapor per 1,000 cu.	
ft. of mixture	0.945	ft. of mixture............	0.721
Dew point................	64.5° F.	Dew point................	56.7° F.

When one considers the conditions making it necessary to ventilate a mine and that the weight of the air passing through a large mine in a year will exceed six million tons, the necessity for the proper construction of the airways and for a correctly proportioned fan is apparent. The fans used in the anthracite field of Pennsylvania vary in size from 3 to 35 ft.; they are operated by engines ranging from 20 to 250 hp., and generate water-gage pressures from 0.5 to 5 inches.

The most economical practice is to have the fans operate on independent intake and return airways. Where two or more fans operate on the same intake and return, the benefit obtained in air volume does not warrant the cost of erecting the additional fan. The volume of air that will flow through the mine depends on the difference in pressure between the intake and return airways. If an exhaust fan, the dimensions of which are within practical limits, operating on a mine and producing a water-gage pressure of 2 in. cannot furnish sufficient air, a second fan placed at the top of the same upcast and operated under equal pressure will not by any means double the quantity of air. Both fans operating together (exhausting), each producing a 2-in. pressure, will not create a greater difference in pressure between the intake and return airways than is created by one fan operating alone. In fact, if the rim speeds of the fans are not equal the volume of air may be reduced. We may go still further and say that it is possible, even in practice, for the rim speeds of the fans to be such that one will receive some of its air supply through the chimney of the other.

Booster fans are usually located underground between the intake and return, or at any point in the intake or return, to help along a feeble air current. Conditions requiring their use in mines exist where the movement of the air is broadcast, and where much of the air produced by the surface fan is lost by leakage. A booster fan placed in a mine to assist a fan located on the surface will not increase the total volume of air passing through the mine to any appreciable extent, unless it is more powerful and generates a greater pressure than that generated by the surface fan; neither will it be of any local value unless it generates a pressure greater than that existing at the point of installation.

Fans placed in tandem, one at the top of the upcast shaft (exhausting) and the other at the top of the downcast shaft (blowing) are of little value to each other. Assuming that an exhaust fan is producing a 2-inch pressure, if a force fan is placed at the top of the downcast shaft and run at a speed sufficient to generate an equal pressure it is only capable of producing a velocity in the downcast shaft slightly greater than that produced by the exhaust fan working alone. If the rim speed of the blowing fan is increased until the water-gage reading is 3 in., while the exhaust fan remains the same, the velocity of the air will be increased. This increase is maintained entirely by the blowing fan, because the velocity in the air is greater than that which the exhaust fan with its 2-in. pressure is capable of producing.

The belief is quite general that two fans working in tandem or two fans exhausting at the top of the same upcast, each generating the same water gage, will deliver twice as much air as one of them working alone; this however, is not true. In order to double the quantity of air flowing through the same intake and return, the pressure must be increased four times and the horsepower eight times.

The volume of air flowing through a mine can be doubled, however, by the application of twice the horsepower if the ventilating units are independent of each other; that is by the installation of two fans, each having its own intake and return.

DISCUSSION

W. B. DALY,* Butte, Mont.—Were any tests made or any measurements taken before the installation of the second fan?

J. J. WALSH.—They were installed many years ago; I think the only purpose was to get a larger volume of air.

W. B. DALY.—We had almost 100 per cent. success in determining our requirements before purchasing our fans; in only one case in over forty did we fail, and that was only by 10,000 cu. ft. We do not make an installation until we know our requirements.

J. J. WALSH.—Many fans have been installed at mines in the anthracite field, few of which produced the quantity of air guaranteed by the manufacturer. The fault, however, is not always with the manufacturer; in the proportioning of a fan he is guided by certain information furnished by the mine operator. In order to proportion a fan correctly, it must be first known what water gage will be required to force the desired volume of air through the mine. I fear many operators do not give this

* Assistant Manager of Mines, Anaconda Copper Mining Co.

factor sufficient attention when ordering a ventilating fan, and, as a result, most fans do not come up to the specifications.

W. B. DALY.—We have found that all of the fans have come up to the guarantee of the manufacturers, allowing, of course, the loss for elevation.

J. J. WALSH.—Then you must have furnished the correct water gage.

W. B. DALY.—Oh, yes.

R. R. SAYERS,* Washington, D. C.—The deficiency of oxygen and the increase in carbon dioxide must be considered with the temperature, humidity, and air movement; they cannot be separated from these factors. Only $\frac{1}{2}$ per cent. of CO_2 in any of the cases would give a slight increase in lung ventilation; 2 per cent. of CO_2 in air would give 50 per cent. increased lung ventilation, which would cause a more rapid fatigue of the man, especially if there were no air movement at that time and if there were oxygen deficiency.

H. H. STOEK,† Urbana, Ill.—Some of us who are trying to investigate ventilation from the laboratory side would like to know what can be done in the laboratory that will be useful in connection with the whole problem of ventilation.

GEORGE S. RICE,‡ Washington, D. C.—We are continuing work similar to that conducted for the New York-New Jersey Vehicular Tunnel Commission and are trying to determine the coefficients of friction of air currents on mine surfaces and loss of head in turns, etc. We have installed apparatus of precision in the experimental mine near Pittsburgh for this purpose and the investigations are under way. I was greatly impressed, when I visited the different wind tunnels of the Bureau of Standards and the Navy, to find what effect the various kinds of obstacles have on the flow of air and the importance of using stream-line cross-sections for any necessary obstructions to the flow of air. For example, the obstructive effect of a round rod to flow of air is four times that of a stream-line section of the same cross-sectional area.

So far, only a measuring station has been installed, but with the help of Professor Willard, of the University of Illinois, and Doctor Briggs, of the Bureau of Standards, we hope to standardize methods and apparatus for the determination of the coefficients of friction, so that we may have comparable results when engineers carrying out tests in mines in the different parts of the country secure data on ventilation.

* Chief Surgeon, U. S. Bureau of Mines.
† Professor, Mining Engineering, University of Illinois.
‡ Chief Mining Engineer, U. S. Bureau of Mines.

' JAMES W. PAUL,* Pittsburgh, Pa.—So·far we have utilized only the measuring station used in the tests made for the Hudson River tunnel and have found it necessary to make some alterations in order to get as nearly as possible a straight flow of air. It is difficult to secure a straight flow of air in any kind of conduit even though it is uniform in cross-section and of very smooth surface. To secure this straight flow of air, we are installing, throughout the entire cross-section of the mine at one point, a honeycomb consisting of 180 galvanized iron tubes 4 in. in diameter by 15 in. long. It is necessary to secure a straight flow of air to measure the static head at the measuring station. Mining men usually speak of the water gage as 1 in., ½ in., or ¼ in., but seldom less than ¼ in. In our investigations we are reading pressures and velocity measurements down to 0.0001 in. by the use of a specially designed static velocity pressure gage known as the Wahlen gage.

D. HARRINGTON,† Denver, Colo.—Physiologists, in general, say that small quantities of CO_2 are not particularly harmful and physiological effects are not easily ascertained. I disagree with that. When the temperature runs above 80° or 90° and the humidity is high, or, in other words, when the wet-bulb is high, small quantities of CO_2 are harmful and manifest themselves rapidly. The effectiveness of the movement of air was proved in an experiment carried out in South Africa. Some natives were put to work in a temperature of 88° in absolutely still air. A fan was started which merely moved the air without changing temperature or humidity and the men increased the amount of work nearly 45 per cent., though they did not know that they were doing so.

T. D. THOMAS,‡ Lansford, Pa. (written discussion).—While the ventilation of mines is becoming more difficult, ventilation methods have improved as the mines have been enlarged and sunk to greater depth. It seems but a short time since Pennsylvania passed a law requiring all mines to have a double-entry system; now mines have four and five parallel entries on the main haulage system and three parallel butt entries. After the installation of modern haulage systems, it was noted that the ventilation was considerably affected. When the locomotives were entering the mine, the air circulation was increased; and it was decreased by the outgoing trip, with the added danger that the air might be reversed. This condition was overcome by driving additional airways. The three-entry system on the butts has shown all the advantages claimed for it. The modern mine is developed on a retreating plan, that is, no breasts or ᴖhutes are turned off until the entry is up to the boundary, so the air

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† Supervising Mining Engineer, U. S. Bureau of Mines.
‡ Company Mine Inspector, Lehigh Coal and Navigation Co.

enters on the first and third entries and returns on the middle. It then enters the return airway, which carries the fumes and gases away from the men and assures a current of air not low in oxygen and containing but a small percentage of noxious gases.

In the steep pitching veins of the anthracite region, which are usually very gassy, improved methods have been adopted. For instance, in working the Mammoth vein, which has a thickness of 50 ft., the gangway is driven in the Skidmore. A return air course is driven in the Skidmore and cross-cuts are driven, not to exceed a distance of 60 ft. on the pitch of the strata, the pitch varying from 27° to 80°; on the opposite side of the gangway (Skidmore) chutes are driven through the rock between the Skidmore and the Mammoth veins, which intercept the coal immediately beneath the Mammoth. Slant chutes are driven parallel to the gangway, apexing every 60 ft., thus forming the third airway. The airways are about 25 ft. above and 30 ft. north and south of the main intake gangway and are connected by driving a rock chute at every fourth cross-cut. This forms an over cast and allows four breasts on a split on which eight miners are working.

Fan manufacturers, also, have improved their product. Not many years ago large fans were being installed, many of which had a volumetric capacity of only 65 to 80 per cent.; today fans having volumetric capacities of 137 to 160 per cent. are manufactured. Fans are now being built to meet the conditions, and their orifice conforms with the mine orifice. In some cases 16-ft. fans have been replaced with 8-ft. fans that deliver twice the quantity of air with the same power consumption.

At least 80 per cent. of the air should reach the last cross-cut; to insure this there must be the minimum amount of leakage at the brattices. The effect of defective stoppings is easily shown. Suppose that a fan delivers 200,000 cu. ft. of air but that only 40 per cent. reaches the working places; if the defective stoppings are made air-tight, and 80 per cent. of the air reaches the working places, only 100,000 cu. ft. of air will be required, reducing materially the power consumed.

Auxiliary fans, better known as booster fans, are probably misused more than any other unit about the mines. They are often placed in gangways where no provision has been made for an outlet of the air; there will be some movement of the air until the pressure in that section equals the pressure the fan is capable of producing, after which the air will only be churned. Auxiliary fans are very useful when the requirements of the district to be ventilated are known and a fan of proper dimensions is installed to meet the requirements. The disk-type have been installed as booster fans where the water gage was too high, and were useless therefore; a centrifugal fan of proper dimensions would have given satisfactory results.

In one case, after a 25-hp. conoidal fan had been installed, the supply

of air was increased from 940–1040 cu. ft. per min. to 11,280 cu. ft. at the last cross-cut and about 6000 cu. ft. was used for ventilating a few breasts. Sixty-three breasts were turned off this gangway, which was 3500 ft. long. Had this air been supplied by the main fan, it would have been necessary to reduce the size of regulators in the other gangways, thus increasing the water gage to 3.8 in., with a corresponding increase in power consumption.

J. J. WALSH (author's reply to discussion).—The statement "The volume of air flowing through a mine can be doubled, however, by the application of twice the horsepower if the ventilating units are independent of each other; that is by the installation of two fans, each having its own intake and return" is not intended to convey the impression that this is the most practical way to increase the volume of air in a coal mine; nor is it intended to infer that the layout of every mine will permit its air to be doubled by the installation of fans in that manner. Such deductions would be erroneous.

The most economical means by which the volume of air can be increased in a mine is by splitting. Were it not for the crowding of all the air through the main intake and return airways for short distances, the volume of air produced, when the power remains constant, would be always proportional to the number of splits. In other words, we would obtain double the quantity of air with the same power by doubling the number of splits.

Metal-mine Ventilation in the Southwest

By Chas. A. Mitke, Bisbee, Ariz.

In the Southwest, mechanical ventilation of metal mines has been receiving consideration for many years. The United Verde Copper Co., in Jerome, has used large mine fans for ventilation and fire-fighting purposes for at least 15 years, while the Copper Queen Branch of the Phelps Dodge Corpn. commenced installing mechanical ventilation in its various divisions 10 years ago.[1] Since then, with few exceptions, every large mine in this part of the country has installed and put in operation a mechanical ventilating system. The properties of some of these companies consist of from two to six divisions, or separate mines, each of which requires an independent ventilating system.

In other words, thirteen copper mining companies, with an aggregate tonnage, when operating at capacity, of approximately 50,000 tons a day, have installed mechanical ventilating systems in their mines. All these mines are within a radius of approximately 200 miles, which constitutes the largest copper-producing area in the world.

The capacities of mechanical ventilating installations in these mines range from 100,000 to 300,000 cu. ft. of air per minute, depending on the size of the mine, character of the workings, tonnage produced, etc. The latter varies from 300 to 20,000 tons per day. The mines include both high- and low-grade deposits, the greatest depth of the former, at present, being approximately 2500 ft.; and of the latter, 1200 ft. The volumes of air at present being coursed through the mines are from 60,000 up to 225,000 cu. ft. of air per minute, which, in their relation to the underground working force, range from 200 to 800 cu. ft. of air per man, per minute. While not to be taken as criteria, these figures are indicative of the character of the ventilation furnished.

Chas. A. Mitke: Ventilation of Copper Queen Mines, *Trans.* (1915) **52, 508.**

The operators in the Southwest have long realized that efficient ventilation goes hand in hand with low operating costs, principally through the large saving effected in the use of compressed air and the higher efficiency displayed by the men. High efficiency depends, to a large extent, on the providing of comfortable and cool workings, and the rapid removal of blasting smoke, gas, and fine dust, which ultimately results in the lessening of fatigue and a general improvement in the health of the workers. While many improvements have been effected along the lines of ventilation, much remains to be accomplished, such as the judicious application of additional water for laying dust, and the enlarging of volumes of intake air, in order to increase air movement in all working places, and thus intensify the ventilation for the more rapid removal of blasting smoke and fine dust.

VENTILATION IN GENERAL

Every large mine is ventilated by a primary and secondary ventilating system. By the *primary ventilating system* is meant the use of a limited number of large units, advantageously located, to ventilate as large an area of the mine as possible. This should approximate 70 to 100 per cent. of the mine workings. The air, in many cases, is drawn down through large concrete shafts, which constitute the main intakes, and are fireproof and have a smooth surface.

The *secondary ventilating system*, consisting of compressed-air jets, booster fans, and small portable blowers with ventilating pipe (some of which are mounted on trucks), is not brought into use until all the possibilities of the primary system have been entirely exhausted.

The most economical method is to ventilate as much of the mine as possible by the primary system; this requires careful coursing of the air and thorough distribution, taking the mine in sections, if possible, so as to ventilate each section with fresh air. In exceptional cases, where the mines consist of only one large orebody, or unit, and where, because of broken ground, it is not feasible to set arbitrary boundaries, the ventilation is adjusted according to the individual problem. Booster installations and small blowers are only used for raises, long drifts, and remote stoping sections. Only a comparatively small number of these auxiliary, or secondary, units have been found necessary.

In planning ventilating systems for mines in the Southwest, the principle of providing adequate air movement at all working faces has been closely followed. While of the utmost importance for efficient ventilation, this principle is not new, as the value of air movement has long been recognized in coal mines in Europe and America. The true test of the efficiency of a ventilating system is the percentage of working places (including faces) supplied with good air movement.

VENTILATION OF DEEP MINES CONTAINING HIGH-SULFIDE ORES

The usual problems connected with the ventilation of deep mines include the removal of heat produced by oxidation of timber, the regular heat gradient according to depth, acid waters, heat produced by electrical equipment, etc. In the southwest, there are a number of comparatively deep mines that contain orebodies having a sulfur content of from 10 to 45 per cent.; most of these are high-grade mines. At times the underground temperatures are augmented by heat produced by friction of moving ground and oxidation of the sulfide ore; the heat from these sources may be very much greater than that from all the other causes mentioned combined. In orebodies in which the percentage of sulfur is high and where continuous stoping operations are carried on, there is an increase in the quantity of heat, year by year; but at any time, should a cave take place, causing a large movement in the broken ore, an enormous amount of heat is rapidly produced, which may result in a mine fire.

When only a few sets are opened up in a sulfide stope and all the surrounding ground is intact, the stopes are usually just as cool as those in other orebodies, but the heat becomes very noticeable when stoping operations have been carried to a point where the ore surrounding the various stoping sections begins to move and break.

Ventilating systems for mines in which sulfide orebodies exist, especially if the sulfide is comparatively soft, oxidizes rapidly, and caves readily when stoped, should always have sufficient capacity to remove the additional heat generated from this source, also to take care of emergencies. By such provision, mine fires may be averted or speedily controlled. For example, at the Briggs mine of the Calumet & Arizona Mining Co., provision was made, about 5 years in advance, for a mechanical ventilating system capable of taking care of emergencies arising from causes similar to those mentioned above; when an emergency arose, it was speedily and efficiently handled by the organization without undue loss of time or a prolonged suspension of operations.

VENTILATION OF MINES WHERE CAVING SYSTEMS ARE EMPLOYED

During recent years, the introduction of the caving systems has brought new problems in ventilation. The success of such methods depends largely on the continuous breaking and blasting of the rock during the entire shift; the passing of the broken rock through drawing-off chutes, or grizzlies, into storage chutes; and continuous drawing off beneath, into motor cars. When it is considered that in such mines shots are fired as frequently as one every minute (and sometimes oftener), also that every shot pulverizes a certain amount of rock, producing a

large quantity of very fine dust, which rises simultaneously and moves with the powder smoke, the necessity for a most efficient ventilating system becomes evident.

In such mines, it is not so much the reduction of excessive temperatures, or high relative humidities, as it is the removal of the smoke and gas produced by blasting, and the fine dust created through continuous mining operations, that is of prime importance. Some of these mines operate three shifts, so that there is a continuous production of smoke, gas, and fine dust during the entire 24 hours. To prescribe rules for such mines, by limiting blasting to certain hours at the termination of the shift, would destroy the efficiency of the caving systems.

In most caving systems, because of undercutting, the ore breaks of its own weight and comes down through raises (incline or vertical) driven for the purpose, to the drawing-off, or grizzly, level. Part of this ore is fine enough to pass directly through the drawing-off chutes, or grizzlies, into storage chutes, but many boulders entirely too large to pass through also come down; these must be broken before more ore can be drawn. If blasting is not permitted until the end of the shift and this rush of boulders occurs early in the morning, operations must cease until the termination of the day's work, or the miner must take a hammer (for the grizzly) or a bar (for the "hung up" chute) and break the rock by hand. All this hand work is slow and expensive and would result in a decided reduction of the daily tonnage per man-shift. The moment a large boulder appears in the chute or on the grizzly, the most efficient method is to blast the rock at once and immediately start the ore moving again. No time is taken to drill the boulder, powder is simply laid upon it and the shot fired.

The mines in which caving systems are employed contain orebodies, with a very low copper content, that could not be worked at a profit by other methods; it is only by using the caving systems, in which the ore is kept continuously moving through the chutes, with large tonnage production, that these mines have made their well-known successes. The prohibiting of blasting, except at the termination of the shift, would hamper operations, thereby greatly reducing the efficiency of the system, by necessitating the return to hand methods, and make the extraction of this ore an unprofitable operation.

In order to remove all the fine dust, smoke, and gas within a few minutes after each blast, without at the same time interfering with mining operations, powerful, flexible, ventilating systems have been worked out and installed, and are being improved from time to time. In some places, the exhaust system has been used; in others, only the pressure system is practicable. With such systems it has been found, in one mine, that blasting smoke is dispersed within a few minutes after being formed; and with almost continuous blasting going on everywhere on

the grizzly level (approximately 2000 lb. of powder being used per day), the air in this mine was found to be clear practically all the time, except immediately after general blasting, which occurred twice a shift (noon and evening). At such times, smoke was present not over 10 min. at any one point.

In mines operated by the caving methods, the workings do not have the permanency that exists in higher grade mines worked by square-set, or cut-and-fill methods. The fact that the surface is constantly moving and that a certain number of mine openings are rapidly being caved and closed while others are being opened up, necessitates a most flexible system of ventilation. The utilizing of incline raises for ventilation, manways, and supply raises in stopes, as well as for the drawing of ore, provides a most economical and efficient method of ventilating the workings. While ore is being passed through certain raises, others remain open for ventilating purposes, and vice versa. As these incline raises are caved and closed by stoping operations, additional ones are driven in the development work, and by utilizing these the ventilation always keeps abreast of the stoping.

ALLAYING ROCK DUST

Necessity for Preventing Dust

The South African Miners' Phthisis Commission, in its general report, 1916, covering the subject of "silicosis," or "miners' phthisis," caused by the inhalation of minute particles of silica, and other insoluble dust, in mine air, stated that:

"Dust is generated in mines by the processes of drilling and blasting the rock, and in shoveling and transport of the broken ore.

"The finest dust is produced by blasting; over 99.5 per cent. of the particles are below 12 microns (1 micron = $\frac{1}{25,000}$ in.), and the average diameter of these is under 2.5 microns. The dust content of the air in a development drive after blasting may amount to over 500 mg. per cu. m., containing in that volume nearly 86,000 million particles of a diameter of 12 microns and downwards.

"Drilling by machine drills without the proper use of water also produces large quantities of dust. The dust content in the air of a development drive when drilling is performed dry may amount to from 100 to 200 mg. per cu. m. This is much less than the quantity formed by blasting, but dust from drilling is produced over a much longer period on each shift. Dust from drilling machines is rather less fine than blasting dust.

"Shoveling, transport, and loading of the broken rock not only stir up much dust which has already settled, but are themselves a source of dust production."

"The committee being desirous of taking immediate steps to cope with the disease (silicosis) was confronted with the difficulty that no standard of purity with regard to dust existed to which to work.   In order to assist in arriving at such a standard, tests of the amount and the character of the dust in the street air of Johannesburg were taken as affording some basis of comparison.   The question arose as to what weight per cubic meter (of air) could be considered permissible, and as this was more or less a matter of conjecture, the committee decided for the time being to adopt the tentative standard of 5 mg. of dust to the cubic meter of air.   This figure was at the time supposed to represent the average amount of siliceous dust under 70 microns diameter present in the air of a Johannesburg street."

Mine air was therefore sampled by the sugar-tube method in order to ascertain the weight of injurious dust present.

Investigations of medical experts, however, showed that the dangerous dust contained in mine air is composed of minute insoluble particles, and that those that injure the lungs, frequently resulting in silicosis, were seldom larger than 12 microns in diameter, and most of them less than 1 or 2 microns in size.

It was further stated that "In mining, it is not the coarse particles, but the fine impalpable dust which is the source of danger, and which, in consequence of the arduous nature of the work in an overheated atmosphere, is drawn by the miner into the recesses of the lungs, owing to the deeper breaths he is obliged to take."

According to the report, "the whole outlook on miners' phthisis (and incidentally on the analytical problems connected with it) was changed by the discovery of the almost infinitesimal minuteness of the silica particles retained by miners' lungs.   It became obvious that the ordinary gravimetric determinations of the total dust floating in the air must (whenever large particles are present) lead to quite false ideas as to the state of the air, since for example, a single particle of 100 microns ($\frac{1}{250}$ in.) diameter, weighs as much as 125,000 particles of 2 microns diameter.   It thus became necessary to exclude such larger particles from all samples undergoing gravimetric determination, although previous to 1912 these particles would have been considered to be very minute, invisible as they are to the naked eye.

"The first stage in the exclusion of superfluous particles was achieved by means of gauze screens, particles over 70 microns in diameter being prevented from getting into the sample by wire gauze of 200 meshes to the inch.   In more recent investigations, wire gauze of 260 mesh was used, excluding 'coarse' particles, viz., those over about 50 microns diameter.   Each such sample was thereafter separated by sedimentation into two portions containing the 'fine' and 'very fine' particles, respectively, with the dividing limit of about 12 microns, and each portion was

separately determined by weight." The committee considered the lighter portion, with particles under about 12 microns in diameter, as very fine or injurious dust, while the other portion, with diameters lying between 12 and 50 microns, was considered non-injurious.

"Concurrently with the exclusion of the larger particles, the total weight of residual 'very fine' dust became naturally very much smaller and it became necessary to invent more refined methods of sampling, so as to deal with very large volumes of air, and thus enable an accurately weighable amount of this 'very fine' dust to be caught. In this connection it should be noted that no less than two or three hundred million of these 'very fine' (say 2-micron) particles are required to be caught to give a milligram of dust by weight."

An important point that was brought out by the investigations was that "as a measure of the injurious character of any air breathed the number of particles of dust per cubic meter is a better criterion than its weight." Steps were therefore taken to devise a simple and direct method of determining the number of particles per cubic meter of air in samples without recourse to the comparatively cumbersome procedure of collection and determination formerly used.

Three years later, in its 1919 report, the Miners' Phthisis Committee recommended a tentative numerical standard of 300 particles per cubic centimeter of air, as determined by the "Konimeter," an instrument devised by Doctor Kotze, Government Mining Engineer of the Union of South Africa. In recommending this tentative standard, it stated that "A recent estimation of dust in the lungs of a person who had died of phthisis showed that they contain about 13 gm. of siliceous dust, consisting of 20 million millions particles. If a man at work breathes 1 cu. ft. of air per minute, containing 300 particles per cubic centimeter, and if they were assumed to be all siliceous, and if he works 300 days of 8 hr. each in the year, it would take him 16 years to inhale the 13 gm. mentioned. As the lungs are normally capable of dealing with some of the dust entering them, and as all the dust in the air does not reach the lungs, it is obvious that the period is much longer than 16 years. If, therefore, a standard of 300 particles per cubic centimeter be adopted as the upper limit of safe working, the probability is that, under ordinary circumstances, the conditions will be such as to approximate to an amount of dust represented by half this number." The committee reiterated the opinion expressed in its 1916 report that as a measure of the injurious character of any air breathed, the number of particles per cubic meter is a better criterion than its weight.

As a result of samples taken with both the Konimeter and the sugar tube, it was found that the number of particles per cubic centimeter of air corresponding to 5 mg. per cubic meter (the tentative standard put forward in the committee's report of 1916) varied from 85 to 700. It

was, however, believed that before the Konimeter should be permitted to supersede the sugar tube as a standard method of sampling dust in mine air, further experimentation with this instrument was necessary; the committee, therefore, advocated the use of the gravimetric method combined with a numerical count of the particles, as determined by the Konimeter, as affording an indication of the dusty nature of underground air.

DUST PREVENTION METHODS USED IN ARIZONA

In 1916, efforts were being made to solve the dust problem in both high- and low-grade mines in Arizona, it being realized that for such purposes a combination of ventilation and humidification was necessary, also that such ventilation must be far more intensive than the volumes required merely for the reduction of the ordinary mine temperatures and relative humidities. One of the problems in this connection was the temperature and relative humidity of surface air. In these copper-producing districts, the surface temperatures range from approximately 15° F. (in the winter) to 120° F. (in the summer), while the relative humidity ranges from approximately 35 per cent. (in winter) to 10 per cent. (in summer), with the exception of brief periods during the rainy season. A large volume of air, approximately 200,000 cu. ft. per min., would therefore have a very drying effect upon mine workings, thus facilitating the formation of dust.

The intake air, in general, is through large concrete shafts, therefore very little dust comes from this source. However, between the shafts and the stopes, there are frequently tunnels or drifts ranging from 300 to 4000 ft. in length, which become thoroughly dried, and the ventilating currents pick up the fine dust and carry it toward the stopes. To maintain the purity of the entering air currents and prevent them from taking up an excessive amount of dust, humidification is necessary. This is obtained, by adding moisture to the entering air, either by means of water trickling down the sides of the shafts or by the use of sprays and other devices, commencing at the intake and continuing at intervals throughout the workings.

In one mine shaft, the relative humidity of the air was raised from 25 per cent., as it entered at surface, to 85 per cent., on the 900-ft. level, through the use of water trickling down the shaft. In some cases, however, the use of water in downcast working shafts lowers the temperature of the entering air during the winter season to such a degree that it has an injurious effect on the men coming from the warmer stopes out into the haulage levels; in these cases other means of humidification must be employed. Sprays and atomizers at the tipples and placed at regular intervals in long drifts and tunnels have given good results. One objection to sprays in busy haulage tunnels is that, no matter how carefully the sprays may be installed, there is a slight leakage which, in the case of

a smooth tunnel floor, makes the track slippery and endangers men who are jumping on and off trains to throw switches, etc. There is a possibility that this can be overcome by the use of slag ballast—as is the practice in some mines—which should also assist in preventing dust, already settled, from rising again, but how much the slag ballast would slow up the movements of the men is as yet undetermined.

If general humidification of the mine air is employed, the relative humidity must approximate the saturation point to obtain results; this should keep the sides and floors of tunnels and drifts moist and prevent a large part of the dust that has already settled from rising. There remains, however, the dust created through drilling and blasting operations, which, if allowed to rise, floats in the mine air and, in most mines, never gets an opportunity to settle. The South African Miners' Phthisis Commission (1919) found that in perfectly still air, dust particles 5 microns ($\frac{1}{5000}$ in.) in diameter took 20 min. to fall through a height of 6 ft., and it took 8 hr. for particles of 1 micron ($\frac{1}{25000}$ in.) in diameter to be deposited. In many mines, the air in any part of the workings is never sufficiently still for all the dust to settle. An effort must, therefore, be made to lay this dust at its source, through the spraying of stopes and manways with hose and water pipe, the use of water drills, small sprays in blower pipe, and, possibly, by the use of water blasts in dead ends, immediately after blasting. However, even with the use of all this water, with the sides and floors moist, water sprays, and water machines for drilling, all the dust is not entirely deposited, and there is a definite limit to the amount of water that can be safely and comfortably used, unless intensive ventilation is combined with humidification.

The detrimental effects of high humidity, when not accompanied by a corresponding air velocity, on the health and efficiency of workers, is too well known to be discussed here; the higher the humidity is raised, the more intensive the ventilation must be.

In one large mine, approximately 1200 ft. deep, through which approximately 225,000 cu. ft. of air is being coursed, the relative humidity of the mine air has been raised to 93.5 per cent., while the temperature of the exhaust air is only 73.5°. It is believed that in this particular case the humidity of the general air might be raised to approximately 100 per cent., and that, with the present low temperature and a velocity of about 250 ft. per min. in at least 50 per cent. of the working places, in stopes the saturation of the mine air would have no detrimental effect on the men.

Water blasts in large stopes appear to be impracticable, because the surface exposed is too large for one blast to cover thoroughly, and a series of blasts would complicate matters. However, in very small square-set stopes and in dead ends, water blasts generally may be used to advantage. Their installation on grizzly levels, in mines worked by the caving sys-

tems, may also prove advantageous, provided only a limited amount of water is used; if too much water is employed, the excess will run down through the grizzlies, making the ore sticky and difficult to draw through the chutes. The number of water blasts that can be used underground is limited by the amount of compressed air that they consume.

Water drifters are in common use. Water stopers have been tried out with varying success. One obstacle to their general use is that the men prefer to take the risk of drilling dry, to getting their clothes wet when drilling upper holes with a water stoper. It must, however, be said that the water stoper cannot be quite so generally applied as the water drifter. For example, there are large shrinkage stopes in this region in which the regular flow of water through the water stoper frequently makes the broken ore too muddy and sticky for it to be drawn through the chutes beneath. In such cases, the compulsory use of the water stoper would be a hardship, and some other means of allaying the dust must be adopted. More intensive ventilation, in such cases, will tend to improve conditions.

In this connection, there is a field for investigation as to the type of water machine that produces the least dust. While water machines have a beneficial effect on the dust created through drilling operations, the passage of the water through the steel does not completely lay all the dust. Investigators believe that some types of water machines produce much finer dust than others; the finer dust, of course, is more difficult to lay. An exhaustive study and comparison of the character and quantity of dust produced by the operation of various types of water machines would prove most helpful in solving the dust problem.

In a number of the mines previously referred to, not only is silica closely associated with the ore, but sufficient sulfur is also present to show a color effect on the miner's carbide light. The particles of copper and iron sulfide are insoluble in water and may be just as dangerous as silica dust, although no comparative study has been made regarding the relative harmfulness of silica and sulfide dust particles.

Determining Dust Content of Air

In making a study of the dust content in mine air in this region, various instruments have been used. Owing to war conditions, a Konimeter was unobtainable at first; so an instrument of somewhat similar construction, manufactured in this country, was used for sampling the mine air, and photomicrographs were made of the dust spots on slides. The chief objection to this instrument was that, being designed to sample a very large quantity of air (5 cu. in.), the dense deposit of dust thrown on the small cover glass made an accurate count of the dangerous dust particles under the microscope almost impossible. Later, a Konimeter was obtained and is now used in making dust determinations. Samples

have also been taken by the gravimetric method. Briefly, the procedure adopted in taking dust samples by the two methods is as follows:

In the *Konimetric method*, a known quantity of dusty air (5 c.c.) is made to impinge, at high velocity through a narrow nozzle, against a glass plate coated with an adhesive mixture (vaseline or some similar substance). The minute dust particles remain embedded in the adhesive mixture. The glass is then removed from the instrument, placed under a microscope containing a ruled ocular, and the dust particles counted at magnifications of 150–180 and higher. The resulting count is the number of particles present in 5 c.c. of air. An attempt is also made to determine the size of the particles by the use of ruled lines on the ocular, the distance between which represent a diameter of 5 microns. This instrument has been freely used in South Africa, and Australia.

The *gravimetric*, or *sugar-tube method*, as it is frequently called, consists in drawing a measured volume of dusty air through a tube containing granulated sugar, which filters out the suspended dust. In the laboratory, the sugar is dissolved in distilled water, to eliminate the soluble particles; the solution is filtered through a special filter paper, ignited, and weighed (proper corrections being made for dust in the sugar prior to taking the sample, and for filter ash). A portion of the solution containing the suspended dust is made up to a measured volume and an aliquot part (1 cu. mm.) placed under a microscope and the dust particles counted.[2] By this method, a weight of the insoluble, or dangerous, dust is obtained, and through sedimentation, under water, a numerical count of the particles.

While of the utmost value in assisting operators to obtain an indication of the dusty character of their mine air, neither form of sampling is perfect. The main objections to the Konimeter appear to be the diversity in counts of the same samples by different experts, the difficulty in obtaining a full count in one operation with the apparatus, and the pitting of the vaseline-coated cover glass with water vapor, causing pitmarks that may be mistaken for dust particles. The South African Chamber of Mines (Standing Committee on Dust Sampling, Nov., 1920) stated that "The Konimeter sample is small, instantaneous, and local; it is difficult to be sure that one is counting dangerous dust, and the personal factor in the counting renders results of doubtful value."

The collecting and counting of dust particles under water, as is done with the gravimetric method, also has objections, and authorities vary as to its accuracy. In a recent publication regarding aerological instruments, Dr. Vernon Hill, of the Chicago Department of Public Health, states: "In dust-counting devices that collect the sample under water,

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[2] Fieldner, Kats and Longfellow: The Sugar Tube Method of Determining Rock Dust in Air. U. S. Bureau of Mines, *Tech. Paper* 278.

the particles are broken up in the water and the final count is often many times that of the original count of the air sample; also the shape and general characteristics are lost by the water collection method."

Doctor Hill is a student of heating and ventilating problems and as such is an authority. For this reason, any statement made by him, such as the above, cannot be ignored. Moreover, when we consider that granite is largely composed of silica, and that water and weathering have a powerfully disintegrating effect on the rock components, there is food for thought in Doctor Hill's statements.

In its 1919 report, the Miners' Phthisis Prevention Committee stated that "Tests with the dust caught by the sugar tube indicate that the number of particles recovered is generally smaller with that method than with the Konimeter." This, may possibly be accounted for by the fact that the Konimeter collects both soluble andi nsoluble particles, while a count of dust particles obtained by sedimentation, after collection in the sugar tube, includes only insoluble particles.

Another apparent objection to the gravimetric method is that by this process, after the collection of the dust sample in a sugar tube, the getting of the sugar into solution, and the separation of the insoluble dust by infiltration, the filter paper containing the insoluble dust particles is incinerated, prior to weighing. In the case of recent investigations at the Broken Hill mines in Australia, it was found that ignition of the dust caused elimination of the sulfur content of the ores.[3] In sampling the dust in mines containing a large percentage of sulfur, such as many mines in this region, this would create a serious error in the weight, as while it is possible to eliminate sulfur by burning the sample in the muffle, these particles are insoluble in water and undoubtedly are dangerous elements to be inhaled in the lungs.

In experiments in both South Africa and Australia it was found that oil atomized by drilling operations and the tarry product of the fuse when blasting collected in the sugar tube and formed aggregates of the dust.

The South African Miners' Phthisis Committee (1919) found that "owing to the varying fineness of the dust, the expression of its amount in weight does not always enable a true comparison between one set of conditions and another, and, in the case of very fine dust, the method underestimates the danger."

Another consideration that has considerable bearing on the correct estimation of the dangerous dust in mine air is the specific gravity of the material stoped in different mines, which varies considerably, that at Broken Hill (Australia)[4] being approximately 1.5 to 2 times that of the South African material.

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[3] P. H. Warren: Mechanical Methods of Allaying Dust. Australasian Inst. Min. and Met. (June, 1921).

[4] *Op. cit.*

So long as medical authorities believe that it is the minute particles of insoluble dust that constitute the dangerous constituents in mine air, an instrument which permits a count of these particles, retaining their individual characteristics, such as the Konimeter and instruments of this type attempt to give, is preferable to the gravimetric method, or counts of samples collected under water by sedimentation. Moreover, for general mine use, the Konimeter type of instrument is more practical and more easily handled, and the laboratory work is simpler than that required for gravimetric tests.

Several instruments, using the principle of the Konimeter, are being developed and perfected, both in this country and in England, and before long we may have an instrument the accuracy of which will satisfy most observers. However, until there is designed an instrument that overcomes the objections raised to those in present use, no definite dust standards can be set.

The Kata-thermometer

The kata-thermometer is also used in taking readings in these mines, in order to measure the efficiency of the ventilation. This instrument is an invention of Dr. Leonard Hill of London, the eminent physiologist. As a result of his researches, Doctor Hill found that:

"The quality of freshness of air is due to its cooling power on the body and that, on the contrary, oppressiveness of ordinary atmospheres is due, not to deficiency of oxygen, or excess of carbon dioxide and organic toxins, but to heat stagnation.

"Cooling power depends on temperature, humidity, and motion of the air, and is increased by low temperature, low humidity, and high velocity of motion." He concluded that the rate of cooling of the human body was the controlling factor in determining comfort or discomfort, and the kata-thermometer registers the cooling and evaporative powers exerted by the air on a surface at body temperature.

Briefly, this instrument consists of an alcohol thermometer with a large cylindrical bulb about 4 cm. long and 1.7 cm. in diameter, with a stem about 20 cm. long, graduated from 95° to 100° F. This range of 95° to 100° F. is arbitrarily chosen to give a mean temperature of 97.5° F. (36.5°C.), which is approximately that of the human body when clothed. A small bulb at the top of the stem furnishes a margin of overheating without bursting the bulb, to give time to place the instrument in the position required, and to allow the rate of cooling to become steady before the meniscus has fallen to 100°. Dry and wet kata readings are taken, the one thermometer being used for both.

. The dry kata registers the heat loss by radiation and convection, while the wet kata gives a measure of the combined effect of the temperature, humidity, and velocity of the air acting on a wet surface at body tem-

perature, and indicates the heat loss by radiation, convection, and evaporation. "The factors which affect the heat loss from the wet and dry kata are identical to the factors which affect the heat loss from the body, so they also affect the comfort of the body."[5]

In taking a dry reading of cooling power, the bulb is heated in hot water (about 80° C.) until the alcohol (colored red for visibility) expands and all air bubbles have been driven to the top of the meniscus (a thermos flask being generally used for this purpose). The instrument is then thoroughly dried and suspended in position. In taking a wet kata reading, the same procedure is followed, with the exception that a thin cotton sleeve is drawn over the bulb of the instrument before it is immersed in the hot water.

The time, in seconds, required for the liquid to fall from 100° to 95° is recorded with a stop watch. Each instrument is carefully calibrated and marked on the back of the stem with a factor or coefficient, such that the factor divided by the time of cooling, in seconds, gives the millicalories per square centimeter of cooling surface per second. For example, in the case of a kata-thermometer with a factor of 488 that takes 100 sec. to cool from 100° to 95° F., the cooling power of the surrounding atmosphere is 488 divided by 100, or 4.8 millicalories per square centimeter per second.

By means of formulas, the readings may be translated into terms of temperature, relative humidity, and velocity, and vice versa.

Doctor Hill states "if the wet kata cooling power be kept high enough in mines, the managers will be sure that men will be able to lose heat by sweating adequately and work without having to take frequent periods of rest for cooling off. If the dry kata cooling power be kept high enough, men can work without sweating or the acceleration of the heat this entails, and in comfortable invigorating conditions."

"Muscular activity is accompanied by the oxidation of glycogen, a chemical process producing heat, a process continually going on to a greater or less degree while life lasts. This heat production varies with the degree of muscular activity, and is least when the body is recumbent, relaxed, and asleep. This heat producing activity, when the body is resting, Doctor Hill calls the "basal or minimal metabolism," and its amount is an important factor in a man's capability to perform work in a hot atmosphere. Heat production is considerably increased when external work is done. The basal metabolism of the average man corresponds to a heat loss of about 1 millicalorie per square centimeter of body surface per second; work may bring it up to 3 millicalories.

"The heat generated by muscular activity must be dissipated in some

[5] O. W. Armspach and Margaret Ingels: Temperature, Humidity and Air Motion Effects in Ventilation, Jn. Am. Soc. of Heat-Vent. Eng. (1922).

way or the temperature of the body will rise and cause serious distur-
bances. The heat is dissipated from the body surface by radiation,
convection, and evaporation."[6]

In order that the body may dispose of its heat without strain, the
cooling power of the surrounding air should be adequate; Dr. Leonard
Hill has recommended the following cooling powers for persons doing
different classes of work, the units being in millicalories per square
centimeter per second:

1. For sedentary workers, 6 by dry kata, 18 by wet.
2. For light manual work, 8 by dry kata, 25 by wet.
3. For heavy manual work, 10 by dry kata, 30 by wet.

Dr. A. J. Orenstein and H. J. Ireland recently conducted a most
interesting study in the Rand mines in South Africa. The instruments
used were ergometers (to register the output of work under various
atmospheric conditions, an effort being made to simulate with these
machines the type of work that causes the greatest fatigue, such as hand
drilling), kata-thermometers (to register the cooling power of the air),
and dry- and wet-bulb thermometer readings. Skin temperatures and
loss of weight of workers were recorded and general observations made as
to the effect produced by the conditions in the various parts of the mine.

These investigators believe the wet kata gives the best standard
of comfort for ventilating purposes in mines, as it includes all the factors
in cooling power on the body—temperature, humidity, and motion.
However, they state that "neglecting the surface readings, there is a
close correspondence in the curves drawn by the wet and dry cooling
power values, the corresponding wet and dry readings of underground
conditions giving practically the same output of work; for instance, a dry
cooling power of 4 gives 480,000 ft.-lb. output; the corresponding wet
power is 11, which gives approximately the same output, so that in esti-
mating working efficiency it does not matter whether we take the dry or
wet cooling power curves."

Table 1, compiled by these investigators, gives an idea of the cooling
power by the wet kata with corresponding physiological effects likely to
be produced. The dry readings are the corresponding values that will
probably occur in underground conditions. These are based on numer-
ous observations taken in the Rand mines. When the dry cooling power
exceeds 8 and 10, sweating ceases, and the dry cooling power should then
be taken as the standard of cooling, if the skin and clothes are dry."

In forming an estimate of the general loss of output incurred in a mine,
because of inadequate cooling power, a representative section of a mine

* [6] Orenstein and Ireland: Experimental Observations upon Relation Between
Atmospheric Conditions and Production of Fatigue in Mine Laborers. *Jnl.*, S. A.
Inst. of Eng., (March, 1921).

TABLE 1.—*Values of Cooling Power by Wet and Dry Kata with Corresponding Physiological Effects*

Cooling Power		Physiological Effects on Man Stripped to Waist
Wet Kata	Dry Kata	
5	1.5	Extremely oppressive; profuse sweating, rise of body temperature and pulse rate especially when work is performed; very small evaporation from respiratory tract.
10	3.5	Distinctly oppressive; body temperature can be kept nearly normal only by profuse sweating; skin flushed and wet; pulse rate high.
15	5.5	Lower limit for comfort.
20	8.0	Quite comfortable for work.
25	10.0	Cool and refreshing for work; too cold for resting, especially after being heated.

was chosen and kata-thermometer and wet- and dry-bulb temperature readings taken in sixty-three positions, in thirty-six of which men were working. The cooling powers, estimated working efficiency, number of men, and equivalent number at full efficiency were tabulated and the overall efficiency determined by dividing the sum of equivalents by the sum of actual men working.

From these investigations, Doctor Orenstein and Mr. Ireland estimated that 21 per cent. of output was lost through inadequate cooling power. "It is not implied that all this loss is avoidable as it may be impossible to bring the cooling power up to the arbitrary standard, but it shows how important it is that adequate cooling power be maintained if at all possible. Moreover, this loss is loss of output only, and does not include the losses due to the increased morbidity which must surely follow the excessive strain and fatigue associated with work in oppressive atmospheres; such losses would be hospital charges and cost of replacing sick men." Moreover, as the investigators state, it must be borne in mind that the above mentioned tests were conducted under constant supervision and observation and frequent periodical readings of output and attention to machine taken, which naturally acted as a constant spur to the subjects of the experiments in keeping them up to working capacity. In the absence of such supervision, it is doubtful whether they would keep up the "best effort" when fatigued.

These investigators found the relation between efficiency and dry cooling power to be as follows:

Dry cooling power....	....	1	2	3	4	5	6
Working efficiency, per cent..........		50	60	70	80	90	100

They believed that the working efficiency would increase with cooling power up to 6 dry kata, but not to make the conditions too rigorous assumed that working efficiency is 100 per cent. at dry cooling power of 6. They state "the curve giving the relation between cooling power and output will rise more rapidly at the low cooling power and less rapidly as it approaches 8 dry cooling power, making a curve convex upwards."

An interesting point brought out by their experiments was the effect in oppressive conditions of stirring up air by means of a fan. After a series of tests had been conducted in a certain drive, a fan was erected so as to blow a current of air over one of the subjects of the test, located 10 ft. from it, and then over a second subject, situated 7 ft. farther away. The air volume was estimated at 2,000 cu. ft.

On the first day the fan was operated, a rest period was taken as in the preceding series (of tests), the kata readings being taken midway between the two employees. The output, when compared with the average output of the preceding series, showed an increase of 46 per cent.

In concluding their observations, they state that:

"When the cooling power of the atmosphere is below 6 units by dry kata and, let us say, 16 by wet kata, the working efficiency falls. In bad places, where the cooling power is only 1.5 dry and 5 wet, or under, the average efficiency is only about 55 per cent., the body temperature rises to an undesirable degree and extreme fatigue may be produced by work.

"Working in hot places is a greater physical strain on the worker in winter than in summer.

"The effect of a short rest at half time is not sufficiently pronounced to make it worth while.

"General kata readings in several mines show that workers under present conditions may be subjected to extremes of cooling power which may severely overtax their defensive mechanism.

"A small fan with a capacity of about 2000 cu. ft. per min. operated about 10 to 20 ft. from the worker may increase the output (under experimental conditions) by as much as 46 per cent."

Unfortunately, in its present form, the "kata" is not practical for general underground use, as a bump or a jolt will easily break it. Were it possible, without affecting the radiant heat loss from the instrument, to mount it in a manner similar to the sling psychrometer, its use would be greatly facilitated.

STANDARD ATMOSPHERE

For some years, a tentative standard, based on readings and observations taken over a long period, and providing for a good working atmos-

phere has been sought in the Southwest. Such a standard consisted originally of a temperature of 77° F., rel. hum. under 85 per cent., a velocity of at least 100 ft. per min., and volume of at least 250 cu. ft. per man per min.[7] This was later changed to a temperature of 78°, rel. hum. of 80 per cent., velocity of 125 ft. per min., and volume of 350 cu. ft. of air per man per min.,[8] which translated into terms of dry and wet kata readings, would approximate 5.7 dry and 14.8 wet, it being understood that these figures of cooling power refer to millicalories per square centimeter per second, according to formulas worked out by Dr. Leonard Hill. While these rates of cooling are somewhat low, compared to the latest standards set by Doctor Hill for the dry and wet kata readings, it must be remembered that this standard atmosphere was promulgated about 9 years ago, before the kata was well known.

It is interesting, however, to note that in their investigations upon the relation between the atmospheric conditions and the production of fatigue on mine laborers, in South Africa, Doctor Orenstein and Mr. Ireland found that with a dry cooling power of 6 they obtained approximately 100 per cent. efficiency. While the standard atmosphere mentioned above has not been attained generally in dead ends and stopes in the average mine, it has been more than surpassed in the main drifts, raises, and haulage levels, and in some stopes.

In one mine in which a ventilating system was installed, and the above standard attained in a large number of the working places, the, then, superintendent stated that "due to the installation of forced ventilation the product per man-shift in stoping has practically doubled."[9]

This standard was evolved at a time when most of the ore was stoped by the square-set and its various modifications, cut-and-fill, and other timbered and semitimbered methods. These systems of mining necessitated very little blasting during the shift, and such a combination as that mentioned appeared satisfactorily to meet conditions in stopes of this character. However, as the workings became deeper and the temperature and relative humidity increased, the velocity and volume of the air had to be increased. Later, when the caving methods were developed, in which the basic principle is frequent blasting at any time during the shift, other changes had to be made in the standard, as it was not merely a question of providing a certain volume of air per man per minute to cool the working place sufficiently to enable the men to perform their tasks efficiently, but the working out of the most effective method of immediately eliminating the powder smoke and dust created through blasting operations. When blasting occurs, in addition to the powder smoke and gas, the force of the explosive pulverizes some of the rock and projects

[7] Chas. A. Mitke: Ventilation of Copper Queen Mine. *Trans.* (1915) **52**, 508.
[8] Chas. A. Mitke: Standardization of Mining Methods. *Eng. & Min. Jnl.* (1919).
[9] *Trans.* (1915) **52**, 525.

large volumes of very fine dust into the air. Should the air have little or no velocity, the clouds of dust naturally mill around and remain in the vicinity of the working place where the blast occurred. This necessitates the use of more intensive ventilation than that generally used in high-grade mines.

Recent investigations have shown the importance of determining the effect of the cooling power of the ventilating currents on the workers, also the necessity of purifying the air currents of dust, which in many instances requires the raising of the relative humidity with a corresponding increase in the velocity of the air currents.

Many other factors also have a bearing on the selection of standards for ventilation. For instance, a standard consisting of a certain temperature, relative humidity, and velocity, that would give a satisfactory rate of cooling in one mine might, for other reasons, be most unsatisfactory in a mine where the mining method is different.

Standards for ventilation must, therefore, be based on practical, scientific, and physiological considerations. Personal opinion, which often enters into the determination of the condition of a working place, is not a scientific solution of the question. What may seem "comfortable" to the observer, may not be comfortable to the worker. On the other hand, the worker, who is accustomed to his surroundings, may find the atmosphere perfectly comfortable while to the observer it may seem just the opposite. To arrive at true standards, it is not only necessary to take the simple readings shown by the hygrometer and anemometer, but to determine the effect of the air currents on the man physically and the amount of fatigue he registers in the performance of his work, under varying atmospheric conditions.

While the scientific solution of the problems connected with mine ventilation appears to be making rapid progress, many points remain unsettled. Dr. J. S. Haldane of Oxford, one of the pioneers in connection with the physiology of respiration, would appear to prefer the wet bulb as a guide in the matter of a good working atmosphere; J. L. Bruce, an Australian investigator[10] prefers the dewpoint rather than the wet bulb, fixing 62 as the permissible maximum dewpoint. Doctor Orenstein and Mr. Ireland, while acknowledging the value of the kata-thermometer, state that it gives no reliable indication of the evaporation from the respiratory tract, this being dependent "not on the motion of the air, but on the amount of air breathed and its absolute moisture content, which latter, can, of course, be found from wet- and dry-bulb readings." According to Dr. Leonard Hill, "A considerable evaporation from the respiratory tract produces a copious flow of fluids from the lymphatic glands to the mucous membrane, and also stimulates increased blood flow to make up the heat loss, both of which processes promote

[10] Inst. Min. and Met., London: *Trans.* (1922).

immunity from bacterial invasion." Mr. Ireland states, "There is little doubt that the very humid atmosphere of the mines tends to lower the immunity for the time being from respiratory diseases and this is aggravated by the strain of working in places where the cooling power of the air is utterly inadequate."[11]   Mr. Ireland raises the question as to whether evaporation from the respiratory tract has any influence on silicosis, and whether a large evaporation would tend to prevent the dust block in the lymph glands.   In this connection, he suggests that a good margin to work with would be a dewpoint of 60° F. (approximately), basing this conclusion on the fact that "In dealing with humidity, it should be borne in mind that so far as the human organism is concerned, it is absolute humidity—actual moisture content—not relative humidity, which matters.   Air at 40° F. though saturated (100 per cent. rel. hum.) will contain much less moisture than air at 80° F. with a relative humidity of 50 per cent.   On a day on the Rand, with a temperature of 45° F. and a rel. humidity of 20 per cent., the evaporation from the respiratory tract per 100 cu. ft. of air breathed is approximately 0.24 lb.; in a good place underground, with a temperature of 65° F., a relative humidity of 90 per cent., the corresponding evaporation is 0.16 lb.; and in a place 87° F. and 95 per cent. humidity, 0.063 lb. or a little over a quarter of the first.   Here, again, he states, "there should be something to aim at, especially as the dry atmosphere of the Rand gives a good margin to work with, and I suggest 0.16 lb. per 100 cu. ft., i.e., a dewpoint of 60° F. (approximately)."

In concluding their investigations into the effect of atmospheric conditions on the production of fatigue on mine laborers, Doctor Orenstein and Mr. Ireland state: "There is a body of opinion that, besides the factors of temperature, humidity and motion, there may be more subtle factors on which at present we have not sufficient information.   For instance, G. Stroede[12] working on the catalysts assisting the transference of oxygen finds a good deal of evidence that such catalysts are to some extent inhibited by exhaled air.   The stuffiness of air caused by its passing over steam radiators, though the final temperature inhaled may be sufficiently low and the cooling power satisfactory, is another matter that requires elucidation.   There is also the question of air passing over decaying organic matter, such as rotten timber, when considerable oxidation goes on and the air feels stuffy.   Whether the stuffiness in the latter case is detrimental to health, or whether, like some bad smells, such as those produced in soap works and glue works, it is objectionable but innocuous, we cannot, at this stage say definitely."

Until, therefore, further physiological research is conducted, and

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[11] Chem. Met. and Min. Soc., Jnl. (August, 1921).
[12] G. Stroede: Ueber den Nachweis organischer Lähmungsstoffe in verbrauchter Luft. Ztschr. f. Schulgesundheitspflege (1917) 30, No. 1.

additional experiments performed on the men during the process of their underground labors, and numerous readings taken of atmospheric conditions during those periods, no absolute standards for atmospheric conditions can be advanced. In the meanwhile, the kata-thermometer, together with the other instruments in present use, will do much toward assisting the improvement of underground conditions. In mentioning the advantages of the kata-thermometer over the wet bulb in giving an indication of atmospheric conditions underground, Dr. Leonard Hill, referring to the investigations of Doctor Orenstein and Mr. Ireland in mines on the Rand, stated, "The fact that Doctor Orenstein and Mr. Ireland have demonstrated 50 per cent. less efficiency of native workers in the places where the kata reading is the worst and a loss of 21 per cent. for the whole mine below what it might be if my dry standard kata reading of 6 per cent. were adhered to, suffices as proof of this claim."[13]

Doctor Hill advocated a dry kata reading of 6 for sedentary workers, and cooling powers, by the dry kata, as high as 8 and 10 for manual work; however, to obtain such cooling powers underground is much more difficult than in buildings and factories on surface. To attain such figures, it is necessary either to reduce the temperatures or greatly increase the velocities. In general, the average stope temperatures range from 75° F. up, and to reduce these much below 75° F. would be prohibitive from an economic standpoint. On the other hand, to increase the velocities sufficiently to obtain dry kata readings of 8 and 10 is not always possible, particularly in old mines with limited intakes and outlets, as there is danger in greatly increasing the speed of the entering air in main intake shafts, which are also working shafts, to say nothing about the additional expense for power, which is quite an item at the present time. As an illustration, in an underground working place with a temperature of 77° F. in order to obtain a reading of 6 with the dry kata, it is necessary to have a velocity approximating 125 ft. per min.; to raise this cooling power to 8 would necessitate increasing the velocity to 275 ft. per min., approximately. Were the temperature higher, say 86° F., to obtain a cooling power of 6 would require a velocity of 550 ft. per min.; and to raise this cooling power to 8 would necessitate increasing the velocity to over 1000 ft. per minute.

From recent investigations, it would appear that a good tentative standard would be a temperature of 77° to 78° F., with a relative humidity of 80 per cent. (wet bulb approximately 73), and a velocity of 125 to 150 ft. per min. This should apply principally to stopes and dead ends and not be the average for the entire mine. Frequently, averages of temperatures and relative humidities in working places in an entire mine include many readings taken in main shafts, main haulageways, and

[13] Inst. of Min. and Metal., London (1922).

main intake air courses, which, of course, greatly reduces the general average of the mine. In many instances, such mines contain many hot stopes in which men cannot do a fair day's work. An average, therefore, unless accompanied by details of readings and locations, is often misleading. In most cases it will necessitate considerable work and careful coursing of the air to approximate the above standard in stopes and dead ends, but were this tentative standard achieved, the general ventilation throughout the mines would be greatly improved. This combination closely approximates a cooling power of 6 by the dry kata and 15 by the wet, and, as has been shown in the investigations of Doctor Orenstein and Mr. Ireland, represents the lower limit for comfort in underground workings.

It has also been demonstrated that in mines where this standard has been attained, the installation has more than paid for itself in increased efficiency of the men (even though the cost of power is 3 c. per kw.-hr.). In one instance, the tonnage per man-shift was increased from $2\frac{1}{2}$ to 7 tons, or approximately 180 per cent.[14]

In most underground working places, it should be possible to attain the above standard by means of the primary ventilating system. In dead ends and remote stoping sections, the more general use of a secondary ventilating system, composed of small fans and booster equipment, will assist in attaining the above tentative standard.

After the standard of 6 dry kata and 15 wet kata has been obtained in nearly all working places in most mines, steps might be taken to raise the standard to approximate Dr. Leonard Hill's standards for manual labor of 8 and 10 by the dry kata, and 25 and 30 by the wet kata.

RELATION OF MINING METHOD TO VENTILATING SYSTEM

An important consideration, which is seldom realized, is the close relationship between the mining method for extracting the ore and the system of ventilation. Having worked out and supervised the installation of ventilating systems for sixteen large metal mines, it has been the writer's experience that the stoping method should first be thoroughly understood before an efficient system of ventilation can be planned. By first carefully studying the mining methods, changes necessary for improvement in the ventilation may also increase the efficiency of the mining method.

For example, in mines worked by square-set methods, where the custom had been to take large sections and use convenient areas, etc., merely for working purposes, it was found that better ventilation and greater efficiency in stoping could be obtained with the use of smaller

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[14] Chas. A. Mitke: Standardization of Mining Methods. *Eng. & Min. Jnl.* (1919) 41.

sections. These smaller sections required more raises, which gave greater facilities for the handling of ore and freer passage for the air, by increasing the number of openings. Later, standard-size raises, standard-size manways and timber compartments were designed, in order to furnish constant areas, which were made as large as practicable. This innovation proved to be most valuable from an economical and efficient as well as ventilating standpoint.

There has been a tendency to install too many doors, stoppings, etc., which increase resistance and lessen the volume of air. A review of the mining method will frequently disclose the fact that considerable development work, done in advance of the stoping, will increase the number of air passages and at the same time decrease the resistance, thus greatly facilitating the flow of air. Instead of adopting the policy of closing off too much of the mine and throttling the ventilation, it will frequently be found that it pays to open up the new stoping area well in advance.

Use of Secondary Ventilating System for Fighting Mine Fires

The use of mechanical ventilation for fighting mine fires has been a common practice in the Southwest, where there have been over thirty-five active mine fires during very recent years.[15] In general, the principle followed has been to close the outlets from the fire district and then put the fire under pressure. This is always accomplished by the use of small independent fans, generally mounted on trucks, and ranging in capacity from 500 up to 30,000 cu. ft. of air per minute. Even should the fire occur in a part of the mine which is already under pressure from the main ventilating system, the portable fan can be carried much closer to the fire, and a direct pressure put on the heated area, in addition to that exerted by the general ventilating system. With small portable blowers, an attack may also be made on the fire through a number of drifts leading to the fire area. Moreover, as the fire is either extinguished or bulkheaded off, the blowers may be shifted to other locations. Portable blowers for fire fighting have been successfully used at mine fires, under the direction of the writer, at the mines of the Cananea Consolidated Copper Co., in the Bisbee district, and at Morenci and Coronado, during the past 8 years.

Formerly, all fire fighting was done by a large number of helmetmen trained for the purpose, but men wearing oxygen apparatus cannot perform nearly as much work as men working in fresh air, without such encumbrances. Experience at various mine fires, with the use of portable blowers, has demonstrated that it is possible to fight mine fires without the use of helmetmen. While this is not advocated as a general rule, in

15 Chas. A. Mitke: History of Mine Fires in the Southwest. *Min. & Sci. Pr.* (August, 1920).

exceptional cases it has been successfully accomplished. At numerous mine fires in this region, all the bulkheading, emergency timber work, putting up of brattices, laying of water lines, etc., has been done by men without oxygen apparatus, and working in good air furnished by portable blowers. A number of helmets are, of course, kept near at hand, but they are used merely for exploratory work. The old type of concrete bulkhead (from 6 in. to several feet in thickness) is seldom used. The general bulkhead used in this region is made of two stulls, with 2-in. boards nailed across, so as to close off the drift or raise. Then a wire netting is nailed on the boards and the whole bulkhead covered with from $\frac{3}{8}$ to $\frac{3}{4}$ in. of cement. These bulkheads have proved satisfactory, economical, and efficient and may be constructed within an hour by men without helmets.

INSTALLATION OF EQUIPMENT PERMITTING REVERSAL OF AIR CURRENTS SHOULD DEPEND ON CONDITIONS AT MINES

The prevailing impression is that so-called reversible fans should be installed everywhere, regardless of underground conditions, the general belief being that in many cases it is absolutely necessary to reverse the ventilating currents in the case of a mine fire.   There are some instances where intake shafts are timbered, and where, in the event of a fire in the shaft, it might be advantageous to reverse the ventilating currents, provided this could be accomplished in time.   However, as there is a general movement toward the adoption of concrete for fireproofing main working shafts (which are usually main intakes), the danger from fire in shafts is greatly reduced.

In only two of the thirty-five mine fires that have occurred in the Southwest during the past 10 years, did the question of reversing the air currents arise; in all the other cases it was so obviously not the thing to do that the matter was not even discussed.   Even in the two cases mentioned, it was decided not to reverse the fan.

There may be cases where an outbreak of fire occurring at certain points in the mine could be advantageously handled by reversing the air currents, provided this could be done in time.   However, it cannot be said that the installation of so-called reversible fans applies equally to all mines, and the installation of such a fan in a mine in which it cannot be used advantageously constitutes a real danger, as in times of excitement, such as a mine fire, some one may hastily order the reversal of the air currents and endanger the lives of everyone in the mine.   Fires are fought more easily with small portable blowers.   The reversal of many ventilating systems would turn main travelingways into temporary upcasts, which would greatly hamper the fire fighting, and little efficient work can be done by sending men through shafts that are airways only, to say nothing of the difficulty of handling equipment and supplies.

Aside from this, the present mine needs require swift air currents on main levels. Supposing that an air current travels at the rate of 1000 ft. a min.; even if the fan could be reversed within 10 min., this brief period would allow the smoke and gas to penetrate all the workings and kill the men long before the ventilating currents could be reversed A far safer method is to arrange for the instantaneous closing of fire doors throughout the mine, to check the rapid air currents within a minute, thus controlling the ventilation and avoiding disaster.

In addition, in some mines a so-called reversible fan cannot be used with safety; for example, mining out the small orebody at the United Verde Extension under mine fire conditions, during 1917–1918.[16] Had the ventilating currents been reversed, it would have been impossible to attack the fire stope. However, by closing off this area from the general ventilating currents and placing it under pressure by the use of portable blowers, it was possible to mine out, successfully, without interruption, the ore that was on fire. At the Coronado Mine of The Arizona Copper Co., at Metcalf, Ariz., mining operations, under mine-fire conditions, are now being carried on. Here, a non-reversible fan is in use. Were it possible to reverse the air currents and some one hastily gave an order for such to be done, there would be no chance for the men to escape. These are only two of many instances which might be quoted, where the installation of doors, etc. for reversing the air currents would be a real danger.

While in some districts of the United States so called reversible fans are used almost entirely, in the Southwest, with all the evidence at hand, great caution must be exercised before a so-called reversible fan is recommended.

POSSIBILITIES OF IMPROVEMENTS IN METAL-MINE VENTILATION

Improvements in ventilating systems in this region are continually being planned. These include additional intake and outlet areas, increase of volumes where necessary, further measures for the elimination of dust, and the increase of velocities in individual working places, etc. With the development of additional and cheaper power, through hydroelectric installations, the future may see innovations in connection with mine ventilation.

The use of ozone in the London subways for a number of years, is most suggestive. While primarily installed as a deodorizer, it is considered an aid to the ventilation. In the past, ozone has probably had many more opponents than advocates. However, a revival of interest

[16] Chas. A. Mitke: Mining Methods of the United Verde Extension Mining Co. *Trans.* (1919) **61,** 188.

in the use of ozone as an aid to ventilation has taken place. E. S. Hallet, chief engineer of the St. Louis Board of Education, has for several years been conducting experiments, both in his own home and in the public schools, with the use of ozone, not merely as a deodorant, but from a belief that it is at least one element of our atmosphere and, as such, a certain quantity of it is vital to health. Apparently, he has found the results of his experiments most satisfactory, as St. Louis, after having had fifteen schools equipped with ozone, has just contracted for the equipment for five more schools.[17]

To this may be added the evidence adduced by the medical authorities of France that ozone increases greatly the oxyhemoglobin of the blood, thereby increasing the oxygen-carrying capacity of it. Recently, a number of experiments were conducted by Dr. Vernon Hill and John J. Aeberly, of the Chicago Department of Public Health. In concluding a description of their experiments, these gentlemen state:

"The fact that ozone is a deodorant and a powerful germicide, and the further fact that it can be generated conveniently at a very low cost, makes it a substance that should receive the careful consideration and study of the engineering profession in connection with ventilation and air conditioning work."[18]

From South Africa also comes a suggestion as to the possible use of ozone underground. Doctor Orenstein and Mr. Ireland state:

"Ozone in very small dilutions (let us say, 1 or 2 parts in 100,000,000) seems to have a salutary effect in warding off bacterial infection, not so much by its direct germicidal action, but by its stimulating or irritating effects on the mucous membrane, producing increased flow of blood and secretion of mucus. It is stated that during the influenza epidemic of 1918, in London, the employees of the Underground Railways who worked in places where the air was supplied with ozone had an absentee percentage of 3 due to influenza, while those whose working places had no ozone had 10 per cent. absent. It would be interesting to find how far ozone would help in the mines to promote immunity from bacterial invasion of the respiratory tract."

Ozone units are now constructed in such convenient form that it is possible to carry on investigations underground in order to determine what effect ozone has on the underground air and the health of the workers. Should it be proved, after exhaustive experimentation, that ozone possesses the decided advantages claimed by its advocates, it will open up great possibilities for further improvement of mine air.

In Europe and in Mexico, liquid air has been used for blasting. About 10 years ago, preliminary investigations were made, by the writer,

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[17] Some Practical Experience with Ozone. *Heat & Vent.* (Sept., 1921).
[18] Hill and Aeberly: What about Ozone? *Heat & Vent.* (March, 1922).

with liquid air in cooling heated drifts, but at that time the cost of power necessary for its manufacture was prohibitive, and the experiments were not conclusive. With prospects of cheaper power, there is also a possibility of utilizing liquid air, both as an explosive and as an aid to ventilation.

DISCUSSION

GEORGE S. RICE, Washington, D. C.—Touching as it does on ventilation and the question of dust, humidity, and high temperature, this paper deals with some of the chief subjects of investigation in metal mines of the Mining Division of the United States Bureau of Mines. In 1910, the first preliminary survey was made by a surgeon of the U. S. Public Health Service, who had been assigned to the Bureau; unfortunately he died before his work was completed and the fruits of his labors were lost. But about 1913 we began work in the Joplin district, which was said to have perhaps the worst conditions at that time in relation to miners' phthisis. That work was carried on by Doctor Lanza of the U. S. Public Health Service, and Edwin Higgins, a mining engineer in the Bureau of Mines. Subsequently the work was extended to Butte, Mont., where we had the coöperation of the Anaconda and other companies. The work has been fruitful both from the physiological standpoint and from the direct returns in better ventilation. When Mr. Higgins left the Bureau's service, Daniel Harrington was appointed to carry on that particular work in connection with Doctor Sayers, detailed from the U. S. Public Health Service to serve as chief surgeon of the Bureau, and who has a staff of surgeons, also detailed from the U. S. Public Health Service. Much of the work of the Bureau in connection with these subjects must be done quietly and without publicity; the fullest information is given to the respective mining company so that each may apply its own measures of correction.

The Bureau has published a good deal of material of a general nature and much more is on the press. While we were greatly indebted to the South Africans in the beginning of our work, just as they were indebted to the physiologists of Great Britain for the starting of their work, the Bureau investigators have carried on and developed their own methods. They have used x-ray apparatus in making field examinations, in addition to studying the details of ventilation. As concerns the latter, with the coöperation of the Anaconda company at Butte, we are trying to determine the fundamental formulas regarding ventilation, and the coefficients of friction under conditions which have been involved in some obscurity. This follows immediately on work done for the New York-New Jersey Tunnel Commission, for which there were very elaborate set-ups in the Bureau's Experimental Mine near Pittsburgh. Those who are familiar with the Bureau's investigations in connection with this

important project know that a great deal of thorough, original work has been done which will be of benefit not only in similar projects but to the mining and metallurgical industries in respect to the health of the employees exposed to gases and dust.

It should be fully understood that the Bureau of Mines is eventually a service organization; the Director and members of the Bureau's staff consider it their duty to do everything in their power to help the mining industry, particularly in problems which it would be difficult for any one company to solve.

DANIEL HARRINGTON, Denver, Colo.—Engineers and doctors of the U. S. Bureau of Mines have been attacking the problem of metal-mine ventilation, dust, and humidity in various parts of the United States for the last five years. In addition to Joplin and Butte, we have studied in the Coeur d'Alene, in some of the mines in California, Arizona, Nevada, Michigan, Minnesota and Alabama, and we have published a few pamphlets from time to time; a more or less general report on ventilation in the Butte district may be issued soon. The general idea is to attack this question first from the viewpoint of health, follow that with the influence of that health and comfort on efficiency, and later translate these results into dollars and cents. The dollars-and-cents viewpoint is, probably, the most important in the long run because if mines cannot be run at a profit they cannot be run at all. On the other hand, the health feature is important, and cannot be wholly eliminated. One of the things that we have found is that the ventilation problem is intimately associated with mine fires, which question is of intense interest at the present time.

The matter of placing mechanical ventilating contrivances in mines, particularly in metal mines, has not been strongly attacked in the United States. Wherever we can bring to the attention of the mining companies the conditions existing in their mines, almost invariably the mines that previously relied upon natural ventilation instal mechanical equipment. I think I can say positively that we have not yet made ventilation examinations in any one state where our work has not been followed by the installing of mechanical equipment and the adoption of measures looking to prevention of underground fires and proper handling of fires should they occur.

ROBERT E. TALLY, Clarkdale, Ariz. (written discussion).—During recent years considerable attention has been directed toward underground ventilation, but very little consideration has been given the dust problem. While the best remedy for dust is an air current of sufficient volume to carry away the dust quickly, how many mines have made a thorough investigation of the dust condition with a view of rendering this condition harmless?

I am not prepared to discuss the dust problem in a technical way, as we have no such problem, and I have not given the subject detailed thought; but every mining company has a solemn duty to perform in the matter of protecting its employees. We hear so much nowadays of the scarcity and inefficiency of mine labor; this is due, to some extent at least, to the fact that the mines have not been interested in making the work and life of a miner more attractive. Good mine management consists, to a considerable degree, in the payment of fair wages, maintaining safe, sanitary, and healthful working conditions, kind and just treatment of its employees, and consideration of the living conditions of the men and their families. Attention to these details will greatly help the labor situation, and one of the most important items in the maintenance of good working conditions is satisfactory ventilation.

Ventilation to be satisfactory must be under perfect control. Used or polluted air should not travel through various workings from one level to another, for after it has traveled a few hundred feet it becomes laden with dust; also, this system is dangerous in case of fire. In a good ventilation system, an inlet is provided from which the required volumes of fresh air can be taken out on the different levels and delivered to the workings as needed, there is also a special outlet raise or shaft, or other workings, where spent air can outlet without interference.

Efficient Ventilation of Metal Mines*

BY D. HARRINGTON,† E. M., DENVER, COLO.

EFFICIENT ventilation of metal mines consists in having such complete control of air currents that there is always supplied at places where men work sufficient moving air to allow working at maximum capacity without injury to health; and in case of underground fire or of surface fire in the vicinity of mine openings, underground air currents may be quickly reversed if desired, or air may be sent into or excluded from any region and fire fumes confined to only part of the mine workings, instead of penetrating practically the entire mine.

Health and safety of workers in mines as well as proper safeguarding and operating efficiency of mining properties are so intimately associated with proper ventilation of mines (both coal and metal) that they are inseparable, yet ventilation is only too frequently neglected, especially in metal mines. The metalliferous operator, generally not confronted with inflammable gas or explosive dust which so frequently force ventilation of coal mines, usually disregards air supply altogether except such as may be obtained from compressed air; and generally, even when attention is given to metal-mine ventilation, the air currents rarely are found where they are needed, namely, the working places, especially the working faces.

After several years' study of miners' consumption, the writer is convinced that long continued breathing of air impregnated with large quantities of any kind of finely divided dust, such as is found in a large number of the working places of our mines, will ultimately produce some respiratory disease, whether it be asthma, bronchitis, or phthisis; and the progress of these diseases is hastened if there are also present high temperatures, high humidities, or harmful gases or if there is present (and allowed to remain to be breathed) large quantities of the finely divided dust when air temperatures are low, hence favorable to maximum performance of work with accompanied maximum breathing into the respiratory passages of the harmful material. It is generally accepted

that coal miners under the age of fifty are much more free of these respiratory diseases than are metal miners, and it is practically certain that this immunity of the coal miner is due largely to the fact that working places in coal mines are much better ventilated than those of metal mines. A close study of dust conditions in metal mines convinces the writer that, except for preventing formation of dust, the most efficient preventive of undue dustiness of air breathed by metal mine workers is the coursing of adequate currents of air past the places where dust is produced or is being thrown into the air—thus replacing the dusty air with dust-free air.

Because of the failure to remove poisonous fumes from detonation of explosives at confined working places, many workers in these places are ill with headache or nausea, which reduces their vitality and ability to work; not infrequently these fumes claim the lives of metal-mine workers. Failure to remove smoke from working faces prevents the worker from properly inspecting overhead conditions and thus preventing many accidents.

When metal mines encounter gases, such as CO_2, nitrogen, or, as occasionally occurs, the explosive methane, inadequate ventilation and knowledge of gases usually results in hospital cases or fatalities, or possibly in occasional closing of the place or the entire property until outside aid or information may be secured.

Metal-mine operators generally see no necessity for paying attention to air distribution but, even of those who admit the necessity of ventilation, many hold that they cannot afford such ventilating plants as are found at coal mines; as a matter of fact, they really cannot afford to continue to fail to ventilate even should the altruistic motive be disregarded. Very few metal mines are immune from fire and generally when a fire occurs, either in the mine or at buildings adjacent to surface openings, mines without ventilation control are helpless for hours, possibly days, during which great damage is done; whereas, with efficient ventilation equipment, there is rarely much delay in overcoming a fire soon after its discovery. It is the writer's opinion that even if ventilating equipment should not be used ordinarily, its installation for use in case of fire would be first-class fire insurance at reasonable cost; in fact, one large western metal-mining company has installed fans chiefly to be able to control possible future fires.

In dead-end working places with stagnant air, the worker, generally on a day's pay basis, loses a considerable part of each shift waiting for smoke to clear away, or works intermittently from illness due to fumes from the muck piles; or, if the place is hot and humid, he must rest frequently or stop to wring perspiration from his clothes or go to the station or to the level below to get a drink or cool off. Where miners' consumption or kindred diseases are found (caused largely through lack of ventilation), employment must be provided for a considerable number

of pensioners whose physical ability is practically nil yet whose rate of pay frequently is that of able-bodied workers.

The above losses caused by deficient ventilation affect chiefly the mining companies; as much as 10 to 25 per cent. of a shift is lost in cool mines and as high as 50 to 75 per cent. in hot, humid mines. There is no doubt that the dollars and cents equivalent of these specific losses to operating companies amounts to many million dollars annually in our metal mines. In addition to the above heavy financial losses to the operators, the employees on a contract basis lose a large sum in time lost while underground, and both day's pay and contract workers suffer material monetary loss through inability to work because of illness. In fact, there are few metal-mining companies or their employees who do not suffer material financial loss, actual or potential, by the failure to establish and maintain a positive ventilation system.

NECESSITY FOR MECHANICAL VENTILATION

After about five years spent largely in a study of metal-mine ventilation supplementing over fifteen years spent in more or less intimate contact with ventilation in coal and, to a less extent, in metal mines, the writer believes that no mine (coal or metal) can be efficiently ventilated without the use of mechanical equipment (especially fans). While there are probably well-ventilated mines with natural currents (that is, without the intervention of fans or other mechanical equipment), he has never visited a property of that description.

Natural ventilation is dependent chiefly on the difference in temperature between the outside air and the underground rock and water; the greater the difference, the more pronounced are the quantities circulating. At times of equalization of temperature, there is likely to be little or no air circulation; this is particularly the case if the elevation of the mine intake and the return air courses at the surface are about equal. Mines with hot rock and water temperature (say over 80° F.) are likely to have rather poor or no circulation in summer and fairly good circulation in winter; mines with rock or water temperature 60° to 80° F. will, in general, lack currents in summer and have fair ventilation in winter. Mines with rock and water temperatures below 60° F. will have fair ventilation in the summer or the winter, but the air currents reverse their direction; in the spring and fall, the air currents will be sluggish and their direction will probably change daily. Whether the underground rock and water are hot or cold or whether the surface air is hot or cold, in general, the mining company dependent on natural ventilation has practically no control of the direction of the current, hence, in the case of a fire, it is helpless; and when there is no circulation provided by nature's forces, there is no method available to provide the necessary air to safeguard the company's property or the employee's health and safety.

MAIN AIR CURRENTS

Each mine should have at least one fan, of comparatively large capacity, to control main underground air currents. Preferably, this fan should be located at the surface, in fireproof housing, and should be so arranged that the direction of currents may be reversed in a very few minutes by manipulating a few doors. The most up-to-date practice is the installation of electrically driven, high-speed fans; and where gases, such as CO_2 or nitrogen, come into workings from strata, it may be desirable to keep underground air under pressure. Local conditions will determine whether the fan should be operated as a suction or as a force unit, but, in general, the main fans exhaust foul air from mine workings. As a rule, it is desirable that the main operating shaft or tunnel be an intake and that the fan should not be located at the main operating shaft or tunnel. Where possible to arrange, the main operating (haulage) shaft or tunnel should not be used as a main air course, but this recommendation can seldom be carried out.

The main fan should be kept running the full 24 hr. instead of being stopped when the shift leaves. If desirable, the air currents may be reversed with the seasons to take advantage of natural ventilating pressure. Care should be taken to keep main air courses as free as possible of such obstructions as platforms, planks, non-working cages or skips, etc., in shafts or raises, and piles of timber, loaded or empty cars, unnecessary standing posts, etc., in crosscuts and drifts.

Smooth lining shafts that form the main air courses will greatly decrease the friction of the air flow (probably as much as 50 per cent.). If the smooth lining is done by concreting or guniting, the shaft will be practically fireproof as well. Occasionally, it is feasible to remove from main air-carrying drifts or crosscuts, timbers that are not under pressure; if the rock is gunited, it remains in place and the air flow is greatly facilitated.

Mines in which many men are employed should rarely have less than 15,000 to 20,000 cu. ft. of air per min., and large mines should have approximately 15,000 cu. ft. per min. for each 100 men employed, if the underground rock and water are cool, and at least that amount (probably more) per 50 men if the workings are hot (say over 80° F.). It is desirable to divide a mine into separate air splits each carrying 15,000 to 20,000 cu. ft. per min., each split receiving its pure air from the main intake, coursing it through certain well-defined workings having 50 to 100 men and removing the used air to the main return air course, thence to the surface before it has become excessively contaminated by smoke, heat, humidity, dust, etc. Each split should, at all times, be kept absolutely separate from other splits.

Shafts or tunnels that carry main air currents into or out of a mine

and are also used for working purposes should have sufficient cross-sectional area so that the velocity of the air may be kept below 1000 lin. ft. per min. with 1500 lin. ft. as a maximum. Underground levels that carry the air for each split of approximately 15,000 cu. ft. per min. should have sufficient area to restrict the velocity to less than 500 lin. ft. per min., and a velocity of 200 to 300 lin. ft. per min. would be preferable. In figuring the area of underground haulage levels that will be used as air courses, allowance should be made for the effective area absorbed by ore cars, locomotives, etc.

Each mine should be independent of adjoining mines for its ventilation. Intermine ventilation is inefficient at ordinary times, is likely to result in disputes and other inconveniences, and forces one mine to use vitiated air from another. In the case of mine fires intermine ventilation has repeatedly caused deaths to men in the mine that did not have the fire. While, generally, it is not feasible to drive parallel workings, as in coal mining, to insure air circulation at working faces, frequently a little forethought in planning metal-mine work, especially in well-established mines, would greatly facilitate handling of air circulation. In many mines, a lateral is driven in the foot wall parallel to and only a short distance from a drift on the vein. The lateral, however, is not driven until long after the completion of the drift; if the two were driven practically simultaneously (with the drift slightly in advance of the lateral to determine the direction of the latter) the working faces of both, by use of short crosscuts and line brattices (as in coal mining), could be supplied with fresh circulating air. Similarly, raises, especially when driven in the vein, could be placed in pairs with a small pillar between and with occasional break-throughs to allow air circulation.

The utility of overcasts, regulators, and doors is only slightly recognized in metal mining in connection with controlling air currents, particularly in case of fires. The door, in particular, is neglected. Near every main shaft, especially those carrying air, doors should be placed in every intersecting level, so that in case of fire any level may be readily closed, as regards the shaft, fire fumes prevented from spreading, and the fire itself controlled. All underground doors should be well constructed and air-tight when shut; if the air pressure is excessive the doors should be in pairs with air locks of suitable length between them, so that when one door is opened the other is closed and maintains the pressure. Every door should have a latch to hold it closed in case the direction of the air current should change. Where automatic contrivances are used with doors, the door should close rather than open when the automatic feature releases. Any door that must be closed by persons after going through it is dangerous, as it probably will not be closed at a time of panic caused by fire or other emergency. Doors constructed to allow considerable air leakage are dangerous at time of fire. All doors should

be of tight construction; if a limited delivery of air is desired, a slide or regulator should be placed in the door.

Metal-mine officials rarely recognize the necessity of sealing abandoned workings in order to force air currents to live workings. The practice of allowing intersecting openings—horizontal, inclined or vertical—to remain open to air circulation, or leakage, dissipates air into places where it becomes impregnated with fungus spores, CO_2 from timber decay, increased temperature and humidity and possibly dust. Frequently, it is not needed in the places where it goes (though needed badly elsewhere) and after having become vitiated passes into places where men work. Closing places that are not being worked would frequently result in the recovery of much material that is now lost and would remove a common source of danger to life and limb in metal mines in addition to greatly aiding ventilation.

DISTRIBUTION OF AIR TO WORKING FACES

Failure to transmit fresh air currents to working faces is the most noticeable weakness of present-day metal-mine ventilation, and unfortunately this fault is only too frequent even where there is an abundance of fresh air in the main air courses. Only rarely does a metal mine use canvas or brattice deflecting curtains, line brattices, or regulators to force air into blind-end drifts, crosscuts, raises or stopes; yet frequently these agencies could be used to advantage. Compressed air from machine-drill exhaust or from an open hose as a blower is generally relied on to ventilate such places; but it nearly always proves inadequate and is expensive.

The usual compressed-air blowers, as well as exhaust from compressed-air drills, release but 75 to 150 cu. ft. of air per min., which is by no means sufficient to carry away the gases, humidity, and dust so nearly universally found in confined working places in the metal mines, as well as the high temperatures frequently met in those places. The quantity of air thus available is not comparable to that brought by ordinary circulating methods to the face of working places in coal mines (usually 5000 to as much as 15,000 cu. ft. per min.), nor to the 700 to 5000 cu. ft. per min. readily obtainable in metal mines by the use of small fans connected to galvanized-iron or some form of flexible tubing.

For the ventilation of blind-end workings with hot rock or water, hence with hot humid air, during the past few years, small electrically driven fans (generally with motor direct connected to fan shaft) have come into extended use for forcing air through canvas or some flexible tubing to the working faces. From 700 to 5000 cu. ft. per min. are thus readily placed near the face and the worker not only has sufficient comparatively fresh cool air to remove the gases and dust, but the comparatively high velocity (generally several hundred linear feet per minute)

supplies evaporation to overcome harmful effects of highly heated and excessively humid air, body perspiration, etc.

After much experimentation, by the coöperation of manufacturers of fans and mining companies, it has been found that, when used with flexible tubing, the ordinary commercial fans are not as efficient as fans of similar construction but only one-half or two-thirds the width. Such fans deliver essentially the same quantity of air as the fan of commercial size with about 30 per cent. saving in power. One company adopted as standard 8-in. flexible tubing with a fan driven by a 3-hp. motor when the distance for air delivery is not over 200 ft.; 12-in. diameter tubing with a fan connected to a 5-hp. motor when the distance is not over 500 ft.; and 16-in. tubing with a fan driven by a 10-hp. motor for distances over 500 ft. In each case air velocity is about 2000 ft. at the end of the pipe.

One mining company found driving 200-ft. raises in hot rock (between 95° and 110° F.) an almost impossible undertaking because of hot humid air and dangerous siliceous dust, but using direct-connected fans with 3-hp. motors to force air from a point near the foot of the raise through an 8-in. flexible tubing into the raises, keeping the tubing close to the worker, cleared those places of dust and hot air and converted the raises into comfortable places. This, together with placing a large surface fan and adopting a system of splitting the air currents, assured the practicability of working this property for many years, even if hotter rock is encountered with greater depth.

A pipe introduced on the Rand for ventilating headings has some advantages over galvanized-iron and canvas pipe, according to S. de Smidt.[1] This pipe is made of concrete, reenforced with wire netting, in 6-ft. or 8-ft. lengths, and is 1 in. thick for 12-in. pipe. Sleeves about 4 in. wide are used, to cover the joint between the pipe; soft clay is pressed into the joint, and from time to time inspected and tightened. The sleeves have the advantage of permitting gradual bends, and if necessary, special curved shapes may be made. The pipe is laid on the floor, which is another advantage as it can be covered with loose material and thus protected from falls of rock. The greatest advantage, however, is that, the concrete being a non-conductor, there is less heating of the air in traveling long distances to a hot level. If conditions favor exhausting the air through the pipe, in the case of gases being given off at the face, either from the strata or from blasting, this pipe can be used, whereas long lengths of cloth pipe would not be as satisfactory. A movable length of cloth pipe at the end of the concrete pipe might be necessary for bringing the pipe close to the face; and this short piece, if used for exhaust purposes, could be reenforced with ribs.

[1] S. de Smidt, deputy inspector of mines, Department of Mines and Industry, Union of South Africa, in conversation with Geo. S. Rice, chief mining engineer, U. S. Bureau of Mines.

In many mines, especially in open stopes, there is sufficient air circulation to remove smoke, gases, and possibly the most dangerous dust from working faces, but the velocity is less than 25 ft. per min., hence not sufficient to afford relief to perspiring workers if the temperature and humidity are high. To meet this situation, a compressed-air driven fan has been devised along the principle of the small electrically driven fans used in business offices. This fan weighs about 50 lb., is readily connected to ordinary compressed-air hose, consumes about 20 to 30 cu. ft. per min. of compressed air and gives a velocity of about 1500 lin. ft. at the fan with a velocity of 200 to 500 lin. ft. per min. 20 to 25 ft. from the fan. After a fan of this description was placed in a stope with saturated air 92° F. the miner worked continuously in apparent comfort; whereas previous to its installation, he went out of the stope regularly for a 15 to 20-min. rest after having worked about that length of time. This arrangement consumes only about one-fourth as much compressed air as the ordinary compressed-air blowers and affords much greater relief with only a fraction of the operating cost of the blower.

COST AND RETURNS

The cost of installing a ventilation system in a metal mine is variable, yet under ordinary conditions it should not be particularly burdensome; generally, if any considerable number of men are employed, the savings effected greatly overbalance the cost. The recently perfected ventilation installation at the Colorado mine at Butte, Mont.,[2] cost over $70,000, but shortly after its installation it was found that the saving in electrical power for compressing air was about $18,000 per year. In addition to several indirect benefits realized, the average efficiency of all underground employees was increased at least 50 per cent. In a conversation with the writer, Mr. Bruce stated that with the mine working to its normal capacity (two full shifts), the various savings effected through the ventilation installation would readily return its entire cost of over $70,000 at least once, and probably twice, annually.

This installation was abnormal in that it demanded the driving of nearly ½ mi. of underground workings in addition to the placing of a fan. In general, a substantial ventilation installation for a comparatively large metal mine can be established for less than $15,000 and generally for not much over $5000. It requires roughly 1.5 to 2 hp. to circulate 1000 cu. ft. of air per min. through an ordinary metal mine and about 3 hp. to force 1000 cu. ft. per min. through tubing to ventilate blind ends.

If equal quantities are considered, it will cost about 100 times as much to ventilate by compressed air as by fan; in some cases, the ratio will be as high as 200 to 1. However, as only about one-tenth to one-

[2] J. L. Bruce: Octagonal Ventilation Shaft of Davis-Daly Copper Co. *Trans.* (1921) **66**, 252.

twentieth as much compressed air is supplied as when ordinary air-circulating methods are used, the cost of ventilating blind ends by compressed air is from ten to twenty times as great as by fan methods, with the additional disadvantage that rarely is ventilation by compressed air jets or blowers efficient.

A most uncomfortable part of a mine, in which about thirty men were employed on each of two shifts in dead air with a temperature over 90° F. and saturated with moisture, could be worked only by paying a bonus of about $1 per man per shift to workers who were able to deliver about one-third to one-half efficiency; moreover, the labor turnover was several hundred per cent. monthly, a combination giving slow progress and extremely high costs. Two electrically driven fans, each connected to about 600 ft. of flexible tubing, were installed, each fan consuming about 7.5 hp. at a total monthly power cost of about $30. The tubing cost about 50 c. per ft. in place, or about $600. On account of hot, humid, wet conditions, the tubing had to be replaced about every four months so that it cost about $150 per month, making the operating cost of the ventilation system about $200 per month. After the ventilation system was put into effect, the bonus was withdrawn but the amount of material moved per man per day was almost exactly doubled, and the labor turnover was reduced to essentially that of other properties. The saving due to the withdrawal of the bonus amounted to over $1500 per month, which not only cares for the monthly cost of operating the ventilation system but more than wipes out the entire first cost of the system each month.

SUPERVISION

The installation of a ventilating system in an established metal mine having extensive workings should be entrusted to an engineer familiar with ventilation problems. After the fan has been placed in position (and it should invariably be arranged that the direction of the air currents may be reversed with minimum delay and effort), the supervision of fan, doors, regulators, and other equipment and all situations with respect to air circulation should, at least for a large mine, be placed in the hands of one person who would have few, if any, duties other than those pertaining to ventilation. This person will have sufficient duties to keep him busy. He should keep in touch with proposed development work and have ready ventilation plans together with necessary equipment, supplies, etc. He should take measures to prevent return vitiated air from mixing with intake air shafts; in some so-called well-ventilated mines 50 to 75 per cent. of the air sent to the workers is recirculated vitiated air. The placing of small fans connected to tubing can be supervised so that these installations are made fireproof (even if the installation is only temporary); the fan-intake air should be kept free of

return air and the tubing should be kept in place and in repair, and the end kept sufficiently close to the working face to be of benefit to workers. Doors should be located correctly to control air currents both at ordinary times and in case of fire; they should be free of leakage, be kept closed positively by latch or otherwise, and, if necessary, disciplinary measures should be enforced to compel all persons to respect doors and rules governing them. Fans should be inspected and kept clean and in repair; such apparent trifles as occasionally cleaning fan blades of grease and mud have a great effect on fan delivery. Places not working temporarily or abandoned permanently should be sealed by canvas, gunite, concrete bulkhead, or otherwise, to prevent loss of air needed in active workings and to prevent vitiated air from abandoned places from mixing with air to be used in active workings. Sprays of cool water in intake shafts, drifts, and crosscuts, or in pipes transmitting air aid in forwarding air currents by cooling them and eliminating dust from them. A map showing ventilation features, including location of fans, doors, overcasts, etc., as well as direction of air currents should be kept up to date. In a comparatively small mine, supervision of ventilation may be given to one person such as surveyor, or a technically trained shift boss, who could also perform his other duties; it is poor policy not to have specific supervision over ventilation or to have the responsibility divided among several shift or other bosses.

CONCLUSION

This paper has been able to touch on only a comparatively few points in connection with ventilation of metal mines, considering safety and health only to a slight extent, with intent to concentrate on suggestion of desirable methods to obtain efficient metal-mine ventilation, with a slight amount of attention to costs and results translated into dollars and cents. There are many phases of the subject that deserve a separate paper, such as the relative advantage of various-shaped shaft cross-sections and various methods of smooth-lining and fireproofing of shafts, shaft stations and other places; the various features entering into matter of selection and placing of main fan, whether on surface or underground, whether direct connected or belt-driven, whether to provide for reversing of air currents, etc. Another subject of vital moment is whether a main working shaft, drift, or tunnel, shall be the intake or the return. In conclusion, it is suggested that those interested in metal-mine ventilation carefully read Mr. Bruce's paper heretofore mentioned. He shows that it pays to ventilate, even when the preliminary work is expensive; he also gives data showing the utility of circular or octagonal shafts for air delivery against rectangular shapes; gives the advantage of smooth-lining over ordinary timber-exposed shafts; and interesting frictional coefficients, costs and other data.

DISCUSSION

R. E. SIMPSON,* Hartford, Conn. (written discussion).—It is true that the failure to remove smoke from the working faces prevents adequate inspection and, therefore, opportunity to provide protection against dangerous overhead conditions. The presence of smoke affects the accident rate in another way. It is a known fact that the human element is responsible for more than three-fourths of our accidents and of this three-fourths the physical and mental conditions of the workers are influential factors. A man whose mental processes are somewhat laggard, when working at an unguarded punch press, is a distinct hazard because his muscular responses will not always be in rhythm with the motion of the press. The removal of smoke and the hot moisture-laden vitiated air, in one case, increased the productive capacity of the workers from 15 to 45 per cent. When these conditions are not alleviated by ventilation, there is an adverse effect on the workers' mental capacity; and this combined with his muscular sluggishness is directly reflected in a higher accident rate than that which prevails in a well-ventilated mine. Efficient ventilation is, therefore, desirable both from a productive and a safety viewpoint.

Sampling and Estimating Zinc and Lead Orebodies in Mississippi Valley

By W. F. Boericke, Chicago, Ill.

The character of the Wisconsin orebodies must be clearly understood to appreciate the difficulties encountered in sampling and estimating them. Unlike the western vein deposits, they do not lie between regular walls, but form large, often highly irregular bodies, on a single floor in the so-called Galena limestone. The bottom of the ore is nearly always on the clay bed (a flat generally well-defined clay seam a few inches thick) which marks the junction of the base of the Galena lime and the top of the next formation, the Trenton lime. Above the clay bed, the orebodies make the unique pitch and flat formation described by Chamberlin.[1] Theoretically, we should find two pitches, one on either side of the top crevice, diverging as they go down, with the final flats a considerable distance apart on the clay bed. The area between, known locally as core ground, should thus constitute a single defined orebody, with the richer portions near the pitches and the leaner portion in the center of the core. Practically, we find few examples of this perfect orebody. Almost invariably one pitch is better defined than the other, and often the opposite pitch is marked only by a faint line of fracture showing stronger mineralization than the wall rock, or it may be totally lacking.

But with an orebody of any size or richness, at least one strong pitch usually tends to follow an east-west trend, or a quartering course. Late mining developments in the rich New Diggings field indicate that these pitches often assume a horseshoe or crescent shape, the richer ore being in the heart of the crescent and the pitches showing less mineralization as they extend outwards. In the northern end of the field, on the contrary, the pitches maintain generally a consistent direction, and have been mined for thousands of feet from the shaft. The length of the orebody may be several thousand feet, the height varying from 10 to 60 ft. and the width from 60 to 250 ft. In fact, on the core side, there is no definite end to the ore. The mineralization simply gets thinner and thinner, and the limit of working depends on the cost sheet and the profit from the recovery.

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[1] Geology of Wisconsin. Wisconsin Geol. Survey.

The mines are worked almost universally with underhand (breast) stoping, using a room-and-pillar method for carrying the ground. In most cases, a drift is carried along the edge of the ore in advance of the stopes, and new stopes are opened up on the pitch as needed. These open stopes are frequently of great extent, sometimes 40 to 80 ft. high and 40 ft. wide between pillars.

Obviously it is impossible to sample such a stope by the usual methods, on account of the rapid changes in the richness of the pitches. Long practice and experience in the field, however, enable the mine operators to make surprisingly good estimates of their ground and the probable mill recovery, from visual examination alone; and by drawing proper amounts of dirt from the different stopes, a generally consistent mill feed is obtained.

It is of course apparent that the mill recovery, in itself, gives only an approximate idea of the ground, as this recovery will vary from day to day depending on the richness of the ore and on the ease with which the lead and zinc may be separated on the jigs from the gangue rock. In other words, with free milling ores, the recovery may be good, and with very chatty ores the recovery may be poor, from ground showing originally the same mineral content. Yet in the absence of elaborate and costly sampling of the mill feed and tailings, slimes, and overflow waters, the mill recovery serves as a rule of thumb to determine the mineral content of the ground, and is so regarded by the operators. I do not know of a case where an effort was made to sample the ground by underground methods alone. Reliance is always placed on visual examination of the ore faces, supplemented by known mill recoveries.

Churn-drill holes, spotted in advance of the heading or stope, gives the most reliable method of sampling or estimating an orebody. This is an expensive way to get the information and is only used when it is necessary to drill out additional ore, or to prospect with the idea of acquiring new leases or new ground for mining. In most cases, the drilling precedes the actual development and mining of the orebody, so that from a careful record and study of the drill logs, a tolerably good idea is gained of the mineral content of the ground, both as regards mill recovery and grade of concentrates, and the height of stopes, direction and dip of pitches, character of the back, etc.

When it is determined to drill ahead of an existing stope, a drill is spotted according to a careful survey about 50 to 100 ft. ahead of the stope, along the general direction of the range. Knowing the respective elevation of the surface and the back of the stope, the drill man is instructed to cut down to 2 or 3-ft. runs when he reaches the depth where he should get the top of the ore. All sludgings, thereafter, are carefully sampled and saved for subsequent examination and assay by the drill engineer in charge of the work. The drilling is continued until the clay

bed, or glass rock, the next formation, is reached, regardless of whether the ore continues down or not. The drill engineer makes a log of the hole, for office use, noting particularly the character of the ground, with any change of formation, the amount of zinc, lead, and marcasite (known locally as sulfur), the presence of open ground and crevices, the water level, and particularly how the ore occurs (as sheet, disseminated, or crevice ore) and any other items of interest that may help to interpret the hole. The record of a typical drill hole might run as follows:

MARTIN LEASE

Hole.......... 46
Location....... 50 ft. south of 45
Elevation...... 846
Oil rock....... 720
Water level ... 70 ft.

Started, June 20, 1919
Finished, June 25, 1919
Drilled by, Stevens
Examined by, W. F. B.

		Zinc, Per Cent.	Iron, Per Cent.
0–10	Clay		
10–46	Brown rock, flints		
46–80	Hard gray, little sulfur		
80–96	Gray lime, little sulfur and lead		
96–100	Gray, good zinc, fair lead	ASSAYS	
100–102	Opening, lost cuttings, strong water	96–100......... 8.0	2.2
102–110	Gray, fair zinc, fair sulfur	102–110......... 5.5	5.5
110–116	Blue rock, little zinc, good sulfur	110–116......... 3.5	6.0
116–126	Blue rock, good zinc, good sulfur	116–126......... 7.0	8.0
126–130	Soft oil rock, little zinc, considerable sulfur	126–130......... 2.0	10.0
130–131	Clay bed.		

From the log and assays of the drill hole, the mineral content of the ground can be readily calculated, in terms of ZnS and FeS$_2$. Taking the above as an example we should have:

	Zn	ZnS	Ore, Feet	Fe	FeS$_2$	Ore, Feet
96–100..................	8.0	11.7	46.8	2.0	4.3	17.2
102–110..................	5.5	8.0	64.0	5.5	11.8	94.4
110–116..................	3.5	5.1	30.6	6.0	12.8	76.8
116–126..................	7.0	12.1	121.0	8.0	17.1	171.0
			262.4			359.4

So, for 28 ft. of ground the average will be 9.3 per cent. ZnS, and 12.8 per cent. FeS$_2$. Although no cuttings were recovered from 100 to 102 ft., it is safe to include them and figure on 30 ft. of ground to work.

The mill recovery, if the zinc and marcasite are free milling, will be about 70 per cent. for the zinc and 50 per cent. for the iron. Using these

ratios, this ground will show a recovery of 6.5 per cent. ZnS and 6.4 per cent. FeS$_2$, about 13 per cent. "dirt." The concentrates should run about 35 per cent. Zn.

It must be understood that these calculations are in no sense exact. They are approximations and much depends on the examining engineer's judgment of the appearance and character of the cuttings, and the care with which the hole was drilled and sampled. It is easy to salt a drill hole by carelessness in saving the sludgings; and if the water is heavy, a run may appear to be much richer than subsequent mining will show. Similarly, open ground may carry away much of the rich ore, leaving only a small sample of what should be a rich run. Even more important is the ability to size up the character of the ore and the formation in which it occurs. Chatty or disseminated ore may assay well, but the subsequent mill recovery may be disappointing, whereas a small run of sheet jack may surpass expectations. Careful visual examination of the cuttings by an experienced man is worth fully as much as assays and is sometimes to be preferred.

After the first hole has cut ore, the next is usually a crosscut hole, at right angles to the strike of the pitch. Knowing the dip of the pitch, the engineer will expect to get ore in his crosscut hole at a higher or lower elevation than in the first, and on plotting these holes in cross-section, he can determine whether or not the range is turning, and can spot his following holes with this in mind.

Blank holes will indicate the outside of the range and lean holes an approach to the edge. When the work is completed, lines can be drawn approximating the lateral extent of the orebody. From the stope heights in the various holes, the average vertical section can be figured and the cubic contents of the block and tonnage calculated.

Such, in brief, is the ordinary procedure in estimating the orebodies in Wisconsin. In general, the results, as shown by subsequent mining and milling practice, have been reliable, but there are many cases of failure. This often has been due to the failure to recognize the character of the ore (that it is not free milling) or of the ground, especially in disseminated deposits. Either condition may give higher assays than will be shown in subsequent development. It pays to be reasonably conservative in reporting on drill holes, and invariably to disregard rich runs and assays if they do not indicate a consistent mineralization of the ground, as checked out by other holes. In other words, a single isolated rich run in a drill hole, if not shown in crosscut holes when logically expected, should be disregarded in estimates. Beware of orebodies with "freak" drill holes. Cases are known where a hole passed through a rich sheet of ore but failed to show good sludgings, because a break, or open crevice, in the rock let the cuttings get away. I have seen a 6-in. flat cave literally packed with cuttings from a rich sheet of ore through which

the hole had passed, without recovering anything. A good drillman, however, will be on the lookout for all openings and, if any is found near by, a poor run may be given more consideration than it would otherwise warrant.

Diamond drilling has been tried in Wisconsin, but found unsuitable on account of the nature of the ground and the difficulty in recovering cores.

Prospect drilling, once ore has been found, can be overdone. In the enthusiasm of getting results and finding good ore, much money can be spent in obtaining information that would probably be learned in subsequent development work, and with more certainty. Where such information will save considerable unnecessary prospecting underground, the case is different; but holes can be spotted merely to settle academic points that, although interesting in aiding to calculate tonnages, will cost more than the information is worth. It is not worth while trying to delimit an orebody too sharply, once the approximate extent is known, and a profit from the operation is in sight. Subsequent drilling, if needed, can probably be better spotted in the light of underground development.

DISCUSSION

J. PARKE CHANNING, New York.—Practically identical conditions are found in the Lake Superior copper mines. The Kearsarge lode has been continuously operated for a number of years; the yield varies from 18 to possibly 24 lb. of copper.

It is absolutely impossible to do any sort of hand sampling on a lode of this kind, the average thickness of which is about 14 ft., so we rent a stamp-head from one of the operating properties and as we drive our drifts on the various levels depend on the judgment of the foreman as to what is copper rock. The copper rock is sent up separately and when sufficient has been stored it is run through one of the heads. The weight is known and the yield, and these are correlated with the total amount of rock taken from the opening. Undoubtedly there is a great deal of copper in the portion rejected as poor rock; but it is almost impossible to determine what that is.

In running drifts, they are usually made wide so that a triangle may be cut out in the foot wall and occasionally in the hanging wall. When calculations are made, we are able to get some rough approximation as to the yield of the rock. But of course many other factors enter into the mining, such as when you stope whether some of the rock will be left behind because of its low grade or because pillars are needed.

In the very flat lodes, the rock must be brought down by a mechanical scraper; if too many pillars are left either for support or because they are considered too poor to mine, they will interfere with the scraper. You

can see the extreme difficulty there is in determining grade in ore of this kind. In going through a mine that is opened up, a person must rely largely upon the eye, the feel of the rock, and his experience in the particular lode being examined.

ARTHUR THACHER, St. Louis, Mo.—In some parts of the Joplin and Miami fields, and to a certain extent in southeast Missouri, they use the drill, but those familiar with the mine in that particular location judge more from the appearance of the rock than from any assays. The most successful mine opened there only had, from the lead content, one fair hole in the drilling and most of the holes lacked all traces of lead and were clearly not workable, and still that was one of the largest mines in the district.

Diamond-drill Sampling Methods

By Robert Davis Longyear,* Minneapolis, Minn.

(New York Meeting, February, 1922)

In diamond-drill work, a true sample consists of all the material cut by the bit—both core and cuttings. As the recovery of this sample is the object of diamond drilling, the utmost care should be taken to secure it. Speed of drilling and low costs are of little value if an inaccurate sample is obtained. The engineer in charge must study his material and lay out the plan of sampling, and then see that the drill runner follows his instructions; for the runner may be more interested in obtaining a large footage than in developing accurate methods of sampling.
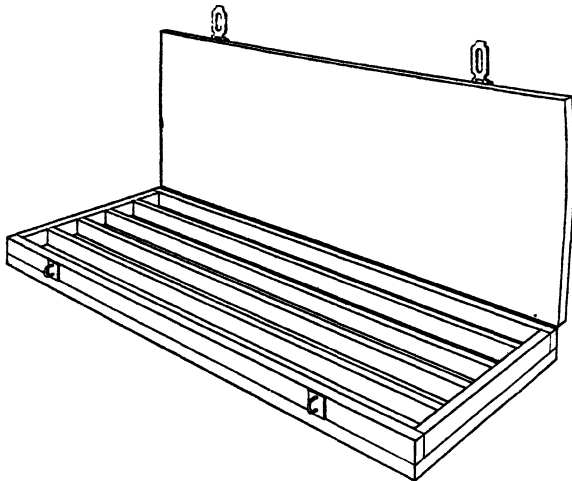
Fig. 1.—Core box.

When the core has been removed from the core barrel, it should be placed in a core box in the exact order in which it is taken from the ground. This box, Fig. 1, is usually a shallow tray 4 or 5 ft. (1.2 or 1.5 m.) long, with partitions running lengthwise between which the core

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* Geologist, E. J. Longyear Co.

fits snugly. It is so designed that, after the cover is fastened in place, the core will retain its position when the box is moved. As the core from each run is placed in the box, a wooden block, upon which the depth of the hole is written, should be placed after the last piece of core. Each run is thus separated and shows the exact depth at which it was made. If the core is to be used for certain types of inspection, such as foundation testing, a mark on each piece should show which end was uppermost in the hole. This will prevent a piece being placed in the box in the reversed position. Frequent checks of the depth of the hole should be made with a steel tape.

In oil reconnaissance work, where the drill is used to determine the structure by establishing the elevations of certain key horizons, the essential information desired is the exact depth at which certain correlated beds are reached in different drill holes. It is, therefore, especially important to preserve the core in its original position in order to study the various criteria of correlation such as fossils, shale partings, color changes, and contacts. It is often desirable to have the runner keep a separate record of the depth of the harder and softer beds, as indicated by the pressure on the drill feed. This will serve as a check against the record given by the core.

Collecting the Sludge

The ideal sample of the material drilled would consist of 100 per cent. core of uniform size, but this ideal is seldom reached. In certain ores, the rich and softer streaks will tend to grind, so an analysis of the core that is left will not give the true grade of the material drilled. In such cases, the cuttings or sludge must also be collected and analyzed. The correct grade of the ore is obtained by analyzing both core and sludge separately and combining the results, as described in Method 3.

In order to obtain a correct sludge sample, all of the water pumped down the rods must be returned, for if any is lost in crevices or in broken and porous rock part of the sample will go with it. To avoid this, the openings in the walls of the hole must be closed by casing, cementing, or other treatment. Care must be taken that the sample is not enriched or impoverished by material caving from the wall of the hole above the bit. If there is any tendency for this to take place, the casing must be kept well down toward the bottom of the hole. There are three methods in common use for collecting the sludge; the one to be adopted depends on the character of the material drilled and the results desired.

Method 1.—When the material shows little tendency toward concentration, the water may be run into a tub and allowed to overflow. Enough cuttings will be retained in the tub to furnish a sample. By this method, however, the sample is usually enriched, for the lighter particles are carried off. This method, therefore, is rarely satisfactory.

Method 2.—The sludge box (or floor box), Fig. 2, will give much better results. This box is 12 to 18 in. (30 to 45 cm.) wide, 6 to 10 in. (15 to 25 cm.) deep, and 4 to 10 ft. (1.2 to 3 m.) long and contains two or more baffle plates to aid in the settling of the cuttings. The water may run into the box through a tee, or the casing may pass through a properly packed stuffingbox or gland in the bottom of the box. Samples may be removed at any time that the rods have to be pulled, but preferably at 5 or 10-ft. intervals. In any case, care should be taken that core and cuttings represent the same interval in the hole. At the end of each interval, the drill is stopped and water pumped through the rods until
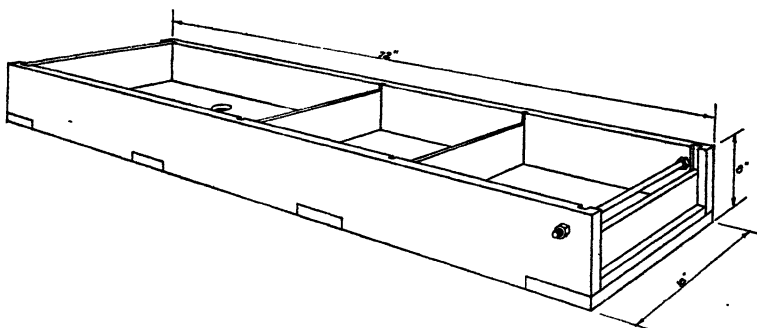
FIG. 2.—SLUDGE BOX.

the hole is clear of all cuttings and the return water clean. After the cuttings have settled in the sludge box, the top water is carefully siphoned or poured off, leaving the residue in the bottom. This is dried and quartered to convenient size for analysis or assaying, a representative sample being permanently preserved. This is the method most commonly used when diamond-drilling iron ore. In most ores, there may be considerable concentration of the heavier minerals, especially if the metallic content is small or if there is a great difference in specific gravity between the ore minerals and waste. The sludge box should be used only with full recognition of its limitations.

Method 3.—In the third method, which is the most reliable, the return water is run into three or more barrels through a pipe or launder. When one barrel is filled, the pipe is swung to the next barrel, and so on. After the material in the first barrel has settled sufficiently so that the clear water can be siphoned off, the barrel is again filled. The process is continued until the end of the interval to be sampled. All of the material is then poured into one barrel and allowed to settle further. After all of the clear water has been drained off, the residue is removed, slowly dried over a fire, quartered, and sacked. The number of barrels used depends on the time necessary for the material in suspension to settle. When this method is used, the hole must be in such condition

FIG. 3.—RELATION OF CORE RE-
COVERED TO TOTAL SAMPLE.
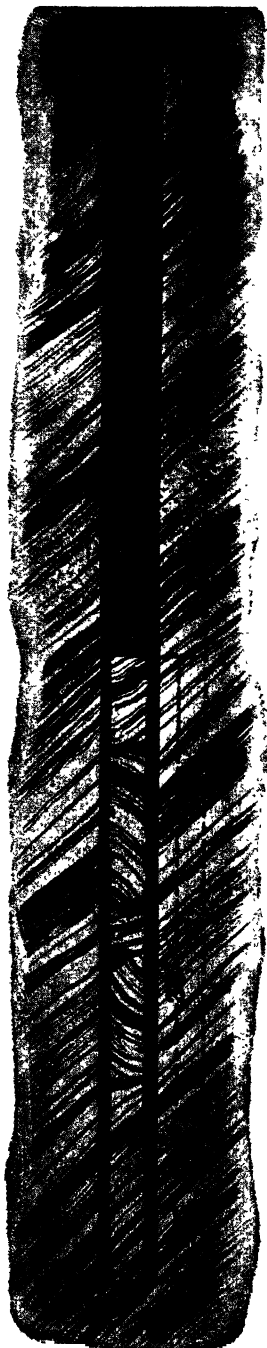
that all of the cuttings are washed to the surface. On one of the large copper deposits in Arizona, this method gave remarkably accurate results even though less than 50 per cent. of the core was recovered. Later test pits and mining operations checked the drill sampling within 0.005 per cent. in carbonate ore and 0.05 per cent. in sulfide ore.[1] A bit recovering a 1⅝-in. core was used. The choice of sampling methods must depend on the engineer's judgment of the character of the material to be sampled. In case of doubt, it is better to err on the side of accuracy than to risk getting a worthless sample.

AVERAGING CORE AND SLUDGE ANALYSES

To obtain the average analysis of the material drilled in a given run, the separate analyses of both core and sludge (cuttings) must be combined in the proportion of their respective masses. Fig. 3 represents a 5-ft. run with an "N" bit, in which only half of the core was recovered. The cuttings represent all that part of the hole which is not core; but it would not be correct to take the arithmetical mean of the two samples because, as the illustration shows, the cuttings are the largest part of the sample and must be weighted accordingly. In this case, the core is only 25.3 per cent. of the total sample, and the cuttings are 74.7 per cent. If the analysis of the core is multiplied by 25.3 and added to the analyses of the cuttings multiplied by 74.7 and the sum divided by 100, the result will be the average analysis.

Table 1 has been prepared to facilitate the calculation of average analyses; the figures are based on the ratios of the volume of core and the volume of cuttings to the total volume of the hole. The values of the sepa-

[1] Ira B. Joralemon: The Ajo Copper Mining District. *Trans.* (1914) 49, 605–607.

rate analyses made for core and cuttings in a 5-ft. run should be multiplied by the percentage found in the column headed by the size of the bit used, opposite the number of inches of core recovered. The results are to be added and divided by 100 for the average analysis for that run.

Assume that in a 5-ft. run with an "A" bit, 34 in. of core are recovered and that the analysis of the core is 57.6 per cent. iron and of the cuttings, 61.0 per cent. Opposite 34 in the column of Table 1 headed Inches of Core, in column A, appear the values 20.6 and 79.4, which are the percentages of core and cuttings of the total material cut in the hole. Then,

$$\frac{57.6 \times 20.6 + 61.0 \times 79.4}{100} = 60.3 \text{ per cent.}$$

Although this table is made for a 60-in. run, it is a simple problem of proportion to adapt it to a run of any other length. For instance, if a 38-in. run is made and 23 in. of core are recovered, multiply 23 by 60 and divide by 38, which will give a value 36, which for the purpose of the table is to be used as the number of inches recovered.

The table is accurate only in case: (1) the bit is of the dimensions specified at the head of the columns, (2) the hole is of uniform size with no increase in diameter in the softer strata, (3) there are no additions from caving in the upper portions of the hole, and (4) all of the cuttings are recovered. If a portion of the cuttings is lost or other material is added, the sample will be correct only to the extent that the material analyzed represents a true sample of the actual cuttings. Hence the necessity for extreme care in recovering and sampling the sludge.

SPLITTING THE CORE

It is often desirable to retain a complete core sample to show structure or banding and at the same time have a portion of the core analyzed, especially where the results of exploration are to be submitted to a prospective purchaser. In such cases the core may be split longitudinally by a core splitter; it is then possible to grind up one-half into a pulp for analysis while the other half is preserved in its original state.

PERMANENT FILING OF SAMPLES

The method adopted for permanently filing samples depends on the needs of the company doing the drilling. Sludge may be preserved in sacks and the core in the boxes shown in Fig. 1. Some companies find it more convenient to keep the cores in drawers with partitions similar to those in the box; the core is then accessible for future study. In the

TABLE 1.—*Percentages in an Average Analysis of a 5-ft. Diamond-drill Sample when Core and Cuttings are Analyzed Separately*

Inches of Core	E Standard Size of Bit O.D., 1¹½₂ In. I.D., ³½₂ In.		E Mesaba Size of Bit O.D., 1¹½₂ In. I.D., 1⁵⁄₁₆ In.		A Size of Bit O.D., 1²½₂ In. I.D., 1⁵½₂ In.		B Size of Bit O.D., 2¼₂ In. I.D., 1⁷⁄₁₆ In.		N Size of Bit O.D., 2²⁵½₂ In. I.D., 2¼₁₆ In.		Inches of Core
	Core	Cuttings	Core	Cuttings	Core	Cuttings	Core	Cuttings	Core	Cuttings	
1	0.6	99.4	0.6	99.4	0.6	99.4	0.7	99.3	0.8	99.2	1
2	1.1	98.9	1.1	98.9	1.2	98.8	1.5	98.5	1.7	98.3	2
3	1.7	98.3	1.7	98.3	1.8	98.2	2.2	97.8	2.5	97.5	3
4	2.2	97.8	2.3	97.7	2.4	97.6	3.0	97.0	3.4	96.6	4
5	2.8	97.2	2.8	97.2	3.0	97.0	3.7	96.3	4.2	95.8	5
6	3.4	96.6	3.4	96.6	3.6	96.4	4.4	95.6	5.1	94.9	6
7	3.9	96.1	4.0	96.0	4.2	95.8	5.2	94.8	5.9	94.1	7
8	4.5	95.5	4.5	95.5	4.8	95.2	5.9	94.1	6.8	93.2	8
9	5.0	95.0	5.1	94.9	5.4	94.6	6.7	93.3	7.6	92.4	9
10	5.6	94.4	5.7	94.3	6.1	93.9	7.4	92.6	8.4	91.6	10
11	6.2	93.8	6.2	93.8	6.7	93.3	8.1	91.9	9.3	90.7	11
12	6.7	93.3	6.8	93.2	7.3	92.7	8.9	91.1	10.1	89.9	12
13	7.3	92.7	7.4	92.6	7.9	92.1	9.6	90.4	10.9	89.1	13
14	7.8	92.2	7.9	92.1	8.5	91.5	10.4	89.6	11.8	88.2	14
15	8.4	91.6	8.5	91.5	9.1	90.9	11.1	88.9	12.7	87.3	15
16	9.0	91.0	9.1	90.9	9.7	90.3	11.8	88.2	13.5	86.5	16
17	9.5	90.5	9.6	90.4	10.3	89.7	12.6	87.4	14.3	85.7	17
18	10.1	89.9	10.2	89.8	10.9	89.1	13.3	86.7	15.2	84.8	18
19	10.6	89.4	10.8	89.2	11.5	88.5	14.1	85.9	16.0	84.0	19
20	11.2	88.8	11.3	88.7	12.2	87.8	14.8	85.2	16.9	83.1	20
21	11.8	88.2	11.9	88.1	12.8	87.2	15.6	84.4	17.8	82.2	21
22	12.3	87.7	12.5	87.5	13.4	86.6	16.3	83.7	18.6	81.4	22
23	12.9	87.1	13.0	87.0	14.0	86.0	17.0	83.0	19.5	80.5	23
24	13.4	86.6	13.6	86.4	14.6	85.4	17.8	82.2	20.3	79.7	24
25	14.0	86.0	14.2	85.8	15.2	84.8	18.5	81.5	21.1	78.9	25
26	14.6	85.4	14.7	85.3	15.8	84.2	19.3	80.7	21.9	78.1	26
27	15.1	84.9	15.3	84.7	16.4	83.6	20.0	80.0	22.7	77.3	27
28	15.7	84.3	15.9	84.1	17.0	83.0	20.7	79.3	23.6	76.4	28
29	16.2	83.8	16.4	83.6	17.6	82.4	21.5	78.5	24.4	75.6	29
30	16.8	83.2	17.0	83.0	18.2	81.8	22.2	77.8	25.3	74.7	30
31	17.4	82.6	17.6	82.4	18.8	81.2	23.0	77.0	26.1	73.9	31
32	17.9	82.1	18.1	81.9	19.4	80.6	23.7	76.3	27.0	73.0	32
33	18.5	81.5	18.7	81.3	20.0	80.0	24.4	75.6	27.8	72.2	33
34	19.0	81.0	19.3	80.7	20.6	79.4	25.2	74.8	28.7	71.3	34
35	19.6	80.4	19.8	80.2	21.2	78.8	25.9	74.1	29.5	70.5	35
36	20.2	79.8	20.4	79.6	21.8	78.2	26.7	73.3	30.4	69.6	36
37	20.7	79.3	21.0	79.0	22.4	77.6	27.4	72.6	31.2	68.8	37
38	21.3	78.7	21.5	78.5	23.0	77.0	28.1	71.9	32.1	67.9	38
39	21.8	78.2	22.1	77.9	23.6	76.4	28.9	71.1	32.9	67.1	39
40	22.4	77.6	22.7	77.3	24.3	75.7	29.6	70.4	33.7	66.3	40
41	23.0	77.0	23.2	76.8	24.9	75.1	30.4	69.6	34.6	65.4	41
42	23.5	76.5	23.8	76.2	25.6	74.5	31.1	68.9	35.4	64.6	42
43	24.1	75.9	24.4	75.6	26.1	73.9	31.8	68.2	36.2	63.8	43
44	24.6	75.4	24.9	75.1	26.7	73.3	32.6	67.4	37.1	62.9	44
45	25.2	74.8	25.5	74.5	27.3	72.7	33.3	66.7	38.0	62.0	45
46	25.8	74.2	26.1	73.9	27.9	72.1	34.1	65.9	38.8	61.2	46
47	26.3	73.7	26.6	73.4	28.5	71.5	34.8	65.2	39.6	60.4	47
48	26.9	73.1	27.2	72.8	29.1	70.9	35.5	64.5	40.5	59.5	48
49	27.4	72.6	27.8	72.2	29.7	70.3	36.3	63.7	41.3	58.7	49
50	28.0	72.0	28.3	71.7	30.4	69.6	37.0	63.0	42.2	57.8	50
51	28.6	71.4	28.9	71.1	31.0	69.0	37.8	62.2	43.1	56.9	51
52	29.1	70.9	29.5	70.5	31.6	68.4	38.5	61.5	43.9	56.1	52
53	29.7	70.3	30.0	70.0	32.2	67.8	39.2	60.8	44.8	55.2	53
54	30.2	69.8	30.6	69.4	32.8	67.2	40.0	60.0	45.6	54.4	54
55	30.8	69.2	31.2	68.8	33.4	66.6	40.7	59.3	46.4	53.6	55
56	31.4	68.6	31.7	68.3	34.0	66.0	41.5	58.5	47.2	52.8	56
57	31.9	68.1	32.3	67.7	34.6	65.4	42.2	57.8	48.0	52.0	57
58	32.5	67.5	32.9	67.1	35.2	64.8	42.9	57.1	48.9	51.1	58
59	33.0	67.0	33.4	66.6	35.8	64.2	43.7	56.3	49.8	50.2	59
60	33.6	66.4	34.0	66.0	36.4	63.6	44.4	55.6	50.7	49.3	60

Lake Superior iron districts, nearly all the mining and exploration companies have filed both core and sludge in tin boxes 12 by 1½ by 1 in. (30.5 by 3.8 by 2.6 cm.) with hinged cover. In the ore, the core recovery is small. Half the split core, or often a few representative pieces, are placed in the tin box with a sample of the sludge and the end of the box is properly labeled. Representative samples of the hard rock above the ore are kept also in tin boxes. By this method samples can be stored in a relatively small space.

Summary

The foregoing is a brief description of a few of the common methods of collecting and preserving diamond-drill samples. Variations of these methods will suggest themselves on every new piece of work. The essential point to bear in mind in all diamond-drill sampling is that the sample must comprise all of the core obtainable and all of the cuttings. This involves the proper operation of the drill as well as the careful collecting of the core and sludge. The engineer must use his own judgment as to the practical methods that will give the desired results under the conditions obtaining.

DISCUSSION

Ira B. Joralemon, Warren, Ariz. (written discussion).—This paper is so comprehensive, and still concise, that it should be adopted as instructions for samplers on diamond-drilling jobs. It brings out one point on which we now know little, namely just how accurate are sludge samples taken in overflow tubs or in sludge boxes compared with those caught in a series of barrels with no overflow. It would be interesting if the flow of sludge from drill holes in different sorts of ore could be divided evenly, part going to an overflow tub, part to sludge boxes, and part to a series of barrels, and the results compared.

J. J. Rutledge, Urbana, Ill. (written discussion).—Fig. 3 is a particularly valuable illustration. I do not recall having seen one that so clearly shows the conditions in drill holes in steeply pitching beds, especially the hard sandstones of the Pottsville. Recently I saw cores just taken from coal measures of one of the middle-western states, with which region I am familiar, where the driller had apparently recovered every foot of the ground passed through, even though some of it was very friable and difficult to core successfully. On inquiry I learned that this driller, who had the reputation of drilling considerably more footage than the others, used a single core barrel and cored successfully strata that others failed to core. The personal equation enters largely into the problem of securing good cores in diamond drilling. In this instance, the diamonds were well chosen and set so as to get the best results. In

another case, recently, I found that at least four-fifths of the soft-shale core had been lost because it had been left exposed to the elements and not placed in a proper core box. If the core had been coated with shellac, or saturated with glue, valuable information, to obtain which thousands of dollars had been expended, would have been preserved for reference.

E. C. HARDER, Philadelphia, Pa. (written discussion).—The object of prospecting with drill is to secure quickly and cheaply as accurate a sample as possible of mineral material beneath the surface. Yet often the real object is lost sight of in the anxiety of the driller to make footage. Frequently drillers make no attempt to recover the fine sludge, only retaining the core and coarser cuttings and thus obtaining results that are unreliable.

Many drilling concerns do not keep a permanent record of the holes they drill, either written or in the form of core. They simply sink holes and throw away core and sludge until they encounter what, in the opinion of the driller, may be ore. Then they take rough samples and analyze. With little additional trouble the cores could be packed into boxes, as described, and retained as a permanent record for future inspection. A great deal of drilling has been done in the Lake Superior iron-ore districts of which no record was kept; this has necessitated drilling many areas twice or even three or four times, involving great expense. Drilling done in a careless manner and without adequate supervision is a most wasteful operation.

Basic Principles of Gravity Concentration—A Mathematical Study

BY THEODORE SIMONS,* BUTTE, MONT.

THE rapid and comparatively recent development of flotation has opened so fascinating a field for study and research that the older processes of gravity concentration no longer receive the attention they deserve.   The early work of Munroe[1] and Richards,[2] and the more recent contributions by Clevenger and Coe,[3] Laist and Wiggin,[4] Ammon,[5] Bardwell,[6] Crowfoot,[7] and Hayden[8] (to mention only some which have appeared in our own *Transactions*) and by Schulz,[9] have furnished so much data that to offer something new seems a difficult task.   It is still possible, however, to present known facts from a different point of view, and, with this in mind, the writer has undertaken to express basic principles of gravity concentration in terms of simple mathematics and mechanics; the conclusions thus reached will be found to correspond, in general, with those obtained by experimental methods.

Rittinger, in his classic *Aufbereitungskunde* (1867), reached analogous conclusions by way of intricate higher mathematics, for which reason his interesting investigations have remained a sealed book to most of us. The writer hopes that a more elementary treatment of the subject, based in part on Rittinger's methods, may arouse new interest in the study of basic principles.

* Professor of Mining Engineering, Montana State School of Mines.
[1] H. S. Munroe: *Trans.* (1888–89) **17**, 637.
[2] R. H. Richards: *Trans.* (1899–90) **18**, 644; (1894) **24**, 409; (1897) **26**, 3; (1897) **27**, 76; (1907) **38**, 210, 556; (1908) **39**, 303; (1910) **41**, 396.
[3] *Trans.* (1916) **55**, 356.
[4] *Trans.* (1914) **49**, 470.
[5] *Trans.* (1913) **46**, 277.
[6] *Trans.* (1913) **46**, 266.
[7] *Trans.* (1914) **49**, 417.
[8] *Trans.* (1913) **46**, 239.
[9] Investigations by kinematography, *Glückauf* (1915) **51**, 457, 481, 510, 540, 562.

GRAVITY CONCENTRATION

The most widely employed processes of gravity concentration depend on the behavior of mineral particles in water (dry processes also have their place) and take advantage of the difference in the speed with which minerals of varying size and density fall in still or moving water; or of the difference in the resistance that they offer to a current of water flowing down an incline.

If a mixture of mineral particles of various sizes and densities is allowed to fall through water from a given height, all starting at the same instant, the individual particles of the mixture will form layers, according to well-defined laws. The bottom layer will contain coarse particles of the densest minerals, but only the largest particles of the less dense ones; while the uppermost layer will contain the smallest particles of the densest minerals, side by side with both coarse and fine particles of the less dense ones. The particles found in any particular layer will be of different sizes and densities, but will have a nearly "equal-falling" velocity in water; that is, they traversed equal distances during the same time interval.

A study of "equal-falling" mineral particles of uniform shape shows that their volumes and densities are interdependent. In vacuo, all bodies are "equal falling," irrespective of size, density, or shape. Falling through a fluid, whether air, water, or other liquid, they meet a resistance, the most perceptible effect of which is a decrease of acceleration; if falling in water, acceleration disappears very quickly, and a particle completes its fall at uniform velocity.

FALL OF A BODY IN STILL WATER

The motion of a body falling through water is not uniform, due to the fact that acceleration is diminished by the frictional resistance of the water.

Let W = absolute weight of the falling body, in pounds.

L = loss of weight, in pounds, due to buoyancy.

= weight of water displaced, in pounds.

W_1 = net weight $(W - L)$, in pounds.

= weight of the body when submerged in water.

V = volume of the body, in cubic feet.

M = mass of body = W/g.

δ = its relative density.

s = specific weight of water (62.5 lb. per cu. ft.).

g = acceleration due to gravity in feet per second per second.

Then,

$$W_1 = (W - L) \text{ pounds.} \tag{1}$$

W_1 is the force that would impart to the mass M of the falling body an acceleration g_1 if the water offered no resistance. In that case:

$$W_1 = Mg_1 = (W - L), \text{ in which} \qquad (2)$$
$$W = V\delta s \text{ pounds} \qquad (3)$$

and

$$L = Vs = \frac{W}{\delta} \text{ pounds}$$

Substituting in equation (2) gives

$$W_1 = W - \frac{W}{\delta} = W\left(1 - \frac{1}{\delta}\right) = Mg_1$$

or

$$W_1 = \frac{W}{\delta}(\delta - 1) = Mg_1 = \frac{W}{g}g_1 \qquad (4)$$

whence

$$g_1 = \frac{g}{\delta}(\delta - 1) \qquad (5)$$

The force W_1 is opposed by the pressure of the water against the falling body. The actual force W_2 imparting acceleration is therefore W_1 minus the water pressure, and the acceleration, which in the absence of that pressure would be g_1, is changed to g_2.

Let v = velocity of the falling body, in feet per second.

A = area, in square feet, of a section through the body at right angles to the direction of fall.

P = total pressure, in pounds, exerted by the water against the falling body.

Theoretically this pressure P is the same as the pressure that a stream of water, ascending vertically with a velocity v, would exert against a horizontal plane of area A. It is equal to the weight w of a column of water having a base A sq. ft. in area, and a height h ft., corresponding to a velocity v, or

$$P = w \text{ pounds} \qquad (6)$$

The theoretical velocity is: $v = \sqrt{2gh}$ ft. per sec. whence

$$h = \frac{v^2}{2g} \text{ ft.}$$

The weight w of the water column is

$$w = Ahs = A\frac{v^2}{2g}s \text{ pounds}$$

whence

$$P = \frac{v^2}{2g}sA \text{ pounds} \qquad (7)$$

In falling through water, the body must also overcome viscosity. At high velocities the effect of viscosity is so small, however, that it may

be neglected. But since velocity and, as a consequence, resistance are affected by the shape and nature of the falling body, the actual resistance P_a may be written:

$$P_a = c\frac{v^2}{2g}As \qquad (8)$$

in which c is an empirical coefficient, the value of which is less than unity.

Thus the force W_2, which causes the mass M of the body to fall through water with the acceleration g_2, is

$$W_2 = (W_1 - P_a) = Mg_2 = \frac{W}{g}g_2 \qquad (9)$$

Substituting W_1 from (4), and P_a from (8) gives:

$$\frac{W}{g}g_2 = \frac{W}{\delta}(\delta - 1) - c\frac{v^2}{2g}sA$$

whence

$$g_2 = \frac{g}{\delta}(\delta - 1) - c\frac{v^2}{2W}sA \qquad (10)$$

from (3)

$$W = V\delta s$$

hence

$$g_2 = \frac{g}{\delta}(\delta - 1) - c\frac{v^2}{2V\delta}A \qquad (11)$$

If the body has the shape of a sphere of diameter d ft., then

$$A = \frac{\pi}{4}d^2 \text{ sq. ft.}$$

$$V = \frac{\pi}{6}d^3 \text{ cu. ft.}$$

and

$$g_2 = \frac{g}{\delta}(\delta - 1) - \frac{3}{4}c\frac{v^2}{d\delta} \qquad (12)$$

A study of equations (10), (11), and (12) leads to the following interesting conclusions:

1. The acceleration g_2 of a body of given density δ falling through water depends, in general, on the variable values in the right-hand terms of equations (10), (11), and (12).

2. The acceleration also depends on the absolute weight and on the volume of the body; it increases with increase in weight or volume, and vice versa; that is, the density being the same, larger particles have a greater acceleration than smaller ones.

3. The acceleration g_2 decreases with the increase of the square of the velocity v^2 and with the increase of the normal section A of the falling body.

4. The acceleration g_2 is greatest when $v = 0$, that is, at the beginning of fall. In that case

$$g_2 = \frac{g}{\delta}(\delta - 1) = g_1 \qquad (13)$$

As volume and weight do not appear in equation (13) it follows that *in the beginning of fall, acceleration is independent of volume or weight of the falling body.* It is influenced solely by the density of the body.

5. The acceleration g_2 becomes zero, and the velocity therefore uniform, when

$$\frac{g}{\delta}(\delta - 1) = \frac{3}{4}c\frac{v^2}{d\delta} \qquad (14)$$

In that case, uniform velocity,

$$v = \sqrt{\frac{4gd}{3c}}\ (\delta - 1)\ \text{ft. per sec.} \qquad (15)$$

Writing

$$\sqrt{\frac{4g}{3c}} = K$$

gives

$$v = K\sqrt{d(\delta - 1)} \qquad (16)$$

Where K is an empirical coefficient, depending on the shape and certain other features of the particle.

This is Rittinger's basic equation for the uniform velocity with which a body of average diameter d and density δ falls through water after its acceleration has become zero. Theoretically this occurs only after the lapse of infinite time; practically, this state is closely approached during the very first instants of fall. According to Rittinger's intricate equations, the velocity of 16-mm. galena, pyrite, and quartz grains becomes practically uniform toward the end of the first second of fall, whereas 1-mm. particles acquire uniform velocity after the lapse of one-quarter second.

More recent experiments have demonstrated that mineral particles falling in water do not strictly obey a universal law, as expressed by equation (16), but that within certain limits of size they follow one law, and within other limits another law.[10] A size may ultimately be reached at which the particles remain suspended for an indefinite time.

FALL OF A BODY IN STILL WATER DURING A CONSIDERABLE TIME INTERVAL

Mineral particles are said to be "equal falling" when they fall ultimately with equal velocities. Assuming uniform motion, and designating by d_1, δ_1, v_1, and by d_2, δ_2, v_2, the diameters, densities, and velocities of two mineral particles, then from (16)

$$v_1 = K_1\sqrt{d_1(\delta_1 - 1)} \qquad (17)$$

$$v_2 = K_2\sqrt{d_2(\delta_2 - 1)} \qquad (18)$$

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[10] As explained by Richards ("Text Book of Ore Dressing," 268) this change occurs at a diameter of about 0.25 mm. (for quartz), coarser grains falling in accordance with Rittinger's law, while finer ones follow more closely Stokes' law of viscous resistance, namely, $v = K(\delta - 1)\ d^2$.

If the two particles are "equal falling," that is, if $v_1 = v_2$, and if the particles also correspond somewhat in shape, then

$$\sqrt{d_1(\delta_1 - 1)} = \sqrt{d_2(\delta_2 - 1)} \qquad (19)$$

$$d_1(\delta_1 - 1) = d_2(\delta_2 - 1) \qquad (20)$$

$$\frac{d_1}{d_2} = \frac{\delta_2 - 1}{\delta_1 - 1} \qquad (21)$$

If the liquid has a density Δ instead of 1, the equation becomes

$$\frac{d_1}{d_2} = \frac{\delta_2 - \Delta}{\delta_1 - \Delta} \qquad (22)$$

For example, consider equal-falling grains of quartz ($\delta_1 = 2.6$) and galena ($\delta_2 = 7.5$).

First, falling in air:

$$\frac{d_1}{d_2} = \frac{\delta_2}{\delta_1} = \frac{7.5}{2.6} = 3 \text{ (nearly)}$$

Second, falling in water:

$$\frac{d_1}{d_2} = \frac{\delta_2 - 1}{\delta_1 - 1} = \frac{6.5}{1.6} = 4 +$$

Third, falling in an artificial quicksand, as described by Chance,[11] composed of 48 parts of water and 52 parts of fine quartz sand, the apparent density of the fluid being 1.83:

$$\frac{d_1}{d_2} = \frac{\delta_2 - \Delta}{\delta_1 - \Delta} = \frac{5.67}{0.77} = 7.4 \text{ (nearly)}$$

Since the separation of equal-falling (sorted or classified) grains by subsequent table concentration is rendered easier by increasing the disparity of their average diameters, the preceding calculations show quite forcibly the theoretical advantage of wet over dry concentrating methods. That is, the average diameters of two equal-falling and medium-sized particles are inversely proportional to their effective densities.

That the above simple relation does not hold for very large nor for very small particles is shown by Richards,[12] whose experiments indicate that the "equal-falling ratio" of quartz and galena is about 6:1 for coarse grains, 4:1 for medium grains, and 2:1 for fine grains.

FALL OF A BODY IN STILL WATER DURING A VERY SHORT TIME INTERVAL

When the body starts from rest, its velocity v being zero, the resistance P_a of the water, equation (8), is likewise zero, and the falling motion is

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[11] T. M. Chance, *Trans.* (1918) **59**, 263.
[12] "Text Book of Ore Dressing," 268; also *Trans.* (1907) **38**, 210.

influenced solely by the force causing acceleration. This force from (4) is:

$$W_1 = \frac{W}{\delta}(\delta - 1) = Mg_1$$

in which, from (5)

$$g_1 = \frac{y}{\delta}(\delta - 1)$$

For very small time intervals t, and very small falling heights h, the increase in velocity, and therefore of the resistance P_a, is so small that the effect of the latter on acceleration is negligible. For practical purposes, the acceleration may therefore be assumed to be constant during the first moments of fall. The velocity in that case is:

$$v = g_1 t = \frac{g}{\delta}(\delta - 1)t \tag{23}$$

Falling height,

$$h = \frac{1}{2}g_1 t^2 = \frac{g}{2\delta}(\delta - 1)t^2 \tag{24}$$

A study of equations (23) and (24), which hold good only for very small time intervals, leads to the following interesting conclusions:

1. During the first instants of fall, the motion of mineral grains in water is practically one of uniform acceleration of the magnitude

$$g_1 = g\,\frac{\delta - 1}{\delta}$$

Since $\frac{\delta - 1}{\delta}$ is always less than unity, the motion is slower than that of the same body outside of water.

2. The falling heights of mineral grains of different diameters and densities depend, in the beginning of fall, solely on the density and not on the volume or diameter of the falling grain. This points to the important principle, utilized in jigging, that with small falling times and small falling heights a separation according to density is possible even when the minerals to be separated have not previously been graded according to size.[14]

3. For two bodies of density δ_2 and δ_1 the difference in falling heights, from (24), is:

$$h_2 - h_1 = \frac{1}{2}gt^2\left[\frac{\delta_2 - 1}{\delta_2} - \frac{\delta_1 - 1}{\delta_1}\right] = \frac{1}{2}gt^2\left[\frac{\delta_2 - \delta_1}{\delta_1\delta_2}\right] = \frac{1}{2}gt^2\,\frac{1 - \dfrac{\delta_1}{\delta_2}}{\delta_1} \tag{25}$$

Obviously this difference, and therefore the ease with which two

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[14] In the practical operation of jigging unsized ore, as explained by Richards, *Trans.* (1894) **24**, 409–486 and (1897) **26**, 1034, suction plays an important part in assisting the separation.

given minerals may be separated, is greater, the larger the difference in their densities.

Calling the ratio $\frac{\delta_1}{\delta_2} = q$, equation (25) becomes

$$h_2 - h_1 = \frac{1}{2}gt^2\left(\frac{1-q}{\delta_2 q}\right) \tag{26}$$

Equation (26) shows that of two pairs of minerals, the density ratio of which is $\frac{\delta_1}{\delta_2} = q$, the difference in falling heights, and therefore the ease of separation, will be greater for the pair in which the density (δ_2) of the denser mineral is the smaller. As shown in the following example, the difference in the falling heights of a coal and a pyrite grain is almost twice as great as that of a quartz and a galena grain, although the density ratio (0.33) is the same for either pair of minerals.

For quartz and galena, $q = \frac{\delta_1}{\delta_2} = \frac{2.5}{7.6} = 0.33$

$$h_2 - h_1 = \frac{1}{2}gt^2\left(\frac{1 - 0.33}{7.6 \times 0.33}\right) = \frac{1}{2}gt^2 \times 0.267$$

For coal and pyrite,

$$q = \frac{\delta_1}{\delta_2} = \frac{1.6}{4.9} = 0.33-$$

$$h_2 - h_1 = \frac{1}{2}gt^2\left(\frac{1 - 0.33}{4.9 \times 0.33}\right) = \frac{1}{2}gt^2 \times 0.413$$

If the minerals are "equal falling" in water, then from (21)

$$\frac{d_1}{d_2} = \frac{\delta_2 - 1}{\delta_1 - 1}$$

From (24)

$$h_1 = \frac{1}{2}gt^2\frac{\delta_1 - 1}{\delta_1} = \frac{1}{2}gt^2\left[\frac{d_1(\delta_1 - 1)}{d_1\delta_1}\right]$$

and

$$h_2 = \frac{1}{2}gt^2\left[\frac{d_2(\delta_2 - 1)}{d_2\delta_2}\right]$$

From (21)

$$d_1(\delta_1 - 1) = d_2(\delta_2 - 1) \tag{27}$$

Hence

$$\frac{h_1}{h_2} = \frac{d_2\delta_2}{d_1\delta_1}$$

or

$$h_1 d_1 \delta_1 = h_2 d_2 \delta_2 \tag{28}$$

Designating by d_1 and δ_1 the diameter and density of the lighter mineral, then

$$d_1 > d_2$$

Adding the equal quantities of (27),

$$d_1 + d_1(\delta_1 - 1) > d_2 + d_2(\delta_2 - 1)$$

or

$$d_1\delta_1 > d_2\delta_2$$

Hence, in order that equation (28) may hold good,

$$h_1 < h_2$$

This points to the useful phenomenon that of two equal-falling minerals, the denser falls, at the beginning of motion, in advance of the less dense one. In the last analysis, this is due to the fact that of two equal-falling minerals, the less dense one has a greater absolute weight than the denser one. A quartz grain, for instance, that is equal-falling with a galena grain weighs 22 times as much as the galena grain. In other words, its mass is 22 times greater and therefore offers 22 times the resistance (inertia) to motion during the period of acceleration, at the beginning of fall. Consequently it lags behind the galena grain during that period, and catches up with it only after the acceleration has become practically zero.

The phenomenon may be, and is being, utilized in practice to make a separation according to density by allowing the particles to fall through very small heights during very small time intervals.

Behavior of a Solid Body in an Ascending Current of Water

A. *In a State of Suspension*

A solid body introduced into an ascending current of water meets the impact of the ascending water. At a certain velocity (v_p) of the water, the body assumes a state of "suspension," that is, it will neither rise nor fall.

Let

$$W = \text{absolute weight of the body}$$
$$\delta = \text{its density}$$
$$d = \text{its diameter.}$$

Then,

$$\frac{W}{\delta} = \text{buoyancy (loss of weight when submerged in water)}$$

and its effective weight,

$$W - \frac{W}{\delta} = \frac{W}{\delta}(\delta - 1), \text{ or absolute weight diminished by buoyancy.}$$

The resistance of the water, ascending with a velocity (v_p) is, from (8)

$$P_e = c\,\frac{v_p^2}{2g}\,As$$

For equilibrium, that is, for suspension,

$$\frac{W}{\delta}(\delta - 1) = c\frac{v_p^2}{2g}As \qquad (29)$$

For a spherical body,

$$A = \frac{\pi}{4}d^2$$

and

$$W = \frac{\pi}{6}d^3\delta s$$

Substituting in (29) and solving for v_p,

$$= \sqrt{\frac{4g}{3c}d(\delta - 1)} \qquad (30)$$

Comparing this with equation (15) leads to the following conclusions:

1. In order that a body may become "suspended" in an ascending current of water, the latter must have a velocity equal to the uniform velocity that the body would acquire when falling through still water.

2. To keep suspended two bodies of diameters d_1 and d_2, and densities δ_1 and δ_2, the velocities of ascending currents of water must be:

$$v_p' = \sqrt{\frac{4g}{3c}d_1(\delta_1 - 1)}$$

and

$$v_p'' = \sqrt{\frac{4g}{3c}d_2(\delta_2 - 1)}$$

or

$$\frac{v_p'}{v_p''} = \sqrt{\frac{d_1(\delta_1 - 1)}{d_2(\delta_2 - 1)}}$$

3. For "equal-falling" bodies, equation (20),

$$d_1(\delta_1 - 1) = d_2(\delta_2 - 1)$$

whence,

$$v_p' = v_p''$$

That is, "equal-falling" bodies require the same velocity of ascending current in order to remain suspended. Equal-falling particles will therefore not separate according to density in such a current.

4. If the bodies have equal diameters but different densities, then:

$$\frac{v_p'}{v_p''} = \sqrt{\frac{\delta_1 - 1}{\delta_2 - 1}}$$

That is, of two equal-sized bodies, the denser requires a greater velocity of ascending current than the less dense, to keep it in suspension. A current which suspends the denser particle will cause the less dense to rise and will therefore permit a separation according to density.

5. If the bodies have the same density but different diameters, then:

$$\frac{v_p'}{v_p''} = \sqrt{\frac{d_1}{d_2}}$$

That is, the larger one requires a greater velocity of ascending current than the smaller. A current suspending the larger particle will cause the smaller one to rise, thus making possible a separation according to size.

In general, an ascending current of water offers a means for separating bodies of equal size according to density, and bodies of equal density according to size.

Since a machine operating with a continuous ascending current would involve mechanical complications in the withdrawal of products separated according to density, certain types of machines (Harz jigs) substitute an intermittent current to which the necessary velocity is imparted by a plunger. The minerals to be separated rest on a screen and are subjected to the current. The ascending lighter particles fall back after each interruption of the current and collect in the upper region of the bed; while the heavier minerals, which have merely been brought into a state of suspension, remain on the screen or reach it after a certain number of pulsations.

B. *In a State of Motion*

If the ascending velocity (v_a) of the water current is greater than the velocity (v_s) required to hold the immersed body in suspension, the body itself will rise.

Let v be the velocity of the rising body; then the water will act upon that body with a residual velocity ($v_a - v$) and the pressure exerted upon the body will be, from (8),

$$P_e = c \frac{(v_a - v)^2}{2g} As$$

This pressure is opposed by the downward-acting absolute weight of the body, diminished by buoyancy, which from (4) is:

$$W_1 = \frac{W}{\delta}(\delta - 1)$$

The residual force F acting in an upward direction, and imparting to the mass M an acceleration g_2, is:

$$F = Mg_2 = \frac{W}{g} g_2 = c \frac{(v_a - v)^2}{2g} As - \frac{W}{\delta}(\delta - 1)$$

Substituting for A and W the values corresponding to a spherical body, and solving for g_2, gives

$$g_2 = \frac{3c(v_a - v)^2}{4d\delta} - g \frac{\delta - 1}{\delta} \tag{31}$$

Writing

$$C = \frac{3c}{4(\delta - 1)g}$$

then

$$g_2 = g \frac{\delta - 1}{\delta} \left[C \frac{(v_a - v)^2}{d} - 1 \right] \tag{32}$$

A study of equations (31) and (32) leads to the following conclusions:

1. For $v_a = 0$, that is, for still water, the acceleration becomes negative. Instead of rising, the body will fall, that is:

$$g_2 = -\frac{3cv^2}{4d\delta} - \frac{g}{\delta}(\delta - 1)$$

(See analogous positive value, equation (12).)

2. The acceleration is positive, that is, the body will rise, when

$$C\frac{(v_a - v)^2}{d} > 1$$

or

$$(v_a - v) > \sqrt{\frac{d}{C}}$$

or

$$(v_a - v) > \sqrt{\frac{4g}{3c}d(\delta - 1)}$$

or from (30)

$$(v_a - v) > v_p$$

That is, when the residual velocity of the water current is greater than the velocity required to keep the body suspended, the body will rise.

3. The acceleration becomes zero, and the velocity of the ascending body becomes uniform, when

$$C\frac{(v_a - v)^2}{d} = 1$$

or

$$(v_a - v) = \sqrt{\frac{d}{C}}$$

or

$$(v_a - v) = \sqrt{\frac{4g}{3c}d(\delta - 1)}$$

or (from 30)

$$(v_a - v) = v_p$$

Whence, uniform velocity of ascending body $v = (v_a - v_p)$ (33)

In reality this velocity reaches the limiting value $(v_a - v_p)$ only after the lapse of infinite time. It is quickly approached, however, and after a comparatively short time the velocity may be assumed to have become practically uniform.

MOTION OF A SOLID BODY IN A DESCENDING CURRENT OF WATER

During the first period of fall, the velocity of a body introduced into a descending stream of water rapidly increases from zero to the velocity of the water. Thereafter, the body descends faster than the water. Consequently it meets a resistance which causes the acceleration to decrease so rapidly that after a very short time it becomes nearly zero, and the velocity of the body practically uniform. This happens in what may be called the second period of fall.

A. *First Period of Fall*

Starting from rest, the body begins to descend with a velocity v depending partly on its weight and partly on the velocity v_d of the descending water. Up to the end of the first period the water velocity is greater than that of the body, that is, $v_d > v$.

As a consequence the descending current exerts upon the falling body a downward pressure corresponding to a residual velocity $(v_d - v)$. This pressure is, from (8)

$$P_a = c \frac{(v_d - v)^2}{2g} A s$$

P_a, added to the absolute weight of the body, diminished by buoyancy, gives the total force F that imparts to the mass M of the body an acceleration g_2.

$$F = M g_2 = \frac{W}{g} g_2 = \frac{W}{\delta} (\delta - 1) + c \frac{(v_d - v)^2}{2g} A s$$

Substituting for A and W values corresponding to a spherical body, and solving for g_2, gives

$$g_2 = g \frac{\delta - 1}{\delta} + \frac{3c(v_d - v)^2}{4d\delta}$$

A comparison with equation (12) shows that in a descending current of water the acceleration and therefore the effect of inertia is considerably greater than in still water, increasing with the velocity v_d of the water. Writing

$$C = \frac{3c}{4(\delta - 1)g} \tag{35}$$

then,

$$g_2 = g \frac{\delta - 1}{\delta} \left[1 + C \frac{(v_d - v)^2}{d} \right] \tag{36}$$

Evidently, the acceleration is greatest when $v = 0$, that is, in the beginning of motion, and it decreases as v increases. For $v = v_d$ (at end of first period)

$$g_2 = g \frac{\delta - 1}{\delta} = g_1$$

This is the acceleration that would prevail in still water if there were no resistance. This actually happens at the instant the body's velocity equals that of the descending water (at the end of the first period).

First Instants of the First Period.—At the very beginning of motion, the falling velocity v of the body is zero, and from (36)

$$g_2 = g \frac{\delta - 1}{\delta} \left[1 + C \frac{v_d^2}{d} \right]$$

For the first short instants of fall this acceleration may be assumed to be constant; hence height of fall

$$h = \frac{1}{2} g_z t^2$$

or

$$h = \frac{1}{2} g t^2 \frac{\delta - 1}{\delta} \left[1 + C \frac{v_d^2}{d} \right] \tag{37}$$

Evidently, the greater v_d, the greater the falling height of any single body.

Introducing value of C from (35) in (37) gives

$$= \frac{1}{2} g t^2 \left[\frac{\delta - 1}{\delta} + \frac{3 c v_d^2}{4 g d \delta} \right] \tag{37a}$$

or

$$h = \frac{1}{2} g t^2 \left[1 + \frac{3 c v_d^2 - 4 g d}{4 g d \delta} \right] \tag{38}$$

The actual magnitude of the falling height h clearly depends on whether

$$3 c v_d^2 > 4 g d \tag{39}$$

or

$$3 c v_d^2 < 4 g d \tag{40}$$

or

$$3 c v_d^2 = 4 g d \tag{41}$$

(a) *Equal-sized Particles of Different Densities.*—For case (39), the second member of the factor within the brackets of equation (38) is a positive quantity, increasing as δ decreases. Evidently, at high velocities of the descending water, the less dense of two equal-sized particles will, at the beginning of motion and during the same time interval, fall through a greater distance than the denser one, thereby diminishing the final difference in falling heights. This demonstrates that too great a velocity of the descending water (suction) is harmful in the separation of equal-sized particles by jigging.

For case (40), the second member of the same factor is a negative quantity which increases as δ decreases. In this case the denser of two equal-sized particles will, at the beginning of motion and during the same time interval, fall through a greater distance than the less dense one, thereby increasing the ultimate difference in falling height. A smaller velocity of the descending current (suction) is therefore favorable in the separation of equal-sized particles.

If δ_1 is the specific gravity of the denser and δ_2 that of the less dense of two equal-sized particles of diameter d, then their falling heights, from (38), will be:

$$h' = \frac{1}{2} g t^2 \left[1 + \frac{3 c v_d^2 - 4 g d}{4 g d \delta_1} \right]$$

and

$$\frac{1}{2} g t^2 \left[1 + \frac{3 c v_d^2 - 4 g d}{4 g d \delta_2} \right]$$

The difference in falling height is:

$$h' - h'' = \frac{t^2 (\delta_1 - \delta_2)}{8 d \delta_1 \delta_2} [4 g d - 3 c v_d^2] \qquad (42)$$

Obviously, this difference is greatest, and the separation therefore easiest, when $v_d = 0$, that is, in the absence of suction. This fact was recognized during the earliest stages of scientific ore dressing, and led to the invention of the *setzpumpe*, described by Rittinger. This device was a jig having no suction and it was used for treating screen-sized material. A more modern type of suctionless jig is the pulsator jig, invented by Richards and described in his textbook.[14]

For case (41) the falling height h becomes

$$h = \frac{1}{2} g t^2$$

In this case two equal-sized particles, irrespective of their density, fall through equal distances during the same time interval. This is analogous to the fall in vacuo.

(b) *Particles of Equal Density but of Different Size.*—In any of the cases (39), (40) and (41), the smaller particles fall in advance of the larger ones.

(c) *Equal-falling Particles.*—From equation (37a) the falling heights are:

$$h_1 = \frac{1}{2} g t^2 \left[\frac{d_1 (\delta_1 - 1)}{d_1 \delta_1} + \frac{3 c v_d^2}{4 g d_1 \delta_1} \right]$$

$$h_2 = \frac{1}{2} g t^2 \left[\frac{d_2 (\delta_2 - 1)}{d_2 \delta_2} + \frac{3 c v_d^2}{4 g d_2 \delta_2} \right]$$

in which

$$d_1 (\delta_1 - 1) = d_2 (\delta_2 - 1)$$

Let d_1 and δ_1 designate the diameter and density of the less dense body; then $d_1 > d_2$ and $h_1 < h_2$.

That is, at the beginning of motion, the denser of two equal-falling bodies falls in advance of the less dense, and much faster than in still water. For the separation of equal-falling minerals, a descending current of water (suction) is therefore favorable.

B. *Second Period of Fall*

At the end of the first period the body has acquired the velocity of the descending water. Immediately after, the velocity of the falling body exceeds that of the water and conditions arise analogous to those in still water. The body is now acted upon by both a downward force $\frac{W}{\delta} (\delta - 1)$,

[14] "Text Book of Ore Dressing," 291–295.

which is its absolute weight, diminished by buoyancy, and by an upward force equal to a water-resistance, that corresponds to a residual velocity $(v - v_d)$. From (8) this resistance is:

$$P_a = c \frac{(v - v_d)^2}{2g} As$$

The total residual force F acting downward upon the body is therefore

$$F = \frac{W}{\delta} (\delta - 1) - c \frac{(v - v_d)^2}{2g} As = Mg_2 = \frac{W}{g} g_2$$

Deductions analogous to those by which equation (36) was derived give for the acceleration during the second period

$$g_2 = g \frac{\delta - 1}{\delta} \left[1 - C \frac{(v - v_d)^2}{d} \right] \tag{43}$$

Evidently the acceleration is greatest when $v = v_d$, that is, in the very first instants of the second period.

The acceleration becomes practically zero and the falling velocity v of the body uniform, when

$$C \frac{(v - v_d)^2}{d} = 1$$

or

$$v - v_d = \sqrt{\frac{d}{C}}$$

whence

$$v = \sqrt{\frac{d}{C}} + v_d$$

Substituting value of C from (35), uniform velocity

$$v = \sqrt{\frac{4gd(\delta - 1)}{3c}} + v_d \tag{44}$$

A comparison with equation (15) shows that the first member of the right-hand expression of (44) is equal to the uniform velocity that the body ultimately acquires when falling through still water. Adding to this the velocity of the descending water, gives the ultimate uniform velocity of the body when falling in a descending current of water.[15]

BEHAVIOR OF SOLID BODIES IN A STREAM OF WATER FLOWING DOWN AN INCLINE

(*Table Concentration*)

Imagine two small mineral cubes a and b, Fig. 1, of approximately the same weight but of different densities, placed on an inclined plane which

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[15] For experimental data bearing out the practical conclusions here developed mathematically, see Richards, *Trans.* (1894) **24**, 409; (1896) **26**, 3; and "Text Book of Ore Dressing," 309–318. Also R. P. Jarvis, *Trans.* (1908) **39**, 451.

is horizontal in the direction at right angles to the slope. Let the inclination be so slight that the gravity-pull of the cubes will be insufficient to move them down the plane against the frictional resistances, which will be the same for both cubes if they have like coefficients of friction.

A sheet of water flowing down the incline, and deep enough to cover both cubes, will attack the less dense and larger cube b with a greater total force than cube a. Obviously, by adjusting the water supply and the inclination of the table, a point may be reached at which the gravity-pull plus the water pressure will become sufficient to move the larger but remain insufficient to move the smaller cube. This constitutes the basic principle of concentration on film tables, as exemplified by buddles, round tables, canvas or "rag" plants, and the finishing surface
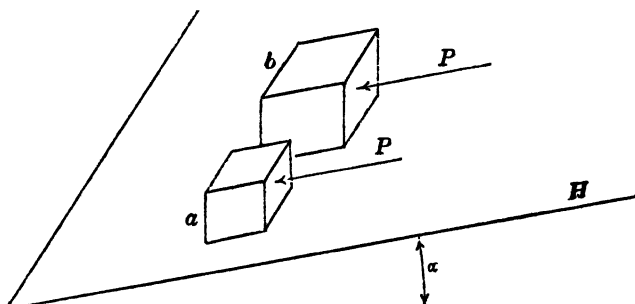
Fig. 1.

of the Wilfley table; in the case of vanners and the various all-riffled tables, other factors are brought into play.

If now the size of cube b is increased by a uniform lengthening of its edges, its volume b^3, and therefore its weight and frictional resistance, will increase with the third power of its edge, whereas the water pressure will increase only with the second power of its edge. Obviously, increasing the size of the cube will demand a constantly increasing water pressure to move it down the incline against the increased frictional resistance, and a point will eventually be reached at which the pressure becomes sufficient to move the smaller cube a also.

It appears then, that in order to effect a separation by this method, the size of the larger and less dense cube must not exceed a limit beyond which the water pressure necessary to move it will likewise move the smaller cube, or move it with the same velocity. It is of importance to ascertain this limit so that the minerals to be separated, prior to being fed to the table, may be graded accordingly.

Let H, Fig. 1, represent the plane, inclined at an angle α from the horizon.

Let a = edge, in feet, of a mineral cube of density δ_a.

b = edge, in feet, of a mineral cube of density δ_b.

f = coefficient of friction of the cubes on the table surface.

s = specific weight of water (1 cu. ft. = 62.5 lb.).

P = water pressure in lb. per sq. ft.

Let the cubes be covered by a sheet of water of depth b, flowing down the incline. Then:

$$\text{Total water pressure on cube } a = Pa^2 \text{ lb.} \tag{45}$$

$$\text{Total water pressure on cube } b = Pb^2 \text{ lb.} \tag{46}$$

To start the cubes down the incline, this pressure plus the gravity pull must be sufficient to overcome the frictional resistance due to the absolute weights of the cubes, diminished by buoyancy. These residual weights are:

For cube a, $sa^3\delta_a - sa^3 = sa^3(\delta_a - 1)$ lb.

For cube b, $sb^3(\delta_b - 1)$ lb.

Gravity pull of cube $a = sa^3(\delta_a - 1) \sin \alpha$

Gravity pull of cube $b = sb^3(\delta_b - 1) \sin \alpha$

Frictional resistance of $a = fa^3s(\delta_a - 1) \cos \alpha$

Frictional resistance of $b = fb^3s(\delta_b - 1) \cos \alpha$

For equilibrium, water pressure + gravity pull = frictional resistance.

$$Pa^2 + sa^3(\delta_a - 1) \sin \alpha = sfa^3(\delta_a - 1) \cos \alpha \tag{47}$$

$$Pb^2 + sb^3(\delta_b - 1) \sin \alpha = sfb^3(\delta_b - 1) \cos \alpha \tag{48}$$

From (47),

$$P = sa(\delta_a - 1)(f \cos \alpha - \sin \alpha)$$

From (48),

$$P = sb(\delta_b - 1)(f \cos \alpha - \sin \alpha)$$

Whence

$$sa(\delta_a - 1)(f \cos \alpha - \sin \alpha) = sb(\delta_b - 1)(f \cos \alpha - \sin \alpha)$$

or,

$$a(\delta_a - 1) = b(\delta_b - 1)$$

or,

$$\frac{a}{b} = \frac{\delta_b - 1}{\delta_a - 1} \tag{49}$$

That is, a given current of water flowing down an incline will simultaneously start mineral cubes of different densities on their way down the incline, when their edges are in the inverse ratio of their densities, each diminished by one.

For example, let the minerals be galena (δ_a = 7.5) and quartz (δ_b = 2.5). To satisfy equation (49), the ratio of the edges of the cubes must be

$$\frac{a}{b} = \frac{\delta_b - 1}{\delta_a - 1} = \frac{2.5 - 1}{7.5 - 1} = \frac{1.5}{6.5} = \frac{1}{4}$$

or the same as the equal-falling ratio in water.

On the above assumptions, a 1-mm. galena and a 4-mm. quartz cube, if placed on an inclined table, would both be washed down the incline by a current strong enough to move the quartz cube, a fact which would seem to prevent a separation by such a method. Moreover, the similarity of equation (49) with equation (21) seems to justify the inference that classified material, in which the individual mineral particles are equal falling, does not lend itself to a satisfactory separation on film concentrating tables, because a water current strong enough to move the larger and less dense particles would also move the smaller and denser ones, presumably with the same velocity. The difficulty of separation evidently increases as the densities and therefore the diameters of the particles approach equality.

The water pressure per unit area, however, is not uniform, as assumed above. It varies with the velocity of the water in the successive layers of the stream flowing down the incline. Close to the table deck the velocity is practically zero, increasing to a maximum near the surface. Thus the larger cube b is exposed to a much greater total pressure than the smaller cube a because, in addition to presenting a larger area to the stream of water, it also reaches into the upper strata of this stream where the velocity is much greater than near the table deck.

According to a law of hydraulics, the velocity increases approximately with the square of the distance from the bottom of the stream, which, in this instance, is the table deck. A stream of water attacking a 1-mm. galena cube at its center with a pressure P_a per unit area will therefore attack a 4-mm. quartz at its center with a unit pressure $P_b = \left(\frac{2}{\frac{1}{2}}\right)^2 P_a$, which is 16 times larger than P_a. And since the surface exposed by the quartz cube is 16 times larger than that of the galena cube, the total water pressure on the quartz cube is nearly 300 times as great as that exerted on the galena cube.

This indicates that classified material is not only eminently suited for separation on concentrating tables, but that, when the difference in densities is as great as in the case of quartz and galena, close sizing or classifying of the feed is not essential to permit a satisfactory separation by film methods. Moreover, it seems reasonable to expect greater ease of separation when the less dense minerals are somewhat larger than demanded by equation (49), as in that case they would be attacked by a still greater water pressure and therefore carried farther away from the smaller but denser minerals. This is accomplished by hindered-settling classifiers, first introduced by Richards and now almost universally employed for preparing table feed.[16]

[16] Richards: *Trans.* (1897) **27**, 76; (1907) **38**, 556; (1908) **39**, 303; (1910) **41**, 396; "Text Book of Ore Dressing," 220–236, 262–276; "Mining Engineers' Handbook," 1662–1664.

CONCLUSION

The fact that ideal conditions, such as had to be assumed in the foregoing study, are hardly ever met with in practice by no means invalidates the general conclusions drawn. Actual conditions, however, modify the theoretical behavior of bodies in still or moving water in a manner that can be ascertained only by experimental methods, as applied by those investigators to whom reference has been made.

Although the results of experimental methods have demonstrated limitations in the application of theory to practice, as well as discrepancies in numerical values obtained by the two methods, they have, in general, corroborated theoretical deductions. They also have made it clear that for an explanation of experimental results, and for an intelligent application of these results to practical problems, a thorough grasp of basic principles is an essential prerequisite.

DISCUSSION

E. V. DAVELER, Butte, Mont. (written discussion).—This paper amplifies and extends the valuable contributions of Richards and Rittinger. The author has reduced Richards' complex mathematics to what he terms "a more elementary treatment of the subject." The paper shows fully the necessity of close sizing in gravity concentration methods utilizing jigs, which fact had previously been emphasized by Professor Richards. The plan of close sizing before jigging has been utilized in various wet concentration plants and where followed with a proper ratio of screen sizes, high recoveries and grade of concentrates have been made.

The author has worked out the mathematics of the behavior of solid bodies in a stream of water flowing down an incline, or the action taking place during table concentration, indicating the necessity of classification before table concentration where the densities of the metals to be separated are very close and bringing out also that where the difference in densities is great, as in the case of quartz and galena, classification is not essential.

CHARLES E. LOCKE,* Cambridge, Mass. (written discussion).—This paper is an admirable collection of the laws and principles covering gravity operation in ore concentration and should fill a need some of us have felt for the gathering of all this information in one place. I only wish that the author had extended the scope of his paper so that it included a wider field, particularly the laws covering settling of very fine particles or sedimentation, which have now become of so much importance to the ore dresser in connection with settling operations for slimes and for flotation products. Perhaps he may feel willing to do this in a later paper and at the same time consider the possibilities of including centrif-

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ugal, pneumatic, and other special concentration processes; also to formulate the laws for flotation on which so much has been written.

The value of such papers lies chiefly in giving a clear understanding of what should happen with ideal particles under ideal conditions. Although the author does not so state, the reader will understand that departures from ideal conditions in the way of irregularities in size, shape, and other characteristics of the particles will give separations in ore dressing that differ materially from the separations that could be obtained under the assumed ideal conditions.

BLAMEY STEVENS, New York, N. Y. (written discussion).—This paper appears to be a study of classification rather than of concentration. The paragraphs on "behavior of solid bodies in a stream of water flowing down an incline" do not in any manner illustrate the principles of table concentration. The oscillating motion of the vanner or table is the all-important factor in concentration.

In the case of the vanner, the transverse accelerations of velocity are large enough to cause the belt to slide to and fro with respect to the grains of less density, but small enough so that the grains of greater density do not part company with the belt. In the case of the table, the acceleration is more in one direction than in the return or opposite direction. The larger acceleration allows the table to travel with respect to the grains of greater density, so that the latter travel gradually along the table.

Expressed mathematically, with the notation used by the author, the frictional resistance is $fa^3s(\delta - 1)$. At the point or limit of sliding the frictional resistances will be just balanced by the force due to acceleration, viz., $\frac{\alpha}{g} a^3 s \delta$; where α is the acceleration of the vanner or table and g that of gravity (viz., 32 ft. or 981 cm. per sec. per sec.). The grain of material is considered to be accelerated in the water without fluid friction. Strictly speaking, the above force is somewhat small even with a frictionless fluid because some water motion is also accelerated. As there are, however, other modifications in practice we may neglect niceties and equate as follows: $\frac{\alpha}{g}\delta = f(\delta - 1)$.

Therefore the lesser acceleration lies between

$$\alpha < fg \frac{\delta_a - 1}{\delta_a} \text{ and } \alpha > fg \frac{\delta_b - 1}{\delta_b}$$

For galena, $\quad \frac{\delta_a - 1}{\delta_a} = \frac{7.5 - 1}{7.5} = 0.87.$

For quartz $\quad \frac{\delta_b - 1}{\delta_b} = \frac{2.5 - 1}{2.5} = 0.6.$

So that in this case there is a great deal of latitude for the determination of α. When the density is the same as water (or $\delta_b = 1$) we get

$\alpha = 0$ as a limiting case.   The greater of the two accelerations of a table must be

$$\alpha > fg\frac{\delta_a - 1}{\delta_a}$$

For minerals of the highest density ($\delta_a = \infty$), we get $\alpha = fg$ as a limiting value.

These accelerations only depend on specific gravity and not on size of grain.   This would remain true as long as the grains were accelerated in a perfect fluid, but fluid friction would, of course, depend on the size of the grains.

While the main function of the inclination of the vanner or table is to wash over the quartz that has been loosened from its frictional mooring, it also somewhat modifies the above results, especially if the larger grains stick up above the water level.

The ideal laboratory conditions for the application of the above formulas are a very quick short vibration in a pool of water without flow. The acceleration should change rapidly from a uniform high to a uniform low value of opposite sign.   The foregoing are not the best conditions for the treatment of quantities of material.   The necessary flow of water and longer slower oscillations of good practice introduce imperfections of concentration, which may sometimes be minimized by a good classification scheme.

CHARLES D. DEMOND, Anaconda, Mont. (written discussion).—The author strikes the keynote of the situation in his statement that standard "processes of gravity concentration no longer receive the attention they deserve."   Facts of mill and smelter practice, together with the results of both mill and laboratory experiments, indicate the advantage of continuing and extending the investigation of gravity methods.   Operators realize that the high recoveries made possible by flotation do not justify the extra expense of all-sliming when most of the values can be saved by cheaper methods.   There is a growing tendency to save as much as possible by jig and table concentration while still using flotation for additional profit.

Years ago attempts were made to discard waste from comparatively coarse ore by both hand picking and jigging.   Some of these efforts failed, even where the ores seemed to promise success, because of poor preparation of jig feed and the inefficient design of jigs.   The poor preparation of feed causes questions of design of both screens and hydraulic classifiers to arise, also of the capacity of the different classes of machines. Trommels do reasonably good screening of material 1 in. or coarser, but it is extremely difficult to have them do good work with material finer than, say, ¼ inch.   One reason is that the ore does not remain in a sufficiently thin layer except over the small space where it first strikes

the screen; beyond that the natural bank is too deep and is not agitated enough to permit efficient work. The Ferraris screen, which is supported in a horizontal position by sloping hangers, also has shown such satisfactory results that it deserves thorough investigation, particularly for wet screening. The method by which it applies the water is very effective. On account of the method of support the screen moves down sharply on the backward stroke and dips lightly into the surface of the water in a tank. This positively prevents blinding of the screen cloth while causing the ore to progress rapidly; and at the same time it produces an effective agitation which allows the undersize material to pass through the screen as it should. Flat screens for wet work that are not subject to effective action of water over the entire surface have virtually the same defect as trommels: most of the screening is confined to a small area near where the feed first strikes. The Ferraris apparatus can be made very economical of water by using simple mechanical means to remove the products.

In many cases wet screening is far better than dry because the slime is thoroughly removed from the coarser sizes and is treated by suitable methods instead of being lost in the tailing of jigs or tables.

When making an investigation of hydraulic classifiers, several years ago, the writer was confronted with the problem of determining efficiencies. While there is available much data purporting to indicate the quality of work done by hydraulic classifiers, the data consist of sizing tests of the products. But, to determine the efficiency, any apparatus must be tested according to its own principle of action: commercial screens by the most accurate screening we can do and commercial classifiers by the most accurate classification possible. Unlike screens, which place together similar sized pieces of minerals irrespective of their specific gravities, a good classifier necessarily separates minerals of different specific gravities, because of their difference of falling velocity in water, and puts smaller particles of heavy mineral in company with larger pieces of light mineral. Classifiers do, of course, separate the coarser from the finer portions of any one mineral, which accounts for the phrase "hydraulic sizing;" and it is possible that this fact partly explains the practice of testing the classifier products with sieves; but this ignores the radical difference between screen sizing and hydraulic classification.

Without going into details, it may be stated that, by using the hydraulic method for testing the products of a certain mill classifier, the settling efficiencies of the three spigots were, respectively, 53, 70, and 51 per cent. The relation of the first and second spigot products illustrates the fallacy of a screening test, for such a test showed the second spigot material to be coarser than the first, whereas the settling tests showed these two products in their logical relation; the first spigot to be the

coarser, that is more rapid settling, than the second. It has been well said that "no science can progress unless accurate measurements are made of the things dealt with." Methods of investigation must be adapted to the needs of each case.

When discussing hydraulic classifiers, a prominent engineer argued that the seemingly perfect classification of sand in the little bubbling craters of natural springs is practically impossible to attain in mill work, because, he said, that it is altogether out of the question to subject the ore to the necessary number of retreatments. But the success of the flotation process depends largely on repeated treatments. With many ores, a glass-sided cell of a Minerals Separation machine shows a shower of concentrate particles falling out of the froth; and numerous retreatments are necessary for a recovery that is at all satisfactory. It may be questioned if there is a commercial plant anywhere in which a single cell of such a machine, producing finished concentrate, recovers more than 50 per cent. of the values entering it; in many cases this recovery is probably as low as 25 per cent. Why, then, should it be thought impracticable to repeat hydraulic classifying, even in separate chambers? Various observations convince the writer that there are large possibilities for the fruitful investigation of this problem.

The advantage of perfecting the work of screens and of hydraulic classifiers lies not only in better recoveries (and probably in increased capacities of jigs and tables), but in the discarding of waste tailing at much coarser sizes than is usually done. The high cost of fine grinding is well illustrated in a particular case where ¾-in. ore is reduced to 0.2 mm. by two-stage grinding in ball mills. Though the first stage makes a 63 to 1 reduction in size (19 mm. to 0.3 mm.) while the second stage reduces only 1.5 to 1, the second stage uses 6 per cent. more power than the first. The excess power would be even greater if the second stage handled the same tonnage as the first; but the 0.2-mm. undersize is removed between stages.

One kind of waste that should be discarded as early as possible is decomposed feldspar. This material constitutes the gummy portion of slime, and without this material the slime problem would not be such a serious problem. It is true that a large part of this material passes into the slime in the earlier stages of crushing, because it is soft; but there are other portions, not so soft and yet capable of being crushed between a man's thumb nails, which are sometimes found in jig and table feed.

There is need for the investigation of jigs. Table 1 shows how irregular is the distribution of water in some jigs. These measurements were made by fitting thin steel-plate partitions above the screen, to divide the width into three equal portions, and catching the overflow from each division separately. In jig A, Fig. 2, there was a great excess of water on the part nearest the plunger; the middle section had approxi-

mately the right quantity for most rates of total flow; but, in all cases, the side farthest from the plunger was very deficient in water, and that part of the ore bed was "dead." The bed on such a jig swings like a door hinged at one side of the compartment, instead of jigging uniformly over the entire surface. This introduces two harmful conditions: first, the jigging action most favorable to good work occurs on only a small percentage of the total sieve area; and, second, a third of the area is doing no work, which overcrowds the other two-thirds. Even where the conditions are ideal in other respects, serious overcrowding is almost certain to produce a loss of profit, because of high metal losses or extra expense at some other point in the mill.

TABLE 1.—*Distribution of Running Water over Width of Jig Series*

Jig A

Clean Screen and No Bed of Ore								With Bed of Ore							
Portion Nearest Plunger		Middle Portion		Portion Farthest from Plunger		Total		Portion Nearest Plunger		Middle Portion		Portion Farthest from Plunger		Total	
Lb.*	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.
8.4	97.7	0.2	2.3	0.0	0.0	8.6	100	22.4	71.8	8.8	28.2	0.0	0.0	31.2	100
30.3	64.6	14.2	30.3	2.4	5.1	46.9	100	43.9	59.3	28.2	38.1	1.9	2.6	74.0	100
43.1	58.7	23.0	31.3	7.3	9.9	73.4	100	58.0	57.9	35.6	35.6	6.5	6.5	100.1	100
50.2	54.4	31.1	33.6	11.1	12.0	92.4	100	69.5	59.5	40.6	34.7	6.8	5.8	116.9	100
54.6	52.3	35.6	34.2	14.1	13.5	104.3	100	66.1	50.5	56.1	42.8	8.8	6.7	131.0	100
60.0	53.7	36.3	32.5	15.5	13.8	111.8	100								

Jig B

Clean Screen and No Bed of Ore								With Bed of Ore							
Portion Nearest Plunger		Middle Portion		Portion Farthest from Plunger		Total		Portion Nearest Plunger		Middle Portion		Portion Farthest from Plunger		Total	
Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.
22.8	53.5	19.2	45.1	0.6	1.4	42.6	100	35.3	30.7	42.9	37.2	37.0	32.1	115.2	100
51.5	29.2	67.8	38.5	56.7	32.3	176.0	100	55.2	32.8	60.3	35.7	53.1	31.5	168.6	100
99.2	30.9	111.2	34.7	110.3	34.4	320.7	100	89.4	31.5	97.0	34.2	97.1	34.3	283.5	100
								89.3	29.4	108.5	35.8	105.4	34.8	303.2	100
								136.5	31.6	143.4	33.1	152.8	35.3	432.7	100
								128.8	29.6	153.1	35.2	153.1	35.2	435 0	100

* Lb. = pounds of water per minute rising through 1 sq. ft. of sieve.

Jig *B*, in general, has a reasonably uniform distribution of the water, in sharp contrast to jig *A*; this shows the need for thorough investigation of the proportions in jig design. The following figures indicate that in jig *A* the under-sieve partition, between sieve and plunger compartments, was too shallow, and that both plunger and throat were too narrow. Sudden changes in velocity of the water between the plunger and the screen are likely to cause harmful eddies. Lengthening the path, by means of a suitably deep partition, tends to lessen such eddies.

	RATIO BETWEEN DEPTH OF PARTITION AND WIDTH OF SCREEN	RATIO OF PLUNGER TO SCREEN WIDTH	RATIO OF THROAT TO SCREEN WIDTH
Jig A..............................	0.23	0.52	0.46
Jig B..............................	0.33	0.69	0.50

A good deal of work has been done toward perfecting the mechanical features of the Wilfley type table. There has also been some study as to the action in the bed of ore, and the effect on this action of the riffles and

Jig A   Jig B
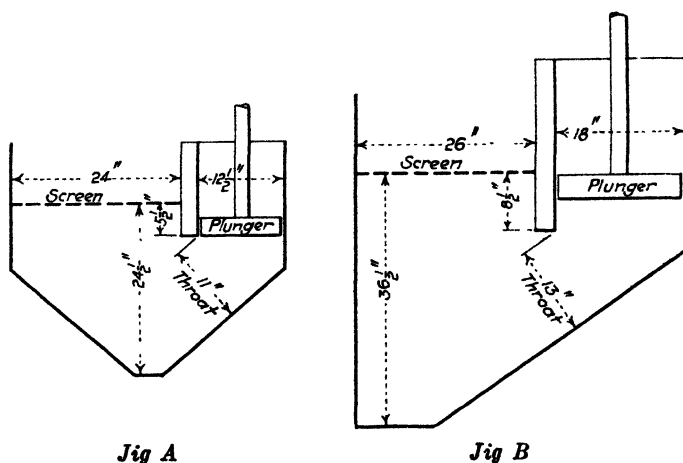
FIG. 2.—JIG *A*, SCREEN AREA 24 × 41 IN.; JIG *B*, SCREEN AREA 26 × 38 IN.

the table motion. These studies need to be continued. It seems to be assumed, by some, that the riffles are merely to guide the concentrate away from the middling. Such evidently was the original purpose but the systematic riffling now used makes a considerable depth of bed, in which the action is entirely different from what takes place in the thin layer of ore on a round table.

Serious losses of value and expense for additional treatment are certain to result from the adoption of unsuitable methods. One fault is the feeding of hydraulic classifier products to jigs. The best results are obtained when ore is prepared for treatment by a principle as far removed

as possible from that on which the treatment machine operates. Screens are based on the principle of exact sizing; hydraulic classifiers act by free or hindered settling; while jigs utilize the principle of hindered settling. Free and hindered settling are one in kind but differ in degree, the ratio between diameters of the light and the heavy minerals being greater under hindered than under free settling conditions. Either of these ratios is necessarily greater than 1.0; but in sizing the ratio is exactly 1.0.
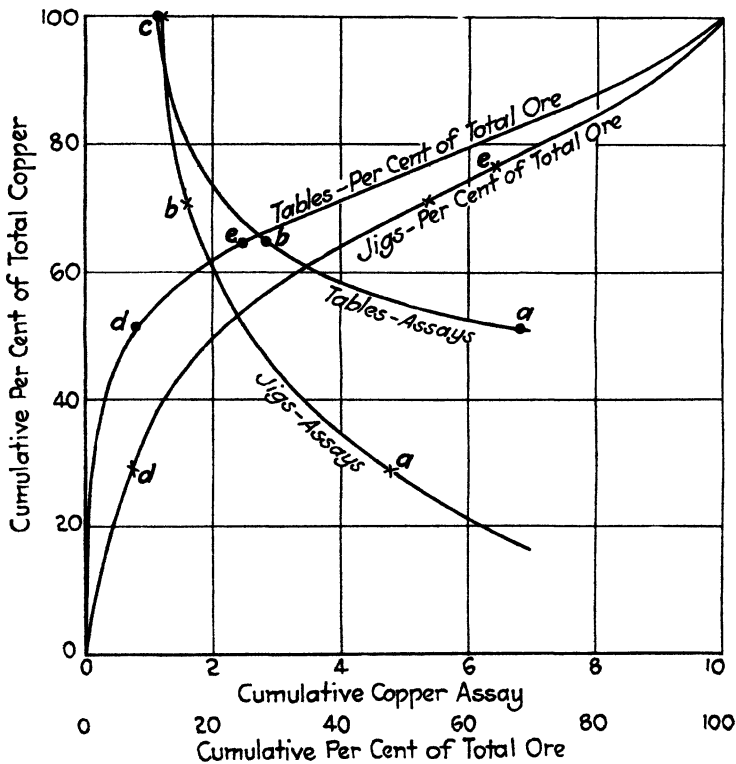
FIG. 3.

Jigs that receive well-sized feed do cleaner work than those fed from classifiers. Classified feed is better adapted to treatment on shaking tables than on jigs. These tables use the principle of crowded settling, for which the ratio of diameters is greater than in the case of either free or hindered settling.

In one case where the feeds ranged from $\frac{1}{16}$ in. to fines and contained about $1\frac{1}{4}$ per cent. copper, the tables made 51 per cent. recovery in a concentrate that assayed 6.85 per cent. copper and 25.4 per cent. insoluble; while the jigs recovered only 29 per cent., in a concentrate that assayed 4.78 per cent. copper and 47.6 per cent. insoluble. This case may be considered unusually favorable to tables; but, in general, when

fed from classifiers they make better recovery in concentrate, or a lower tailing loss, or both, than jigs.

When comparing jig and table mills with flotation plants, the latter are often approved for the simplicity of their flow sheets; but this argument seems to be overdone when we consider the numerous dewatering and thickening tanks, filters, vacuum pumps and driers required. The high recovery from slime, of course, justifies the adoption of flotation in most cases; but a serious defect in perhaps most flotation products is the high percentage of alumina in the insoluble portion of the concentrate. For lead and copper smelting, a certain amount of silica must be left in the concentrate in order to flux the iron originally present as sulfide; but silica combined with alumina is harmful, because it increases the amount of slag, and therefore the total loss of metal in slag. Moreover there is reason to believe that, in slag, alumina often forms spinels and other compounds, which increase the loss of valuable metals. The following figures from one plant show this high alumina:

| | ASSAYS | | RATIO OF Al_2O_3 to SiO_2, PER CENT. |
	Al_2O_3 PER CENT.	SiO_2 PER CENT.	
Ore....................................	7.6	57.4	13
Coarse gravity concentrate.......	1.5	16.2	9
Fine gravity concentrate..........................	1.6	15.4	10
Coarse flotation concentrate......................	4.9	18.4	27
Fine flotation concentrate........................	9.3	19.8	47

In the gravity concentrate, the ratio of alumina to silica is less than in the original ore; but in the flotation concentrate, it is very much greater than in the ore. Similar results are found in other plants.

Gravity methods have been seriously handicapped by the excessive quantity of water often used. Classifiers and concentrators have done much better work when the usual floods of water were available. Here and there, dewaterers and thickeners were used, but, in general, they were avoided because the operators did not wish to bother with them. However, the necessities of modern development in fine grinding and flotation have accustomed mill men to such equipment; and this experience will contribute to the advancement of gravity concentration.

When making comparative tests of two machines, it is nearly impossible to obtain the same grades or same quantities of products, which often makes it difficult to compare results. But graphical methods make it easy to show what the results from either machine would have been under different conditions. Fig. 3 represents the results of the comparative test of jigs and tables mentioned. One pair of curves shows the recoveries in relation to the assays of the products. Point *a* shows the concentrate, *b* the assay of combined concentrate and tailing, and *c* the assay of total concentrate, middling, and tailing. The curves show

that if the tables had made a concentrate having the same assay as that of the jigs, their recovery would have been 56 per cent. of the total value instead of the actual recovery of only 51 per cent.

The other pair of curves shows the relation of the copper recovery to the tonnage of products. Point d shows the percentage of the total ore that appeared as concentrate, e the combined percentage of concentrate and middling, while the 100 per cent. point combines the tailing with the
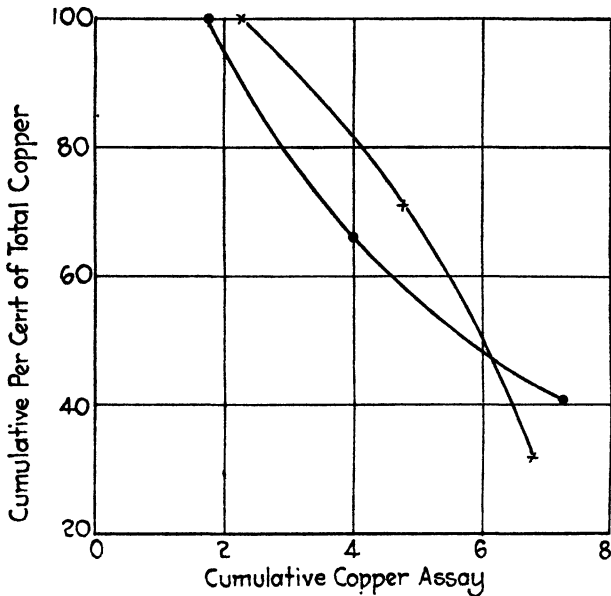
FIG. 4.

other two products. From these curves the percentage of the total copper that would be recovered in any weight that might be taken as concentrate, or for any weight discarded as tailing, can be found.

Fig. 4 shows assay results from two tables operated in comparison with one another. On such plots, one curve is likely to be above the other all the way along, as in Fig. 3; but in Fig. 4 they cross; and in such case the graphic method is even more necessary and useful than in a case like Fig. 3.

ROBERT H. RICHARDS, Boston, Mass. (written discussion).—A paper on mining, ore dressing, or metallurgy that treats the subjects from the purely theoretical point of view, on the basis that the materials considered are perfect, is good as it gives us a view of perfection toward which we should strive. This paper belongs to this class. The author assumes that the grains under consideration are uniform, perfect cubes or perfect spheres, and that included grains, consisting of a portion of

concentrate mineral included in the tailing grain, do not exist. In other words, investigations of this class deal with only part of the facts.

But papers of this class do not do away with, or take the place of, writings that strive to present all the facts and give average results that should be approached, and can be occasionally attained, when we have installed all improvements that are commercially possible. Writings of this class depend on the results of experiment and include in their consideration all variations of form of the grain (cubic, spheric, flat, and needle-like grains) and also every possible included grain, ranging from pure gangue to pure mineral concentrate; they also include the struggle to overcome the infinite variety of eddy currents when the grains are being treated.

In the second paragraph on page 431, the author says that Rittinger reached conclusions by higher mathematics that have remained a sealed book to most of us. I was in that predicament until I worked out a simple derivation of Rittinger's fundamental formula.[17] I do not know whether Rittinger obtained his formula by my method, by Professor Simons', or by some other.

The author then devotes $3\frac{1}{2}$ pages to formulas investigating acceleration and finally reaches Rittinger's fundamental formula. On page 436 he develops formulas for settling of grains of like form in a medium denser than water (the quicksand of T. M. Chance) showing the increased ratio of diameters of equal-falling particles of different gravities. This can be found in my text book (p. 269) in a briefer form and when dealing with average grains.

On pages 436 to 439 the author develops formulas for acceleration and proves that before the ultimate uniform velocity has been reached the small denser mineral grain has a slight advantage over the larger gangue grain even if they are so-called equal falling. This same idea was investigated by me in one of my papers using Rittinger's data; I cannot put my hand on the reference at this time.

On page 439, the author discusses the behavior of a grain in an ascending current of water; and on page 442, in a descending current of water, the inference is that the behavior of the sand and water is relatively the same as if the water was still and the sand was settling in it.

I have found in the tubular classifier[18] that there are eddy currents for 2 or 3 in. up and down, where the mass is descending on one side of the tube while ascending on the other, and that by replacing the steady upward current by an upward current pulsating 300 to 400 times a minute, these adverse eddy currents are entirely eliminated and the work of the tube greatly improved, because at the instant of high velocity, the resis-

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[17] Text Book of Ore Dressing," 264.
[18] "Text Book of Ore Dressing," 270.

tance due to friction is greatly increased according to some power of the velocity, and this causes the rising current to have practically the same velocity in all parts of the section. When discussing classification previous to film sizing on a table (page 446) the author develops formulas and arrives at the conclusion (*a*) that if the velocity of the water is the same at all depths of the film on the surface of the table, the classification will not aid the separation on the surface of the table; but he says (*b*) that the velocity at the top of the film is greater than at the bottom and that therefore the gangue grain is moved down the slope faster than the small heavier grain and, in consequence, the classifier does help the separation on the table. I have only dealt with the position taken in (*b*) in my experimental work.[19]

THEODORE SIMONS (author's reply to discussion).—Fearing that certain omissions in the paper may have given the impression of lack, on the part of the author, of appreciation of the great services of Professor Richards to the profession, or that the author intended to claim credit for discovering facts already established by preceding investigators, he would call attention to both the introductory remarks and the conclusion of the paper.

As explained at the San Francisco meeting, one of the objects of the paper was to demonstrate that, by the aid of pure mathematics, facts can often be predicted that may afterwards be proved by experimental methods; for instance, the harmful effect of suction when jigging closely sized material. The message that the paper intended to convey, particularly to the younger engineers, was that for making scientific investigations, mathematics offers the most helpful, and at the same time the most perfect, tool at our command.

It is a common experience that investigators of the same subject have often come to the same conclusions independently of one another. For this reason the author carefully quoted references known to him. Had he read Professor Richards' paper referred to in his discussion of pages 436 to 439, he would have quoted it just as he quoted T. M. Chance's paper, although he has used and expounded these relations in his lectures for the last twelve years or more.

It would be incorrect to assume that the author "arrived at the conclusion (*a*) that if the velocity of the water is the same at all depths of the film of water, the classification will not aid the separation." What he did say is that "the similarity of equation (49) with equation (21) seems to justify the inference that classified material . . . does not lend itself to a satisfactory separation, etc. His actual conclusion (*b*), after explaining the fallacy of the assumption of uniform velocity throughout

[19] "Text Book of Ore Dressing," 376.

the water film, was, of course, quite different from inference. All of which points to the risk assumed by the writer who attempts to discuss a large subject in a short paper. In this case, however, the risk was amply justified by the fact that it elicited a valuable contribution by the pioneer of scientific ore dressing in America.

Factors Controlling the Capacity of Rock Crushers

By Ernest A. Hersam,* Berkeley, Cal.

The rate of output of a rock crusher is based upon a certain space relation, the calculation requiring that the size and position of the jaws, the principle of motion, and the speed of the machine be known. The result is a calculated maximum output. The purpose of this paper is to develop a method of calculating this maximum output, to judge the general applicability or utility of such a calculation, and, so far as possible, to record calculated values as a basis for the comparison of actual results, or for estimating the capacity of a new crusher or of a standard crusher working under new conditions.

A method for calculating the capacity of crushing rolls, which has been in common use for a long time, is well presented by R. H. Richards.[1] The output is conceived as a ribbon of rock, of which the breadth, thickness and length are governed respectively by the width of the rolls, the space between them, and their peripheral velocity. The weight of the rock thus crushed in a given time can be calculated from its specific gravity, after applying a factor representing the percentage of voids in the hypothetical ribbon.

The calculation of crusher capacity is made in a somewhat similar manner. While the action of rolls is simple, with few irregularities in operating conditions, in rock crushers the material advances intermittently, usually under the force of gravity alone, and is subjected to a variety of variable conditions. The effect of these variable influences upon the calculation of capacity is now to be shown. In this paper, capacity will be discussed quite apart from any consideration of mechanical efficiency; that is, results will be stated not in terms of output per horsepower, but of output for a given machine when operated at a known speed, constructed and adjusted to receive material of a known size, and to yield a product having a definite maximum coarseness.

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  [1] "Ore Dressing," 1, 101; "Text Book of Ore Dressing," 67; "Mining Engineers' Handbook," 1632.

In crushers, of either the jaw or the gyratory type, a fragment of rock passes forward intermittently into the space between the jaws. It is arrested when the jaws begin to close upon it; it is crushed while the jaws are closing; it passes in limited amount from the crushing area while the jaws are opening, and is replaced, at the same time, by rock received from above. The amount of rock discharged during a single cycle of the machine thus depends, primarily, upon the size of the opening at the bottom of the jaws, but is modified by the form and the angle of the jaws, the amplitude of the movement, the state of the surface of the jaws, and by other conditions. The attainment of the calculated capacity depends upon the maintenance of an adequate feed, but, as will be shown, capacity is not greatly affected by any excess of feed. The rock, arrested in its progress, acquires fresh velocity at each cycle of the machine. The propulsive force, usually the force of gravity, is independent of the mechanism of the crushing jaws. The speed of the machine should therefore be maintained in proper relation with the acceleration due to that force. The amount of retardation in the descent of the rock, as the jaws open, depends on the form, surface, angle and position of the jaws, and on the speed of the machine.
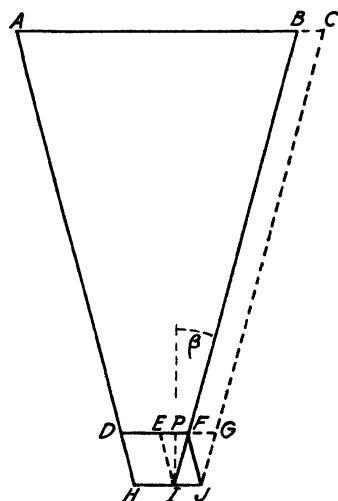
FIG. 1.

Fig. 1 represents a longitudinal vertical section through the jaws of a reciprocating crusher. The construction may be supposed to be such as to provide equal amplitude of movement of the jaw at top and bottom. If applied to a gyratory crusher, it would be a radial vertical section of the space between liner and breaking head; or if applied to a machine of the disk type, it would be a radial section of the space between the crushing cones. In the case of a jaw crusher, the line BI indicates the crushing surface of the movable jaw at its position of closest approach to the fixed jaw AH. The lines BC and IJ are represented as straight lines; in most jaw crushers, the movable jaw describes a short arc, of which BC and IJ would be chords, but measurement along the straight lines does not introduce an appreciable error. The lines FG and IJ are shown as of the same length and the dotted line CJ is made parallel to BI, although not strictly representing the motion of most jaw crushers. The space indicated by the quadrilateral $ABIH$ shows the entire volume between the jaws when closed, while $ACJH$ represents the corresponding space when the jaws are open. Of this space, a certain restricted part $DFIH$

governs the quantity of rock discharged at each cycle of the machine, and may be regarded as the *displacement volume* of the crusher. The rock, which occupies the space $DFIH$ while the jaws are closed, descends between the jaws and falls from the opening HJ as this becomes wider by movement of the jaw from I to J; rock at the level of DF thus reaches the position indicated by HJ. Since the dimension HJ fixes the position of DF, it also determines the amount of rock discharged at each opening of the jaw.

If w is the width of the jaws, the volume S of the crushed and compacted rock discharged at each cycle of the machine is obviously

$$S = \frac{1}{2} (DF + HI) \times PI \times w \tag{1}$$

This may be called the potential displacement volume.

The dimension w, in the case of a jaw crusher, is found by direct measurement. In a gyratory crusher, w is the circumference of a circle concentric with the conical jaw and midway between the liner and the breaking head when nearest together. The dimension PI, or the height through which the rock drops as the jaws open, is calculated from the distance and the angle between the jaws when closed. If β is one-half of this angle,

$$PI = \frac{1}{2} EF \cot \beta = \frac{1}{2} (DF - DE) \cot \beta = \frac{1}{2} (DF - HI) \cot \beta \tag{2}$$

For subsequent computation, let:
$a = DF$ or HJ, the distance between jaws at bottom, when open.
$b = HI$ or DE, the distance between jaws at bottom, when closed.
$c = PI$, the depth of rock replaced at each cycle.
$d =$ total vertical depth of the crushing space, between AB and HI.
$e = AB$, the distance across the top, between jaws, when closed.
Substituting these terms, equation (1) becomes

$$S = \frac{1}{2} (a + b) \times c \times w \tag{3}$$

Equation (2) becomes
$$c = \frac{1}{2} (a - b) \cot \beta \tag{4}$$
Substituting this value of c in equation 3,
$$S = \frac{1}{2} (a + b) \times \frac{1}{2} (a - b) \cot \beta \times w \tag{5}$$
Since
$$\cot \beta = d \div \frac{1}{2} (e - b), \tag{6}$$
$$S = \frac{a + b}{2} \times \left(\frac{a - b}{2} \times \frac{d}{\dfrac{e - b}{2}} \right) \times w \tag{7}$$

or, more conveniently for numerial substitution,

$$S = \frac{a^2 - b^2}{2 (e - b)} \times w \times d \tag{8}$$

In the practical operation of a crusher, it is obvious that certain mechanical irregularities may cause deviations from theoretical values. The causes of these discrepancies may be grouped as follows:

1. Shattering.—A necessary increase above the calculated capacity results from the shattering and disentanglement of the rock during the approach of the jaws upon it. This effect reaches its maximum when a dry rock is of such fine size as to require but a slight reduction to pass through the opening between the jaws. The same result occurs when a highly elastic rock disrupts violently at the instant of fracture.

2. Resistance.—The theoretical output is diminished when the rock fails to drop stably into the displacement space as the jaw of the machine opens. This may be due to friction, or to adhesion of the rock to the jaw liners, and is most pronounced when the surface of the liners is rough, when the rock is angular, adhesive, damp, or low in density.

3. Speed.—The theoretical capacity is diminished when a machine is run at so high a speed that the rock is unable to fall the entire distance to be traversed at each cycle. If the force of gravity alone does not provide the necessary falling velocity, either the machine must be run more slowly, or force must be developed in some other manner, as, for example, by the centrifugal action of the disk crushers. Depending upon gravity alone, increasing the speed of the machine obviously cannot cause a proportionate increase of capacity beyond a rather low limit. The theoretical output corresponding to a high speed can rarely be attained. The effect approaching the speed limit therefore requires to be examined.

4. Slipping.—The theoretical capacity of a crusher will be diminished by the slipping and upward displacement of the rock as the jaws advance upon it. This result is to be expected when the angle between the jaws is unduly large; when the distance between the jaws is considerably greater than the size of the rock fragments; or, when the surface of the jaw liners is too smooth. This upward thrust may sometimes be detected at a point higher than the displacement zone under consideration.

5. Interference.—The theoretical capacity is diminished by the fact that the rock, even under compression between the closed jaws of the machine, will not occupy the full displacement volume. Voids are always present. This irregularity is probably the principal cause of the discrepancy between theoretical and practical crushing capacity. Since ore or rock is practically incompressible, some free space is necessary to permit readjustment of pieces as fracture occurs. Any condition which diminishes this inetrstitial space increases the output, but at the expense of friction and excessive comminution of the material. The presence of fine material, whether entering with the feed, or retained by failure of the crusher to release it, diminishes the volume of voids.

In the following calculations, the factor k is made to embrace all of the

influences above described, which cause divergence between theoretical and actual capacity. The theoretical weight of material which should be discharged at each cycle of the crusher then can be derived by extending equation (8) as follows:

$$M = \frac{a^2 - b^2}{2\,(e - b)} \times w \times d \times s \times 0.036 \tag{9}$$

where M is the weight in pounds delivered at each cycle; a, b, e, d, and w, in inches, are the dimensions previously described; s is the average density of the solid rock; and 0.036 is the approximate weight of 1 cu. in. of water.

The actual capacity in tons per hour is:

$$T = \frac{a^2 - b^2}{2\,(e - b)} \times w \times d \times s \times 0.036 \times \frac{60n}{2000} \times \frac{k}{100} \tag{10}$$

where n is the thrusts or revolutions per minute, and k is the percentage of the theoretical capacity actually attained.

Simplifying equation (10) gives

$$T = 0.0000054\,\frac{a^2 - b^2}{(e - b)} \times n \times w \times d \times s \times k \tag{11}$$

or

$$k = \frac{T\,(e - b)}{0.0000054\,(a^2 - b^2)\,nwds} \tag{12}$$

The value of the factor k, corresponding to a given set of operating conditions, could now be ascertained by test. Most of the tests were conducted with small crushing machines so adjusted as to allow an operating period of three minutes or more, and dimensions were measured with all possible accuracy. The most difficult measurement was the width of the opening between bottoms of jaws, when these were closed, or HI of Fig. 1. This distance was averaged to the nearest 0.01 in., and the surfaces and edges of the liners were kept in as good condition as possible. In some of the tests, a much larger quantity of material and a longer period of operation had to be chosen, to suit special requirements as to weighing or measurement. Time was measured to ⅕ sec., and the output was deflected for a definite period, and then weighed. Of the following-named crushers tested, the first and the third were most used.

1. A Dodge jaw breaker having a top opening of 4 by 6 in., with 6 by 8-in. liners at an angle of 30°. This was adjustable as to speed, and was modified to change the position and form of jaws, angle between jaws, and the amplitude of movement at top and bottom.

2. A Sturtevant laboratory jaw crusher having a top opening of 3 by 6 in., with 6 by 6-in. liners at 30°, and amplitude of $\frac{5}{64}$ in. at the bottom of the liners.

3. A McCully model crusher, having an opening 9 in. in diameter at the top, an angle of $22\frac{1}{2}°$ between breaking head and liners at the bottom, and an eccentricity of 0.0769 in.

4. A Comet crusher, laboratory size, with opening 9 in. in diameter, angle of 20° between breaking head and liners at the bottom, and eccentricity of $\frac{1}{8}$ in.

5. A McCully No. 5 crusher, with opening 36 in. in diameter at the top, and an angle of 25° between breaking head and liners as worn at the bottom.

6. A Hercules-Blake crusher, with 6 by $7\frac{1}{2}$-in. opening, and angle of 26° between jaws.

The results of the following tests are grouped in such a way as to show the effect of each variable condition, by itself, on the value of the factor k. These variable conditions were: (a) toughness of the rock; (b) density of the rock; (c) coarseness of feed; (d) uniformity in size of feed; (e) fineness of product; (f) speed of the crusher; (g) angle between jaws; (h) condition of jaw faces; (i) amplitude of movement; (j) position of jaw fulcrum; (k) principle of motion, whether gyratory or reciprocating.

(a) *Toughness of Rock.*—The three materials selected for this test were as nearly as possible identical in density, size, and other qualities, but varied in toughness. They were: hard quartz, having high elasticity but little plasticity; a less elastic granite; a tough and only slightly elastic trap rock. The averaged results of closely duplicating are shown in Table 1.

TABLE 1.—*Variation in Toughness of Rock*

CONSTANT CONDITIONS

Crusher used: Dodge type.

a—Distance between jaws (open) at bottom.................... 0.520 in.
b—Distance between jaws (closed) at bottom.................... 0.313 in.
e—Distance between jaws (closed) at top...................... 4.67 in.
w—Width of jaws.. 6.25 in.
d—Vertical depth of crushing space........................ 8.12 in.
   Screen size of feed.................................... 1–1.5 in.
n—Rev. per min... 304

MATERIAL	DENSITY (s)	TONS PER HR. (T)	k
Quartz..................................	2.68	0.743	84.1
Granite.................................	2.66	0.665	76.0
Trap...................................	2.61	0.617	71.8

The results indicate a slightly higher value of k for the more elastic rock. Apparently the tougher rock breaks in such a manner as to produce

less fine material, so that displacement space is less closely packed with the crushed product. Screen tests of the products confirmed this general observation.

(b) *Density of Rock.*—A number of materials, varying widely in density, were tested, the results being shown in Table 2.

TABLE 2.—*Variation in Density*

CONSTANT CONDITIONS

Crusher used: Dodge type.

    a—Distance between jaws (open) at bottom.................... 0.520 in.
    b—Distance between jaws (closed) at bottom.................. 0.313 in.
    e—Distance between jaws (closed) at top..................... 4.67 in.
    w—Width of jaws... 6.25 in.
    d—Vertical depth of crushing space........................ 8.12 in.
    Screen size of feed...................................... 1–2 in.
    304

MATERIAL	DENSITY (*s*)	TONS PER HR. (*T*)	*k*
Coke.................................	1.11	0.169	46.2
Coal.................................	1.91	0.288	45.8
Granite...............................	2.66	0.644	73.5
Stibnite in quartz......................	3.03	0.736	73.8
Chalcocite in quartz....................	4.40	1.083	74.7
Galena in quartz.......................	6.15	1.607	79.3

Although certain properties of the above materials, other than density, such as toughness, crystalline structure, and frictional coefficient, might influence the results, it is thought that the range in density is sufficiently wide to indicate a trend which is not entirely obliterated by the effects of other physical qualities. From the wide variation in the value of *k* it is apparent that a material of low density occupies a relatively small proportion of the effective crushing space, probably because the force of gravity acting upon a light material is not sufficient in the time available to effect the compactness and stability that is possible with heavy material.

(c) *Coarseness of Feed.*—The effect of variation in the coarseness of feed was tested with a uniform grade of granite; results are shown in Table 3.

The actual output largely exceeds the theoretical in the case of material composed of particles which are capable, without crushing, of passing through the machine. Further, the output diminishes with an increase in coarseness of the material. From this, it might appear that the limiting zone, when crushing coarse rock, must lie well above the discharge opening, at the place where the larger pieces are being shattered. This, however, is not the case. In all crushers now in use, whether of the Blake or the Dodge type, the crushing capacity is greater in the upper part, where the space between the jaws is wide, and where the larger

TABLE 3.—Variation in Coarseness of Feed

CONSTANT CONDITIONS

Crusher used: Dodge type.

a—Distance between jaws (open) at bottom.................... 0.450 in.
b—Distance between jaws (closed) at bottom.................. 0.240 in.
e—Distance between jaws (closed) at top..................... 4.60 in.
w—Width of jaws... 6.250 in.
d—Vertical depth of crushing space......................... 8.12 in.
n—Rev. per min.. 304
s—Density... 2.66

SIZE OF FEED, IN.	TONS PER HR. (T)	k
3–4	0.299	40.8
2–3	0.507	69.1
1–2	0.544	74.2
½–1	0.693	94.5
⅛–½	1.252	170.4

fragments are being crushed. It is the failure of the coarsely shattered rock, when falling into the zone of final crushing, to lodge in a compacted condition that causes the retardation in output. In this connection, it is interesting to note that when a small proportion of coarse fragments is added to a given grade of finer material there is no appreciable diminution of output, thus indicating that it is not so much the crushing of large fragments that limits the capacity as the condition in which the material lodges in the displacement space below. When the feed is already fine enough to pass the discharge opening, its flow is governed by conditions which are largely independent of the motion of the crusher.

(d) *Uniformity in Size of Feed.*—Variation in the proportion of coarse and fine particles in the feed introduces a larger number of variable elements than any of the characteristics previously described. The matter may be considered from two directions: first, the influence of a small but increasing amount of fine material added to a fixed amount of coarse material; second, the effect of adding a small but increasing amount of coarse material to a fixed quantity of relatively fine rock. That is, a mixture of coarse and fine is to be compared with either the coarse alone or the fine alone. Tests were conducted in several crushers and with various rocks.

These tests indicate that when two sizes occur in the feed, the finer material, if of a size to require crushing, exerts the controlling influence on the value of k; and that the presence of the coarse has little influence until its amount exceeds a certain limit. So long as the quantity of fine material is sufficient only to fill the interstitial spaces between the coarse pieces, giving the maximum compactness in the feed, the ratio of actual to theoretical output, or the value of k, is practically that recorded by the fine alone. The addition of a small amount of fine to a large proportion of coarse material increases the output to an extent nearly equalling the

entire quantity of fine material added, up to the point when the capacity is practically identical with that observed for the fine alone; at this point the interstices between the coarse pieces will be most fully occupied by the fine. Beyond this point capacity is uninfluenced by the presence of the coarse, but is defined by the quality of fine material. Minor differences were observed in the behavior of various materials.

(e) *Fineness of Product.*—The adjustment as to fineness of product is made differently in different machines. In some, a change in size of aperture alters the character of the thrust. This group of tests was made with a crusher of the Dodge type, in which the distribution of velocities and character of the thrust were not appreciably changed by the adjustment of the discharge opening. The results are shown in Table 4.

TABLE 4.—*Variation in Maximum Size of Product*

CONSTANT CONDITIONS

Crusher used: Dodge type.

a–b—Amplitude of jaw movement at the bottom.................. 0.210 in.

w—Width of jaws.. 6.25 in.

d—Vertical depth of crushing space......................... 8.12 in.

Size of feed... 2–3 in.

n—Rev. per min... 304

2β—Angle between jaws when closed......................... 30°

s—Specific gravity of rock (granite)......................... 2.66

DIST. BETWEEN JAWS (CLOSED) AT BOTTOM, IN. (b)	DIST. BETWEEN JAWS (CLOSED) AT TOP, IN. (e)	TONS PER HR. (T)	k
0.240	4.60	0.507	69.1
0.490	4.85	0.809	63.7
0.740	5.11	1.086	60.3
0.990	5.38	1.630	57.7

The results indicate a larger ratio of actual to calculated capacity when the crusher is set to discharge the finer product. Tests with other materials, not reported, indicated a similar trend, differing in extent, and the same relation was observed with other crushers and other initial sizes.

(f) *Speed of the Crusher.*—Variation in the speed of the 6 by 6-in. jaw crusher was obtained by combinations of pulleys to give the desired range. The results are shown in Table 5.

The results, within the range covered, indicate a progressive increase in ratio of actual to calculated capacity with decreased speed. Variation in speed affects the results in two ways, one being effective when the machine is driven at slow speeds and the other when at an exceedingly high speed. The velocity acquired by the falling rock in a crusher driven at high speed is limited by the distance through which it may

TABLE 5.—*Variation in Speed*

CONSTANT CONDITIONS

Crusher used: Dodge type.

a—Distance between jaws (open) at bottom..........   0.520 in.
b—Distance between jaws (closed) at bottom..........   0.313 in.
e—Distance between jaws (closed) at top............   4.67 in.
w—Width of jaws...................................   6.25 in.
d—Vertical depth of crushing space..................   8.12 in.
Size of feed.......................................   1–1.5 in.
s—Specific gravity of rock (trap) 2.61

REV. PER MIN. (n)	TONS PER HR. (T)	k
160	0.361	79.9
255	0.520	72.2
304	0.617	71.8
348	0.628	63.9
534	0.648	62.8
926	0.887	49.9
O		
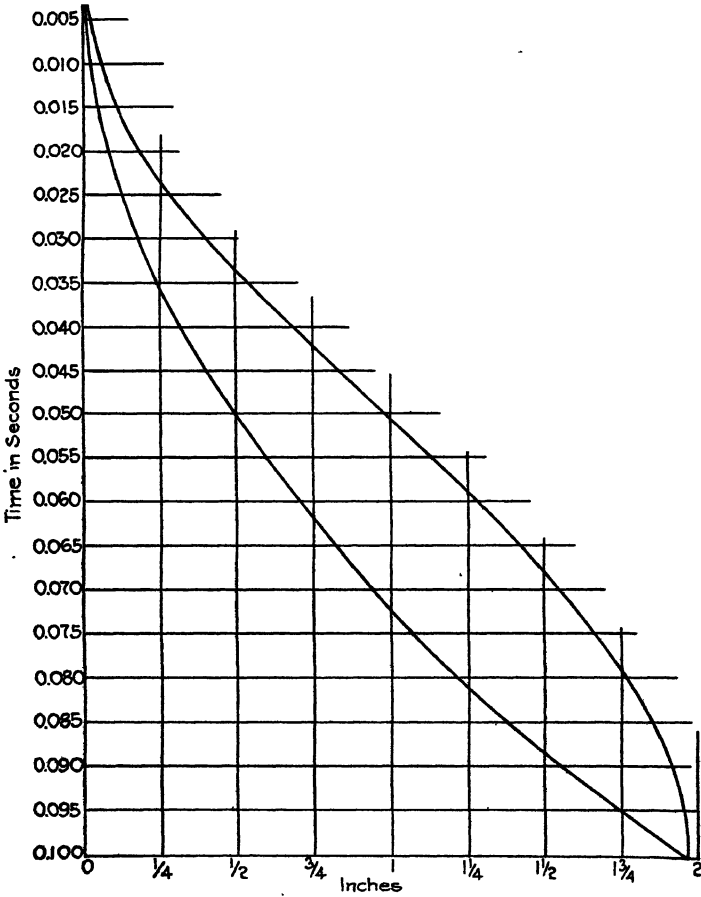
FIG. 2.

freely fall in the available time. Where gravity, as is usually the case, is depended upon to cause the fall, the jaws must obviously be allowed to remain open long enough, to permit the rock to enter the space made available. Since gravity is constant, the desired relation can exist only when speed is held within certain limits. In all cases, the rock begins to fall and acquires increased velocity as the jaws open. When the speed of the crusher is slow, the rock advances by sliding downward in constant contact with the surface of the jaws. On the other hand, when the speed is excessively rapid, the rock is allowed to drop freely, and in so doing it has opportunity to segregate; after thus shifting its position, it is less compact than it would be if in the form of a sliding wedge-shaped mass compacted by the previous thrust. Thus, while the force of gravity defines the limit to the capacity of a crusher driven at excessive speed, it is the speed itself which determines the capacity when the speed is low.

To interpret results of tests of this feature, it is necessary to observe the height through which the rock must drop to secure the full capacity within the time provided in the cycle, and to compare this height with the distance through which rock falls under the force of gravity. Table 6 shows the computed vertical distance, disregarding friction, through which material will fall by gravity at intervals of 0.005 sec. up to 0.25 second.

TABLE 6.—*Falling Heights under the Action of Gravity*

TIME, SEC.	DISTANCE, IN.	TIME, SEC.	DISTANCE, IN.
0.005	0.0048	0.130	3.264
0.010	0.0193	0.135	3.521
0.015	0.0435	0.140	3.786
0.020	0.0773	0.145	4.064
0.025	0.1207	0.150	4.347
0.030	0.1739	0.155	4.641
0.035	0.2367	0.160	4.946
0.040	0.3092	0.165	5.260
0.045	0.3912	0.170	5.582
0.050	0.4831	0.175	5.916
0.055	0.5845	0.180	6.259
0.060	0.6955	0.185	6.612
0.065	0.8162	0.190	6.976
0.070	0.9467	0.195	7.345
0.075	1.087	0.200	7.728
0.080	1.237	0.205	8.120
0.085	1.396	0.210	8.520
0.090	1.565	0.215	8.930
0.095	1.744	0.220	9.350
0.100	1.932	0.225	9.781
0.105	2.130	0.230	10.22
0.110	2.338	0.235	10.67
0.115	2.555	0.240	11.13
0.120	2.782	0.245	11.60
0.125	3.019	0.250	12.07

In Fig. 2, the curve at the left shows the distance through which rock, acted upon by gravity, will drop in periods of time increasing by 0.005 sec. up to 0.10 sec. The curve at the right, based on the rate of recession of the jaws, shows the corresponding free vertical distance between jaws available for the fall of the rock, in a crusher of the plain eccentric type, when driven at 300 r.p.m. or 0.20 sec. per cycle. This provides 0.10 sec. for the opening of the jaws sufficiently to meet the requirements of the gravity curve at the end of the same period. Under these conditions, the rock is free to fall throughout the entire interval of time until the approaching jaw abruptly arrests its descent. The vertical distance through which the rock is required to drop in order to keep in contact with the receding jaws is *PI* of Fig. 1. In the tests recorded in Table 5 the speed was increased nearly to the theoretical limit, which in this case was 674 r.p.m.

(g) *Angle between Jaws.*—In the design of a crusher, the angle between the two jaws must be well below the coefficient of friction, in order that the fragments of the rock may not be projected upward. This required angle depends upon the nature of the rock and the condition of the metal surface, but in any particular crusher the angle is fixed and small enough to suit any ordinary use. In the following series of tests, the angle was varied by special castings for jaw liners, the results being shown in Table 7.

TABLE 7.—*Variation of Jaw Angle*

CONSTANT CONDITIONS

Crusher used:

a—Distance between jaws (open) at bottom.................... 0.592 in.
b—Distance between jaws (closed) at bottom.................... 0.375 in.
w—Width of jaws.. 6.25 in.
d—Vertical depth of crushing space........................ 8.12
n—Rev. per min... 304
Size of feed... 1–1.5 in.
s—Specific gravity of rock (trap)............................ 2.61

ANGLE BETWEEN JAWS, DEG. (2β)	TONS PER HR. (T)	JAW SPACING AT TOP (e)	k
30	0.754	4.74	72.0
27	0.770	4.27	65.7
20	0.873	3.24	54.8
14	0.862	2.37	37.7

The apparent effect of decreasing the angle is to diminish the compactness of the rock. The speed was sufficiently low to allow time for the material to drop as rapidly as the recession of the jaws permitted.

(h) *Condition of Jaw Faces.*—For these tests, three conditions of surface were compared. The surface designated as rough was a cast-iron liner roughened by use, and studded with steel bolts ⅜ in. in diam-

eter at ¾-in. centers over the entire surface, and projecting 1⁄16 to ⅛ in. from the surface of the cast iron. The surface described as medium was a cast-iron jaw roughened by service but not misshapen. The condition described as smooth was a machined cast-iron surface. The results are given in Table 8.

TABLE 8.—*Variation in Roughness of Jaw Faces*

CONSTANT CONDITIONS

Crusher used: Dodge type.

    a—Distance between jaws (open) at bottom.................... 0.450 in.
    b—Distance between jaws (closed) at bottom.................... 0.240 in.
    e—Distance between jaws (closed) at top...................... 4.60 in.
    w—Width of jaws.. 6.25 in.
    d—Vertical depth of crushing space.......................... 8.12 in.
    n—Rev. per min... 304
    Size of feed... 2–2.5 in.
    s—Specific gravity of rock (granite)........................... 2.66

CONDITION OF JAW SURFACES	TONS PER HR. (T)	
Rough..	0.495	56.5
Medium..	0.611	69.8
Smooth..	0.649	74.2

Increasing the roughness of the jaw surface apparently reduces the percentage of occupancy of the displacement space.

(*i*) *Amplitude of Movement.*—The necessary adjustments for this test were made by raising the liners above the fulcrum of a Dodge crusher to the height desired, thus changing the movement of the jaw. Results are shown in Table 9.

TABLE 9.—*Variation in Amplitude of Jaw Motion*

CONSTANT CONDITIONS

Crusher used: Dodge type.

    b—Distance between jaws (closed) at bottom.................... 0.375 in.
    e—Distance between jaws (closed) at top...................... 4.474 in.
    w—Width of jaws.. 6.25 in.
    d—Vertical depth of crushing space.......................... 8.12 in.
    Size of feed... 1–1.5 in.
    n—Rev. per min... 304
    s—Specific gravity of rock (granite) 2.66

AMPLITUDE OF JAW MOVEMENT, IN.	JAW SPACING AT BOTTOM, WHEN OPEN (a)	TONS PER HR. (T)	k
0.125	0.500	0.632	106.9
0.156	0.531	0.758	99.3
0.187	0.562	0.853	90.7
0.217	0.592	0.874	77.0
0.250	0.625	0.990	73.3

Lower values of k were found with larger amplitudes, owing to the rapid increase in theoretical tonnage with the larger amplitudes ($a - b$ of equation (11). In the tests, the size of feed, the speed, and the maximum coarseness of the crushed product were held constant, the speed being slow enough to allow ample time for the rock to drop into the region of final crushing. Additional tests with other machines, though not so definitely conclusive, indicated the same trend of results.

(*j*) *Position of Jaw Fulcrum.*—Comparison was made between crushers of the Blake and the Dodge types, in which the greater amplitude of movement is respectively at the bottom and at the top of the jaw. The results obtained were subject to the influence of surface differences which could not be absolutely correlated under the conditions of working. The comparative tests gave values of k higher in some cases, and lower in others, for the machines of either type. The average of results was somewhat higher for the Dodge and lower for the Blake, under ordinary working conditions.

(*k*) *Gyratory vs. Jaw Crushers.*—The results of tests with the Comet and the small McCully crusher, operating on granite 2-in. and 1-in. in size, and crushing to ⅜-in. did not reveal differences beyond the limits of accuracy attainable in the preparation of the surface of the liners and the control of dimensions. The effect of the main difference in principle, namely, curved surface as contrasted with flat surface, is small in comparison with other influences which have been considered. It is to be noted that in milling practice the gyratory crusher often is not supplied with enough feed to develop its full capacity, a condition which was not allowed to occur in the tests.

CONCLUSION

When estimating the capacity of crushers under given conditions, it is necessary to employ an empirical factor which has here been designated by the letter k, representing the percentage ratio between actual crushing rate, and a theoretical capacity based on the dimensions and other data correlated in equation (12). The accuracy of this factor limits the precision of the estimate. The actual value of k is a resultant of numerous contributing influences. The tests were designed to study the effect of each dominant influence by itself, and to approximate a correct value for the factor under known conditions. Under ordinary operating conditions, k will rarely exceed 75 per cent.; for particular conditions, a somewhat more accurate value can be deduced from the data referring to the individual tests.

HENRY HANSON, San Francisco, Calif. (written discussion).—The experiments, as tabulated in the paper, show many interesting angles to rock crushing not previously recorded, or at least not so definitely established. The experiments show certain definite factors governing the capacity of a crusher and also indicate to what extent these variable factors influence the capacity of a crusher. The most important of these are the character, density, and size of ore fed to the machine; angle of jaw to back liner plate; size of opening; amplitude of jaw movement; and speed of crusher, in relation to the capacity. These factors are clearly shown and should stimulate work in the field to confirm such phases of the experiments as can be made of practical application to crushers now in use.

The experiment under Table 3, showing a progressive increase in output by a decrease in the size of rock in the feed, is interesting. In these experiments the value of k is increased to a point far above the theoretical. In this connection it would be of interest to set the crusher opening at what was the mean of the 0.45 and 0.24-in., namely, 0.345-in. opening, supplying the crusher with a feed that would pass that opening freely when the crusher is at rest and note the amount that could be put through under those conditions.

In the sixth test in Table 5, there seems to be an undue increase in the tonnage passed through the crusher as a result of the increase in speed. It would have been interesting to have increased beyond the theoretical point in order to have established in actual practice where the turning point is when the output from the crusher would decrease. The greatest difficulty encountered, in practice, in coarse crushing is clayey or spongy rock.

DISCUSSION

ERNEST A. HERSAM (author's reply to discussion).—The effect of feeding material of a size below that of the discharge opening, as in the case of the last item of Table 3, is a condition of a flow of sands or crushed material through orifices, rather than of crushing. The results observed as this condition was increased to a varied extent ranged from a slight increase above the calculated value to a high limit determined solely by the size of the aperture or discharge space below. The subject of the flow of sands through orifices has been discussed previously.[2] Higher speed, which in the case of Table 5 would have been desirable, was inadvisable because of the severe vibration and mechanical difficulty with a crusher of the type available running at this excessive speed. At the

. [2] The Flow of Sands Through Orifices, *Jnl.* of the Franklin Institute (April, 1914) **177**, No. 4, 419–444.

limit of increase in capacity, toward which the sequence of observations leads, it is to be expected that the residual influences, depending on the kinds of rock such as those to which Mr. Hanson refers, would give irregular results concealing the actual point of limitation except by taking the average of many results.

Surface Tension and Adsorption Phenomena in Flotation

By ARTHUR F. TAGGART* AND A. M. GAUDIN, NEW HAVEN, CONN.

(San Francisco Meeting, September, 1922)

FLOTATION of ores is a practical utilization of the energy that resides in the surfaces of solids and liquids. The best known manifestation of this energy is called surface tension; an equally important, though less apparent, phenomenon is adsorption. This paper reviews some of the recent investigations of these phenomena, and gives the results of certain experiments performed by the authors.

Surface tension is a manifestation of energy having its seat at the bounding surface between the adjacent phases of a multiphase system. Thus, at the interface between a liquid and a gas, or between two immiscible liquids, or between a solid and a gas or a liquid, there are forces capable of doing work, their magnitude depending on the nature of the adjacent phases and the extent of the bounding area. These surface forces are inappreciable when the ratio of surface to volume of the adjacent phases is small, but they become powerful when this ratio becomes large, as it does when one of the phases is highly dispersed. Froth flotation, which consists in buoying solids heavier than water to the surface of water by means of gas bubbles, utilizes the force of surface tension to attach the solid particles to the gas bubbles and again to maintain the floating mass of bubbles as a more or less persistent froth at the surface of a pulp for a sufficiently long time to allow its removal.

It follows from the second law of thermodynamics that the potential energy of a system in equilibrium is a minimum. Therefore a multiphase system, tending toward equilibrium, will arrange itself in such a way that the total energy of the interfaces will be the least possible. This end can be accomplished in two ways: by a change in shape that will reduce the interfacial area; by a change in the nature of the surface layer in

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such a way that the energy per unit of area is diminished. The tendency of freely flowing liquid drops to take a spherical shape is an illustration of the first method of reducing surface energy to a minimum; the coalescence of drops in an emulsion is another illustration of the same tendency. The second method of decreasing the potential energy of a system manifests itself as adsorption.

Adsorption is a process of rearrangement of a system, which results in establishing a difference in concentration, as between the interfacial layers and the bulk of the phases, of a substance or substances initially uniformly dispersed in one or several of the phases. Rearrangement occurs in such a way that the energy of the interface is reduced by concentration of the dispersed substance. The diminished energy is shown by a decrease in surface tension.

For adsorption to occur in a two-phase system, it is necessary that one of the phases, at least, be of such nature that segregation can take place in it. In a two-phase system composed of pure substances, as, for instance, water and water vapor, no adsorption can occur; in a system composed of air and a solution of salt in water, adsorption can and will take place.

In a system composed of air and a solution of a given solute in water, the surface layer of the solution will have a different composition from that of the bulk of the liquid. If the solute increases the surface tension of water, as common salt does, the surface layer will have a solute concentration less than the bulk of the liquid; if the solute tends to decrease the surface tension of water, as acetic acid does, the surface layer will have a solute concentration greater than the bulk of the liquid. These two kinds of adsorption are known, respectively, as negative and positive adsorption.

The tendency of the solute to go to or away from the surface is opposed by the tendency it has to go into solution, that is, to equalize the concentration at all points. When equilibrium is reached, the excess of concentration in the surface layer over that in the bulk is given approximately by Gibbs's equation:

$$U = - \frac{c}{R\theta} \times \frac{dT}{dc} \qquad (1)$$

where U = excess concentration in surface, in gram-molecules per square centimeter;

R = gas constant, in ergs per degree centigrade = 8.32×10^7;

θ = absolute temperature, in degrees centigrade;

c = concentration, in any convenient unit, as percentage or gram-molecules per liter;

T = surface tension, in dynes per centimeter.

In case the solute is ionized, the dissociation constant i appears in the formula

$$U = -\frac{c}{iR\theta} \times \frac{dT}{dc} \qquad (2)$$

By rewriting these equations thus

$$\frac{U}{c} = K \times \frac{dT}{dc} \qquad (3)$$

K being a constant for a given system, it may be seen that degree of adsorption and change in surface tension are directly related.

An important qualitative conclusion that can be drawn from an examination of the above formulas is that a small quantity of dissolved substance may reduce the surface tension considerably but may increase it only slightly. This agrees with all available experimental data.

Colloidal Solutions and Emulsions.—Gibbs's equation applies only to dilute solutions. It assumes molecules to be in a state of constant kinetic agitation. If, instead of molecules, there are aggregates of molecules in solution, that is, association or colloidal solution, the movement of these aggregates will be much less violent. If the aggregates are large enough, they can be seen, with an ultra-microscope, to have an irregular motion known as Brownian movement. Still larger aggregates, such as the globules in an emulsion, do not exhibit the Brownian movement. To none of these aggregated dispersions can Gibbs's equation be expected to apply. But, although the equation is not available to measure the degree of adsorption, its assumption points to the probability that the more sluggish groups of molecules will adsorb, although to a less degree than individual molecules and ions. Experiment shows that this is so. From the same reasoning, no adsorption is to be expected when no solution is present, that is, when the disperse phase is a suspensoid or emulsoid, mechanically dispersed in a pure liquid second phase. Experiment confirms this conclusion to the extent of showing that such dispersions have no effect on the surface tension of the dispersing phase.

Emulsions are suspensions of droplets of one liquid in a second liquid with which the first is immiscible but in which it is not necessarily insoluble. The droplets are larger than colloidal size, by definition, and are therefore proportionately more subject to gravitational action and less to molecular action. A two-phase emulsion is in a state of unstable equilibrium because of the tendency toward coalescence of the droplets of the dispersed phase when these are crowded together by the action of gravity. The tendency to coalesce exists by reason of the general tendency to lower the total potential energy of the system by a change of shape that will decrease the total interfacial area. To be stable, emulsions require the presence of a third substance, which may be miscible or non-miscible with either or both phases but which, by passing into the in-

terface between the phases, will lower the total potential energy of the system. In case the contaminating substance forms a viscous film at the interface, coalescence is prevented mechanically. If the phenomenon involved in the passage of the contaminant into the interface is one of adsorption from solution and is not accompanied by the formation of a viscous film or skin, it is probable that coalescence is prevented by reason of the fact that the variable surface tension induced in the interface between the phases allows considerable deformation of the interfacial film without coalescence. The breaking down of emulsions just described should not be mistaken for adsorption of the dispersed phase at the surface, and while it may affect a surface-tension measurement taken after its occurrence, because a layer of the dispersed phase has replaced the dispersing phase, it should not be considered as a proof of adsorption in emulsions.

Degrees of Dispersion of Flotation Oils.—In flotation, the performances of both solutions and emulsions are important. Most of the frothing agents used in flotation are oils and a majority of these are complex mixtures, the individual constituents of which vary widely in solubility. Consequently, in any mixture of a flotation oil with water under the conditions of agitation usually employed to distribute the oil through the pulp, the state of dispersion is likely to vary from that of emulsion, as above defined, to one that is molecular at least. For practical purposes of flotation terminology, it is probably sufficient to consider as a solution any mixture that is water-clear to the naked eye, and not to investigate this mixture further by means of the ultra-microscope in order to determine whether the state of dispersion of the oil therein is colloidal; or by such tests as osmotic pressure, conductivity, variation in freezing point, etc., to determine whether the state is molecular or ionic.

Surface Tension of Solids.—Solids possess surface energy, but because they lack mobility this energy cannot be quantified by a determination of surface tension. In the absence of any measurement, the argument for supposing the existence of surface tension in solids is based on experience in physical continuity, thus: The surface tension of liquids increases from zero at the critical point to a maximum at the freezing point; it is reasonable, therefore, to infer that surface tension continues to exist in the solid state and is likely to increase in the change of state from liquid to solid.

Adsorption at Solid Surfaces.—Solids, possessing surface energy, should adsorb solutes that decrease the interfacial tension between themselves and the neighboring phase; particularly they must adsorb molecules and ions from solution. This phenomenon, however, should not be confused with the tendency of certain solids to pass from one liquid phase to another, as from the dispersing to the dispersed phase of an emulsion.

In a system composed of finely dispersed solid matter in a water solution of a flotation oil, adsorption will occur and equilibrium will be reached by a chain somewhat as follows:
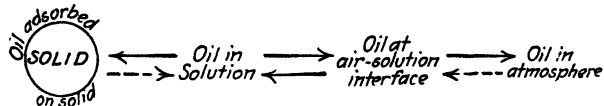
Two of the equilibrium arrows are shown dotted because of doubt as to the reversibility of the movement expressed by the adjacent full-weight arrows.

APPARATUS AND PROCEDURE

Adsorption Measurements.—The principle underlying adsorption measurements on flotation oils is simple. The surface tension of a given mixture of the oil with water is measured, adsorbent is added, the mixture is agitated for a given period, and the surface tension is again measured. If the second observation denotes a surface tension nearer to that of water, a part of the oil has been taken from the water, either by evaporation or by adsorption. The amount of evaporation is determined by running a blank. The amount adsorbed can be determined by the use
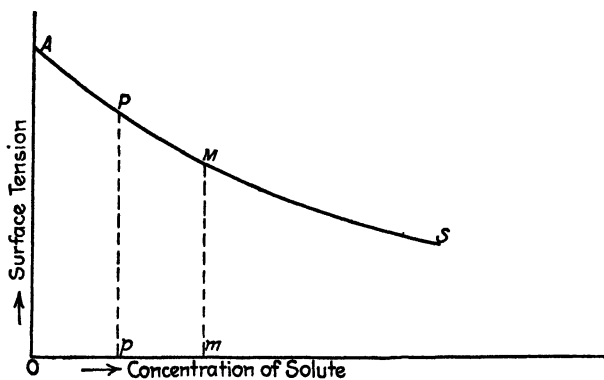
FIG. 1.—TYPICAL SURFACE TENSION-CONCENTRATION CURVE FOR ORGANIC SOLUTES.

of a concentration-surface tension diagram. Let *AS*, Fig. 1, be such a curve for a given oil-water mixture. If *Mm* represents the surface tension of a solution of concentration *Om*, and *pP* is the surface tension after adsorption has taken place, *Op* must be the concentration of the solution after adsorption and *pm* is a measure of the amount of solute adsorbed. This method requires a determination of the concentration-surface tension diagram for any oil-water mixture before adsorption tests

are conducted. However, if the solutions tested are dilute enough for a straight-line relation between surface tension and concentration, the problem is simplified. Another reason favoring the selection of a dilute solution is that in such a solution the ratio of the change in surface tension to change in concentration is maximum. Practically the best conditions are attained when the surface tension of the solution is from 5 to 10 dynes per cm. below that of water, say, about 65 dynes per cm.

Apparatus for Surface-tension Tests.—The apparatus used for surface-tension tests in the following work is a modification of an apparatus described by A. W. Fahrenwald[1] for Wilhelmy's method. Essentially,
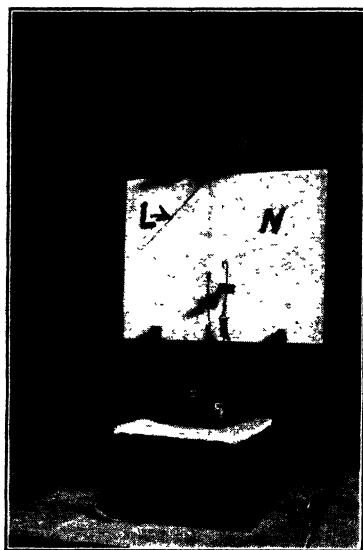
FIG. 2.—SURFACE-TENSION APPARATUS.

the apparatus (Figs. 2 and 3) is a balance for weighing the pull exerted by the surface under investigation. A grooved aluminum pulley A mounted on a steel shaft B is supported by centered knife edges on curved, round, glass bearings C. The bearings are carried on members D at the top of a rigid wooden frame E. A silk thread F passing over the pulley supports at one end a knife edge G and at the other a counterweight H. Metal hooks J of No. 20 (B. & S. gage) brass wire carry corks K, which act as weight pans. A needle L and a counterweight M are mounted on the pulley A; the needle swings in front of a dial N. The liquid of which the surface tension is to be measured is placed in a flat crystallizing dish resting on four rubber cushions glued to an asbestos sheet supported on a ring; the ring can be raised or lowered by a low-pitch screw.

The principle of the apparatus is as follows: A double film of the

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[1] *Min. and Sci. Press* (Aug. 13, 1921) **123**, 227.

liquid of which the surface tension is desired is drawn along the metal
knife edge G, between the prongs O. The film exerts a pull on the knife
edge and causes the pulley A to turn. This raises the center of gravity
of the needle L and increases its lever arm until the pull on the knife edge
is balanced. When the film is first drawn, a high, variable pull is noted;
the needle then returns to a fixed point and remains there until the film
breaks. The variable maximum pull appears to be caused by liquid that
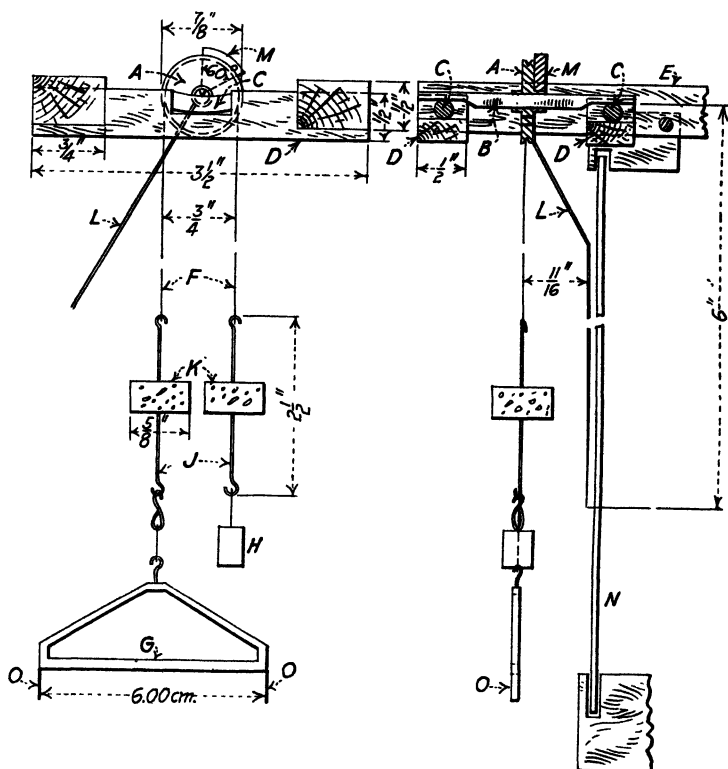
FIG. 3.—DIAGRAMMATIC VIEW OF SURFACE-TENSION APPARATUS.

quickly drains from between the surfaces of the film; it depends on the
thickness of the knife edge and on the extent to which it is wet.

The Fahrenwald apparatus, as described in the article referred to,
is subject to the following difficulties of operation: (1) To produce the
deflection of the needle corresponding to the pull of a film of high-surface
tension requires considerable vertical movement of the crystallizing dish
and this entails too much jarring of the apparatus when the film is
being drawn out. The smaller wheel in the apparatus here illustrated
gives the same deflection for a much smaller movement of the dish and
consequently gives steadier readings. (2) The large cork pulley without

counterweight lacked sensitivity. This is obtained by the small counter-weighted pulley of the present apparatus. (3) A groove in a cork pulley, no matter how carefully cut or finished, contains projecting pieces of cork that catch the thread and cause inaccurate weighings; the aluminum pulley of the present apparatus eliminates this difficulty. The present apparatus has curved glass bearings that give a low frictional resistance to turning of the knife edges. It carries corks for weight pans on both sides of the pulley; this is a convenience in weighing the film. A crystallizing dish is used instead of a watch glass for holding the liquid, thus insuring an even depth of liquid at all points. The best size of dish for a knife edge 6 cm. long is 10 to 12 cm. diameter. With a dish of this size, there is no tendency for the knife edge to hug the walls and the volume of liquid necessary for a test is small. More accurate results can be obtained by weighing the pull of the film each time than by reading directly from a graduated dial; this is done by marking the stopping point of the needle on the dial N and, after drying the knife edge, bringing it back to this point by proper weights.

Manipulation.—The liquid to be tested is placed in the crystallizing dish and the screw is turned to raise the dish until the knife edge is submerged. The dish is then lowered to draw out a film. The sensitivity of the apparatus is such that the arc through which the needle turns for a film tension of about 40 dynes per cm. is 90°. It is desirable that the arc described by the needle be substantially the same in magnitude and position for all measurements in order to eliminate the effect of any inequalities in the bearings. In order to insure this, weights are added to the knife-edge side of the balance when the surface tension is below 30 dynes, and to the other side when the surface tension exceeds 30 dynes. When the film is drawn out and the needle ceases to oscillate, its stopping point is marked on the dial, and the film is broken by further lowering the crystallizing dish. The knife edge is next cleaned and dried by treatment with acetone and ether. Weights are then placed on the film side of the balance, until the needle returns to the marked point. The pull weighed is that of the double film, including that on the prongs. The temperature of the liquid is recorded.

To obtain the surface tension of the liquid at 20° the following computation is made: (1) The observed pull is corrected for the prong pull. In the case of the apparatus illustrated this was determined to average 5.5 per cent. of the total pull of the knife edge including the prong pull. (2) This corrected pull is divided by 12 (2 × 6 cm. or length of the film drawn), and by (1000 ÷ 981) to convert the force from milligrams to dynes. The dividing factor thus obtained, 12.24, gives the actual surface tension in dynes per cm. at the observed temperature. (3) A temperature correction, amounting to about 0.2 per cent. per degree centigrade in the case of systems composed mainly of water, is applied to give the

surface tension at 20° C. This correction is additive when the actual, temperature is above 20° C. and subtractive when it is below.

Cleaning the Apparatus.—Early work showed that the surface tension of every preparation dwindled appreciably, often several dynes, with lapse of time; this was particularly true of emulsions. This dwindling was eliminated by cleaning the knife edge, after every dip in any solution
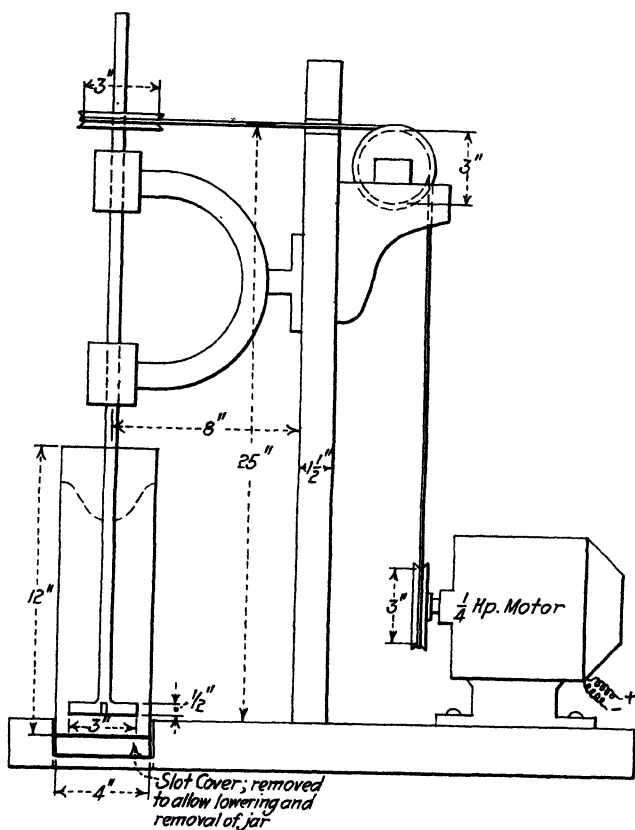
FIG. 4.—SQUARE GLASS JAR STIRRING APPARATUS.

or emulsion, by immersing it first in acetone and then in ether. The acetone removes any adhering liquid and the ether removes the acetone and effects perfect drying in a short time. Another method of cleaning would consist in burning the knife edge after each immersion. This would practically require a platinum edge and would appear to have no advantage over the method selected.

Preparation of Solutions and Emulsions.—The substances used in all the tests were flotation agents. Some, such as phenol and cresol, were

completely soluble in water; others were only partly soluble; one, Nujol, was insoluble.

Emulsions were prepared as follows: In a clean, square, glass jar, 4 in. on an edge and 12 in. high, was put 2000 cc. of clean tap water. This was slowly stirred by an impeller while the required amount of reagent was added. The impeller speed was next raised to 1700 to 2000 r.p.m. and after sufficient agitation at that speed the jar was
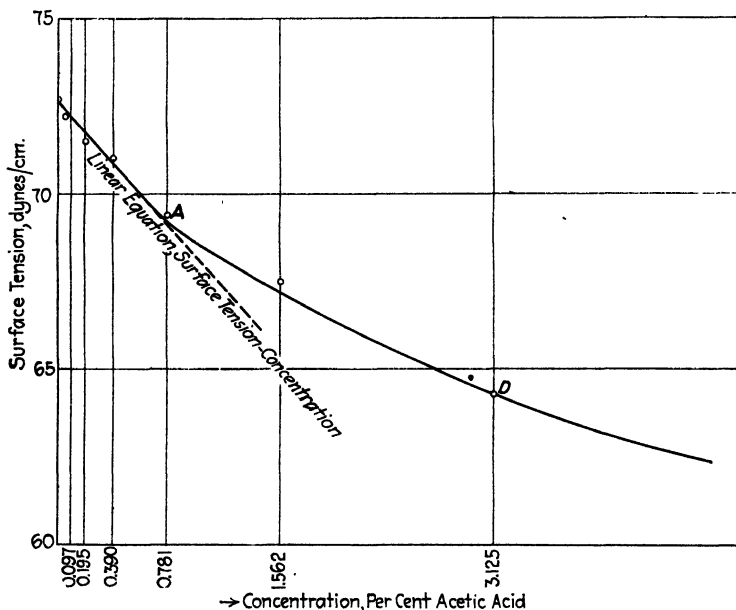
FIG. 5.—SURFACE TENSION-CONCENTRATION CURVE FOR DILUTE ACETIC-ACID SOLUTION.

removed and the liquid was sampled by a siphon into Erlenmeyer flasks. From two to four samples were taken each time. The stirring apparatus is shown in Fig. 4.

To obtain a solution, the procedure was the same, except that during the last minute of agitation from 5 to 10 gm. of infusorial earth were added to the emulsion, and the contents of the jar were subsequently filtered through a 5-gm. bed of infusorial earth under a 25-in. vacuum. The filtrate was refiltered, if necessary to clear. Sampling was done by siphoning, as in the case of emulsions. The samples were kept in corked Erlenmeyer flasks, and observations were made as soon as possible after preparation.

Surface tension-concentration curves were determined by varying the concentration by dilution with water and measuring surface tension, cor-

recting to a standard temperature, usually 20° C.; concentrations were then plotted as abscissas and surface tensions as ordinates, using a simple scale for short ranges of concentration, and a logarithmic scale for long ranges. Results for acetic acid are plotted on Figs. 5 and 6; cresol, Figs. 7 and 8; phenol, Figs. 9 and 10; the soluble parts of steam-distilled
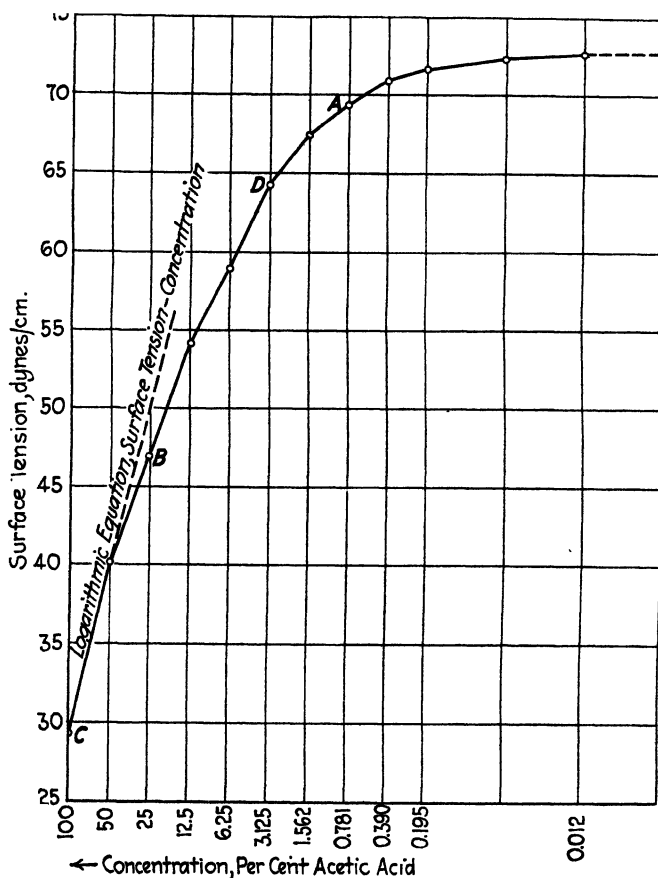
FIG. 6.—SURFACE TENSION-CONCENTRATION CURVE FOR CONCENTRATED ACETIC-ACID SOLUTIONS.

pine oil (General Naval Stores No. 5) Fig. 11; and of Barrett No. 4 coal-tar creosote, Fig. 12.

Figs. 7 to 10, inclusive, indicate that, in the case of homologs, the effect of solutes on the surface tension of a solution increases as solubility decreases. This conclusion is confirmed by data from Szyskowski's paper[2] summarized in Fig. 13. Figs. 7 to 10 and Szyskowski's data also

[2] Bohdan von Szyskowski, Zeit. Physik. Chem. (1908) 64, 385–414.

show that in the case of very dilute solutions the concentration of the solute and the surface tension are related by a straight-line law, in the case of concentrated solutions by a logarithmic law, while with intermediate concentrations the relation between the two is variable.
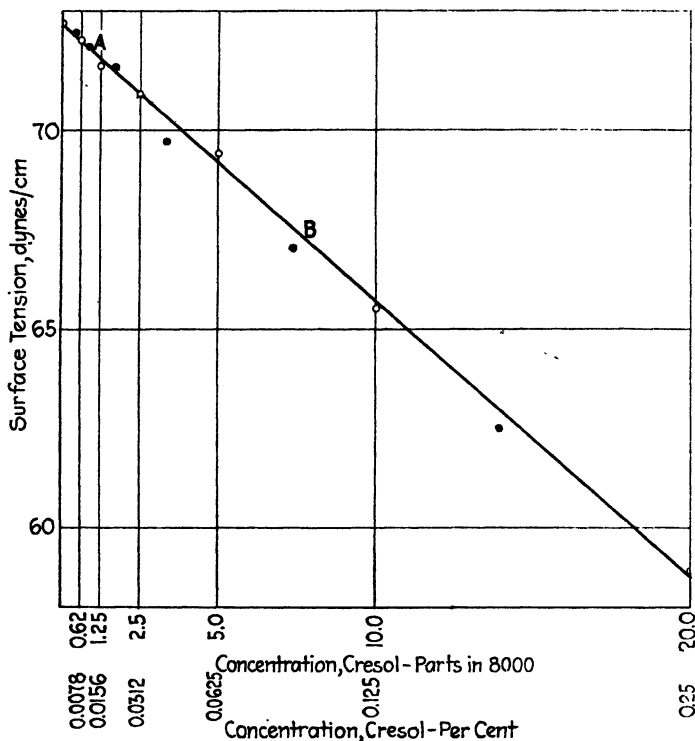
FIG. 7.—SURFACE TENSION-CONCENTRATION CURVE FOR DILUTE CRESOL SOLUTIONS BLACK CIRCLES, POINTS OF TEST 2F; LIGHT CIRCLES, POINTS OF TEST 11F.

FROTHING PHENOMENA

Pure water does not froth appreciably; neither do other pure liquids, unless they are highly viscous, in which case some large bubbles may linger for a while. On the other hand, practically all solutions froth, if the solute changes the surface tension of the solvent in any way. Water solutions of most flotation agents froth considerably; shaking in a corked flask for 2 or 3 sec. will produce a froth $\frac{1}{4}$ to $\frac{1}{2}$ in. deep, which will persist for several seconds. Insoluble oils agitated with water, but with no dissolved substance present, do not froth. If a small amount of powdered solid, such as ore, is added to a frothing solution, the froth may last for several hours, because of increase in viscosity of the bubble films; in fact, the latter may closely approach the character of a plastic solid.
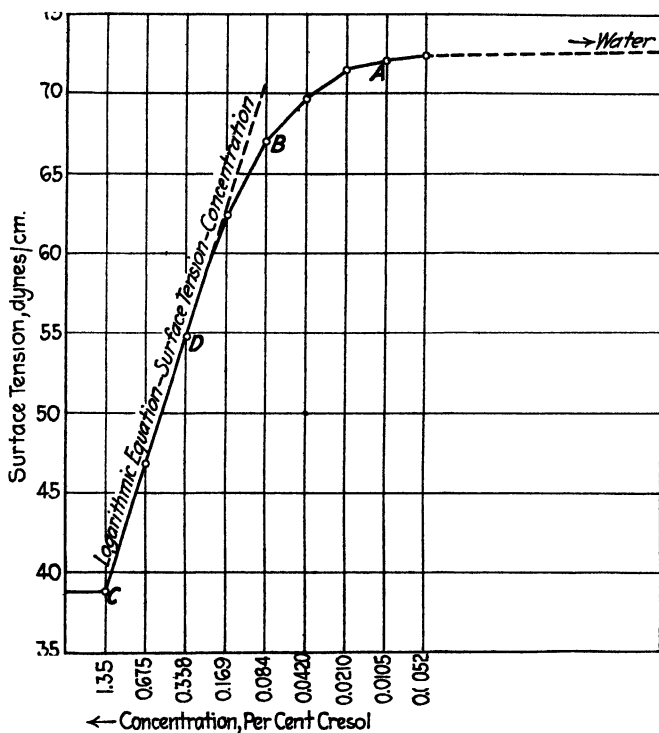
FIG. 8 —SURFACE TENSION-CONCENTRATION CURVE FOR CONCENTRATED CRESOL SOLUTIONS.
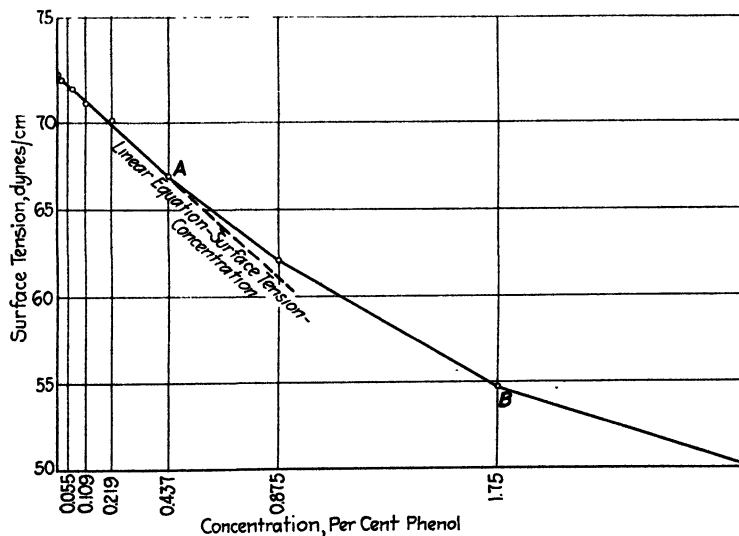
FIG. 9.—SURFACE TENSION-CONCENTRATION CURVE FOR DILUTE PHENOL SOLUTIONS.
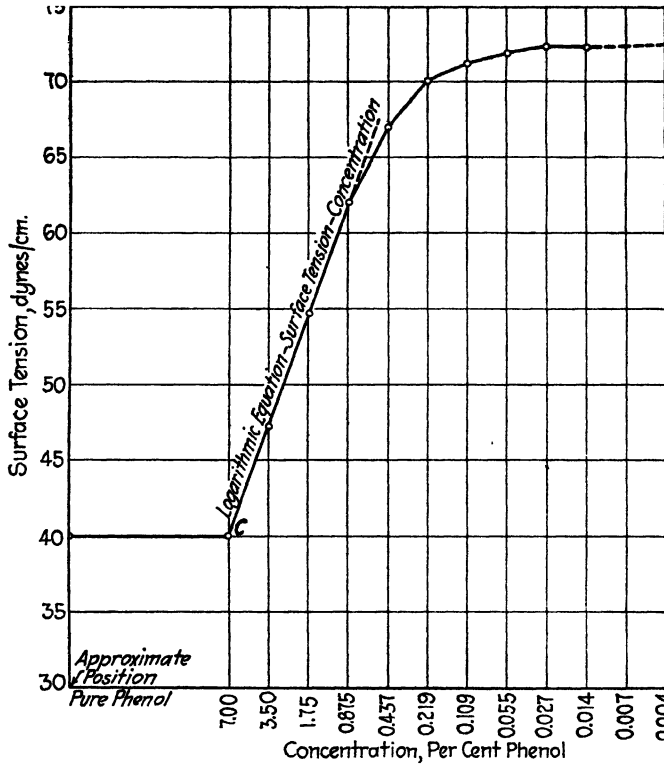
FIG. 10.—SURFACE TENSION-CONCENTRATION CURVES FOR CONCENTRATED PHENOL SOLUTIONS.
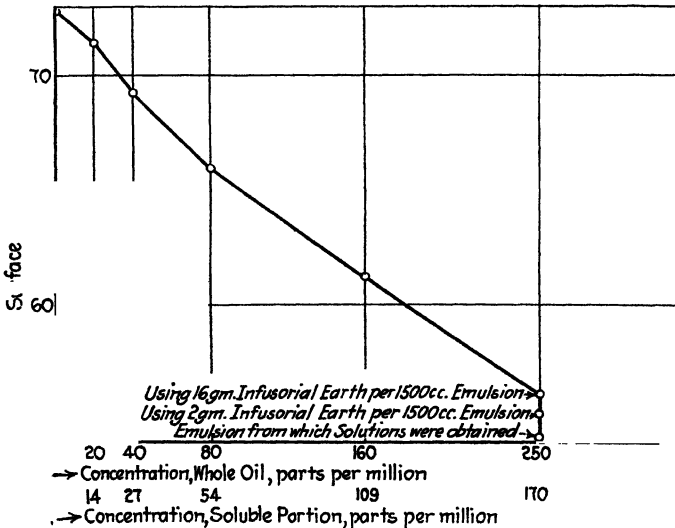
FIG. 11.—SURFACE TENSION-CONCENTRATION CURVE FOR SOLUBLE PORTION OF G. N. S. NO. 5 STEAM-DISTILLED PINE OIL.

Thus, two factors are essential to the production of a stable froth: the presence of a dissolved substance and an increase in viscosity of the surface film.
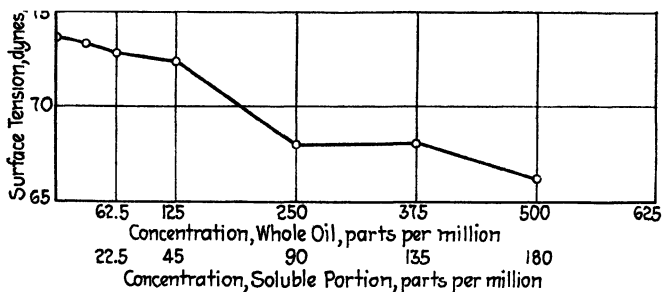
FIG. 12.—SURFACE TENSION-CONCENTRATION CURVE FOR SOLUBLE PORTION OF BARRETT NO. 4 COAL-TAR CREOSOTE.

The efficacy of dissolved substances in stabilizing froths does not lie in lowering the surface tension of the solution, but in imparting to the surface tension the properties of variability and adjustment to imposed
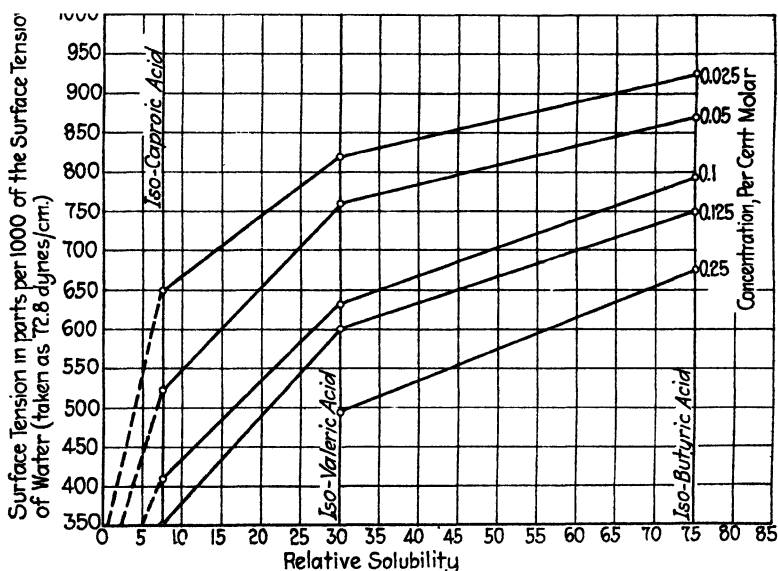
FIG. 13.—RELATION BETWEEN RELATIVE SOLUBILITY AND EFFECT ON SURFACE TENSION OF WATER SOLUTIONS OF THREE MEMBERS OF A FATTY-ACID SERIES.

strains. The soluble substance changes the surface tension considerably more than it would if no adsorption took place. Agitation results in bringing to the surface portions of the liquid that were in the bulk, thus forming new surfaces with higher tension than that of the old gas-liquid

surfaces and having, therefore, a tendency to diminish toward an equilibrium value which is that of the static surface tension. If a film, after reaching the static surface tension, is stretched, it thins down and liquid from the inner part reaches the surface. This liquid, because of its lesser concentration in solute, will have a higher surface tension than that of the old film and hence will counteract the force that tends to break the film. Thus the presence of a soluble substance in solution causes bubble films to have an adjustable surface tension, and, between the limits of the surface tension of the adsorbed film and that of the bulk of the liquid, to act as a film of rubber. Maximum range of adjustability will correspond to maximum frothing power. Fig. 14 represents an
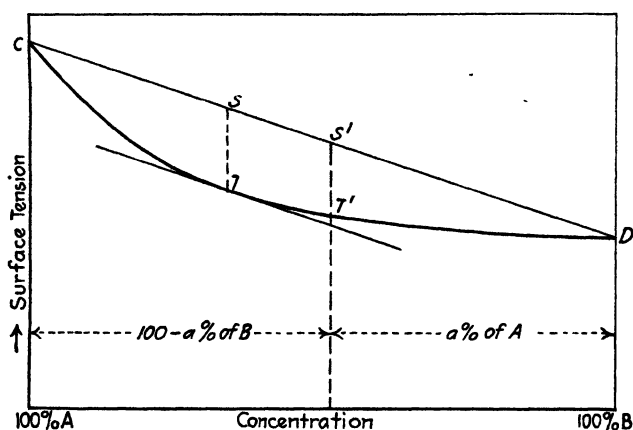
FIG. 14.—RELATION BETWEEN FROTHING POWER AND CONCENTRATION OF SOLUTE.

ideal case. CSS^1D is a hypothetical concentration-surface tension curve for the bulk of the liquid, which cannot, of course, be experimentally confirmed. It assumes a straight-line relation, as any other assumption would be an unjustifiable refinement. $CTT'D$ is the corresponding curve for the surface film; this is generalized from experiment. The distance $S'T'$ between the two curves is a measure of the frothing power, of the mixture containing a per cent. of the solvent A, and 100-a per cent. of the solute B. Point T, at which the tangent to the curve $CTT'D$ is parallel to $CSS'D$, corresponds to the greatest frothing power.

Frothing Experiment.—No accurate quantitative measure of frothing was devised, but an approximate measure, based on thickness and persistence of the froth layer, was used. Results for water solutions of cresol and phenol are shown in Figs. 15 and 16. These indicate that frothing is greatest for solutions having a concentration about one-half the saturation concentration, decreasing to zero for pure water and to a comparatively small figure for saturated solutions. The distance between the

hypothetical bulk surface-tension curve and the actual surface-tension curves for these liquids is a maximum in the neighborhood of this same concentration, confirming the ideal of Fig. 14. Similar results were obtained with water solutions of Barrett No. 4 creosote.

These tests show that saturation marks a critical point in the frothing of solutions and that beyond the saturation point the mixture behaves
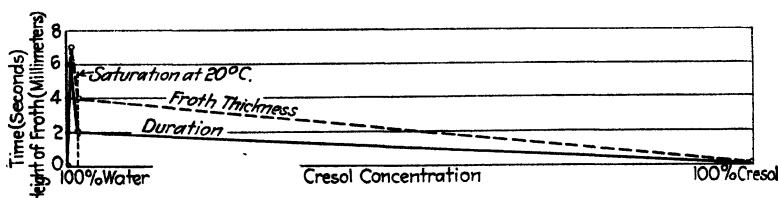
FIG. 15.—FROTHING OF WATER-CRESOL MIXTURES.

much like a pure liquid or a mixture of two liquids mutually insoluble. This is explained on the supposition that the range in surface tension between the gas surface and the hypothetical bulk surface, or, more properly, between the static and dynamic surface tensions, is so small, in the case of saturated solutions, that too little variability in surface tension of bubble films is available for adjustment to strains induced in the bubble-making operation.
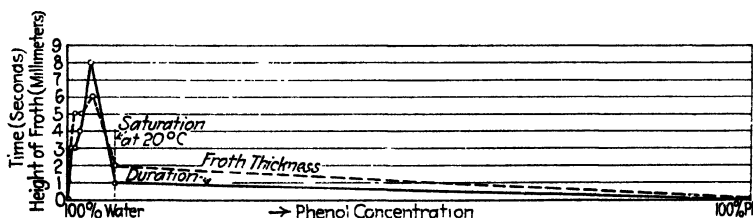
FIG. 16.—FROTHING OF WATER-PHENOL MIXTURES.

Milner's explanation[3] of the non-frothing of concentrated acetic-acid solutions is based on the time during which variable surface tension exists, rather than on the range between static and dynamic tensions. He claims that the new film formed at the surface acquires very rapidly a surface tension equal to that of the old surface film and that, therefore, time is not available for the new, higher-tensioned film to oppose the added strain on the film; hence it stretches until it breaks.

SOLUTION PHENOMENA

Surface Tension.—If there are in solution simultaneously the solutes A and B (but no immiscible substance) which independently depress the surface tension of water by T_A and T_B dynes per cm. respectively, the

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[3] Milner, *Phil. Mag.* (1907) **13**, 96.

most probable relation between their combined depression and that due to each acting individually is that the depression due to $(A + B)$ is $(T_A + T_B)$. To confirm this hypothesis, different mixtures composed of solutions of cresol (U.S.P., a mixture of the three cresols) and c.p. phenol were used. The experimental data are summarized in the ternary diagram, Fig. 17. The results indicate that in the main the depression for $(A + B)$ was equal to $(T_A + T_B)$, but that for certain
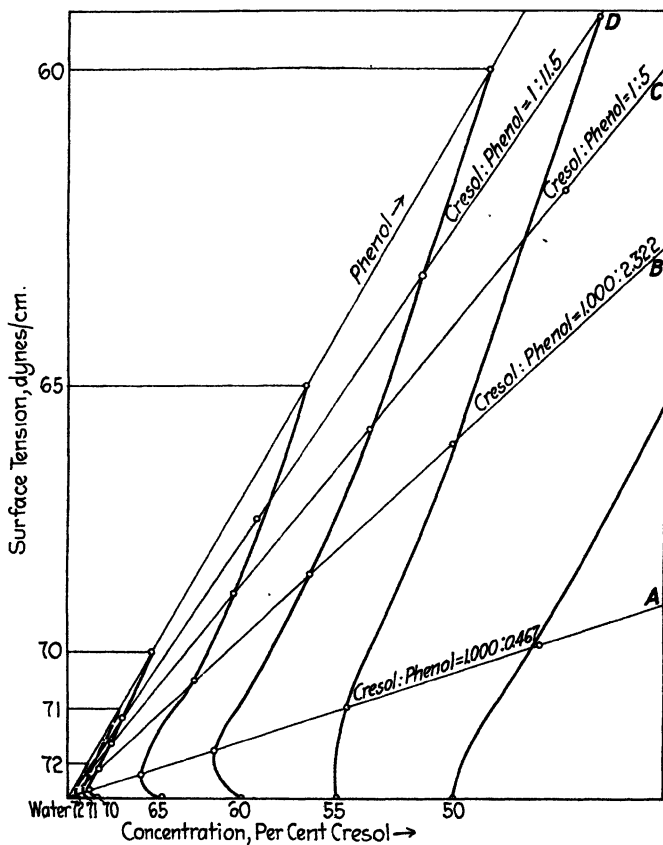
FIG. 17.—PHENOL–CRESOL–WATER TERNARY DIAGRAM; LINES OF EQUAL SURFACE TENSION. NUMBERS REFER TO SURFACE TENSION IN DYNES PER CENTIMETER.

mixtures of phenol and cresol the depression was greater than the sum of the individual depressions. The data at hand are insufficient to warrant an explanation of the apparently consistent deviations of the lines of equal surface tension along line OA from the straight lines that would represent adherence to the additive rule.

A similar series of tests with fractions obtained by distillation from the tar acids extracted from International creosote (a coal-tar product)

gave the results shown in Table 1. In all cases, the total concentration of solute in the water was 150 parts per million.

TABLE 1.—*Depression of Surface Tension by Tar-acid Fractions of International Creosote, Acting Alone and Acting Together*

Fraction	Distilling Temperature, Degrees C.	Surface Tension of Solution, Dynes per Cm.	Depression in Surface Tension Below that of Water, Dynes per Cm.	Depression to be Expected from Additive Rule, Dynes per Cm.
C..........................	205–214	71.1	1.7	
D..........................	214–225	71.4	1.4	
E..........................	225–260	68.5	4.3	
90% D + 10% E............		70.7	2.1	1.69
75% D + 25% E............		68.0	4.8	2.12
50% D + 50% E............		68.0	4.8	2.85
33% C + 33% D + 33% E....		70.1	2.7	2.54

The experimental results are not closely concordant with the theoretical, but show the same general trend.

Szyskowski[4] found an additive rule to hold with mixtures of several fatty acids, but not in co-presence of a fatty acid and its salt. In the latter instance, the depression in surface tension is larger than the sum of the depressions caused by the solutes individually. He explains this discrepancy as caused by a decrease in the ionization of the acid on addition of its salt.

Solutions containing mixtures of tar acids and tar bases from International creosote show discrepancies in the reverse direction. The depressions in surface tension of solutions containing respectively 0.0204 per cent. of tar acids, and 0.0102 per cent. of tar bases, were 11.1 dynes and 8.9 dynes per cm. A solution containing 0.0204 per cent. of tar acids with 0.0102 per cent. of tar bases exhibited a drop in surface tension of only 9.5 dynes per cm. as against 20 (11.1 + 8.9) dynes per cm. expected from the additive rule. The explanation may lie in the fact that tar acids and tar bases form compounds,[5] and, therefore, do not act independently to affect the surface tension.

Adsorption at Air-liquid Surface.—From Gibbs's equation, knowing the number of molecules[6] per gram-molecule ($N = 6.062 \times 10^{23}$), it is possible to calculate the number of molecules present in the surface film per square centimeter. Thus, from the surface tension-concentra-

[4] *Loc. cit.*

[5] W. H. Hatcher and F. W. Skirrow, *Jnl.* Am. Chem. Soc. (Sept., 1917) **39**, 1939–1978.

[6] R. A. Millikan, *Jnl.* Amer. Chem. Soc. (Sept., 1917) **39**.

tion curves of phenol solution (Figs. 9 and 10), at point A (Fig. 9) $\frac{dT}{dc} = \frac{-5.9}{0.437}$; $c = 0.437$; and, from equation (1), $U = -\frac{0.437}{8.32 \times 10^7 \times 293}$ $\times \frac{(-5.9)}{0.437} = 2.42 \times 10^{-10}$. Multiplying UN gives 14.66×10^{13}, as the excess in number of molecules per square centimeter of surface film over the number in an equal area in the bulk of the solution. Similarly, at point B (Fig. 9); $U = 4.80 \times 10^{-10}$, $UN = 29.1 \times 10^{13}$. At point C (Fig. 10); $U = 5.91 \times 10^{-10}$, $UN = 35.8 \times 10^{13}$. The spacing of the excess phenol molecules in the surface film is, therefore, at A, one phenol molecule per 68.2×10^{-16} sq. cm.; at B, one phenol molecule per 34.4×10^{-16} sq. cm.; at C, one phenol molecule per 27.9×10^{-16} sq. cm.
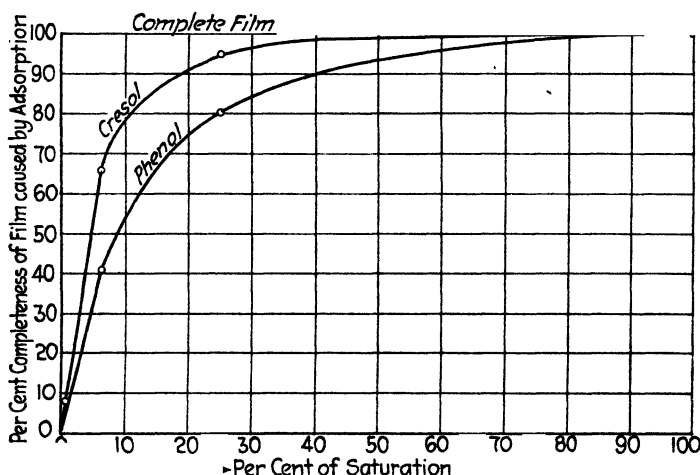
FIG. 18.—RELATION BETWEEN CONCENTRATION OF SOLUTE AND COMPLETENESS OF ADSORBED FILM OF PHENOL AND CRESOL.

A gram-molecule of phenol (C_6H_5OH) weighs 94 gm. and occupies a volume of 88.7 c.c.; or there are 6.85×10^{21} molecules per cubic centimeter. Assuming, for the moment, cubical molecules and close packing, the volume of a molecule is 146×10^{-24} c.c.; its edge would be 5.27×10^{-8} cm.; and it would occupy an area of 27.8×10^{-16} sq. cm. Assuming a molecule having the dimensions a, a, and $2a$, standing on end, the area occupied by each molecule would be 17.6×10^{-16}. Other assumptions involving increased length of molecule would, of course, result in further decrease in the area per molecule, but a marked increase in length would be required to change the order of magnitude of the area from that arrived at on the assumption of the cubical molecule. Accepting, therefore, for the purpose of these calculations, the cubical assumption, the area per molecule is substantially the same as that determined above as the spacing of the excess molecules at C.

The foregoing figures lead to the conclusion that the point C corresponds to a complete monomolecular film of adsorbed solute at the surface. On the same assumptions, the film at B is 81 per cent. complete, and at A, 41 per cent. complete.

The results of these calculations are plotted in Fig. 18, and in the same figure are plotted similar results obtained with cresol. The curves indicate that as the concentration of a solute in the bulk of a solution increases, its concentration in the adsorbed film increases, at first rapidly in proportion to the increase in concentration in the bulk, then more and more slowly, until the adsorbed film is a complete monomolecular film of solute. It also indicates that the rate of adsorption increases, at least for this pair of homologs, as the solubility decreases. This relation is further indicated by the fact that with yet higher homologs of phenol
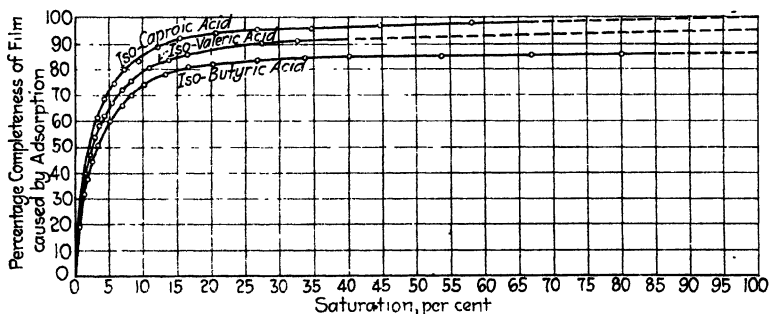
FIG. 19.—RELATION BETWEEN CONCENTRATION OF SOLUTE AND COMPLETENESS OF ADSORBED FILM IN CERTAIN FATTY ACIDS.

the lowering of surface tension for a given weight is greater than for cresol, and, in the case of minute quantities, the frothing effect is also greater.

Fig. 19, derived from data published by Szyskowski,[7] confirms the conclusions stated in the preceding paragraph for homologs of one of the groups of fatty acids.

EMULSION PHENOMENA

Surface Tension.—Few, if any, oils are completely insoluble in water; the least soluble are the higher aliphatic hydrocarbons. Of these, Nujol is the most convenient to use because it is remarkably free from soluble impurities and does not evaporate to an appreciable extent.

Several measurements of the surface tension of an emulsion of Nujol in water, 125 parts of oil per million (1.4 lb. per ton of water), indicated that Nujol has no effect on the surface tension of water. The average surface tension of the emulsion was 72.6 against 72.8 dynes per cm. for water, which discrepancy is within the limits of accuracy of the instru-

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[7] *Loc. cit.*

ment. More concentrated emulsions could not be tested because they broke down readily, giving a surface layer, not a film, of Nujol and a dilute emulsion at the bottom.

The effect of the presence of Nujol on the surface tension of solutions was next investigated, under two conditions: Nujol and the water solute are not appreciably soluble in each other; Nujol and the water solute are miscible. As can be readily appreciated, this classification is not rigorous. Solutes of the first class are plentiful; but of the second, few were found; and of these, the most satisfactory was methyl salicylate. (artificial oil of wintergreen).

First Type of Soluble-insoluble Systems.—For this investigation, cresol and Nujol were used. Results are presented in Table 2. These

TABLE 2.—*Surface Tension of Cresol-Nujol-Water Mixtures*

Ratio: $\frac{Nujol}{Cresol}$	Concentration		Surface Tension		Differences in Surface Tension, Dynes per Cm.
	Cresol, Parts per Million	Nujol, Parts per Million	Cresol Alone, from Fig. 7, Dynes per Cm.	Cresol + Nujol, Observed, Dynes per Cm.	
1/20	2500.0	125.0	58.9	58.8	−0.1
	1250.0	62.5	65.7	65.4	−0.3
	625.0	31.2	69.2	69.7	+0.5
	312.0	15.6	70.9	70.9	
	156.0	7.8	71.8	71.6	−0.2
	78.0	3.9	72.2	72.2	
1/30	187.5	62.5	71.6	71.7	+0.1
1	62.5	62.5	72.3	72.2	−0.1
4	25.0	100.0	72.5	72.5	·

Total difference... −0.1
Average difference... −0.01

TABLE 3.—*Surface Tension of Methyl Salicylate-Nujol-Water Mixtures*

Concentration of Methyl Salicylate, Parts per Million	Surface Tension Depression Below that of Pure Water, Dynes per Cm.		
	All Methyl Salicylate	75 Per Cent. Methyl Salicylate 25 Per Cent. Nujol	50 Per Cent. Methyl Salicylate 50 Per Cent. Nujol
270		0.8	
295	1.3		0.6
590	9.2	8.8	

results indicate that in such a system Nujol has no effect on the surface tension.

Second Type of Soluble-insoluble Systems.—In this investigation, some trouble was experienced because of the volatility of the available soluble substances. In this respect methyl salicylate is not entirely satisfactory but it is the least volatile solute tested. In order to reduce error due to volatility, several readings were taken for each concentration on several samples of the solution or emulsion under investigation. The average results are arranged in Table 3. The maximum deviation from average was 0.35 dyne and the average deviation 0.18. It should be noted that the soluble and insoluble substances added to the water were first dissolved in each other, then emulsified in the water.

The table shows that instead of increasing the depression of surface tension the Nujol reduced it, which result may be explained as follows: There are present in the emulsion, water as the dispersing medium and Nujol as the dispersed phase, with methyl salicylate dissolved in both and probably, also, adsorbed at the interface between them. Methyl salicylate is much more soluble in Nujol than in water; therefore, when equilibrium is reached, the concentration of the solute in Nujol is much larger than in water. Hence only a part of the methyl salicylate is active as solute in the water; the other part is held by the Nujol and does not affect the surface tension of the emulsion. The depression of surface tension caused by a given amount of methyl salicylate when Nujol is present is, therefore, less than that caused by the same amount of methyl salicylate present alone.

The same phenomenon is exhibited by International creosote. The drop in surface tension due to the whole oil is 4.1 dynes per cm. for a concentration of 0.3 per cent.; the tar acids contained in the same amount of oil, alone, cause a drop in surface tension of 11.1 dynes per cm.; the tar bases, 8.9 dynes per cm.; and the so-called neutral oil, 0.8 dyne per cm. Here the neutral oil holds large amounts of tar acids and tar bases, which cannot, therefore, affect surface tension. It should be noted that this so-called neutral oil is not completely insoluble in water.

Adsorption at Interface of Dispersed and Dispersing Phases.—It is to be noted that in the foregoing experiment, part of the methyl salicylate must be present as an adsorbed film around the Nujol globules, just as methyl salicylate is adsorbed at the air-liquid interface. The chain of equilibria may be represented as follows: Let A = liquid insoluble in water, which forms dispersed phase, and B = liquid soluble both in water and in A. If the amount of dissolved substance is diminished
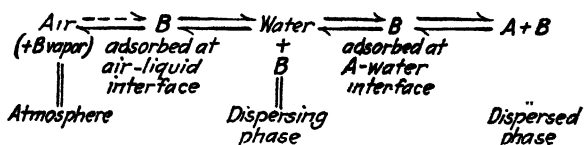
in the dispersing phase, more of this substance will pass in solution from the dispersed phase, which thus acts as a sort of self-adjusting reservoir for the dissolved substance.

Surface Tension-concentration Curves for Soluble-insoluble Systems of Second Type.—In the present work on surface tension of solutions early experiments showed that the surface tension-concentration curve with concentrations plotted on a logarithmic scale is always convex in the region of low concentration (see Figs. 6, 8 and 10). Emulsions, on the contrary, give curves alternately convex and concave (see Fig. 20). This is explained by the fact that the portion of an oil undissolved at a certain concentration, say 0.5 lb. oil per ton of water, dissolves partly in a more dilute solution, such as 0.05 lb. oil per ton of water. If the undissolved part of the emulsion were taken out of the system and agitated with a new portion of water, another portion of the oil would dissolve. If, then, a certain emulsion is diluted to, say, one-quarter of its former concentration, the soluble portion is diluted to one-quarter of its former concentration, but at the same time some of the previously undissolved portion is dissolved and the depression of the surface tension is larger than would otherwise be expected; the concentration-surface tension curve will sag more, and, if plotted with logarithmic abscissas, will be partly concave and partly convex. It may, then, be said that in the case of complex oils, such as coal-tar oils, pine oils, etc., the parts that are soluble in dilute emulsions and not soluble in more concentrated emulsions constitute the reserves. When the emulsion is diluted, either through addition of water or removal of the dissolved portion, part of the undissolved oil goes into solution and a given frothing operation is thus enabled to continue.

FIG. 20.—SURFACE TENSION-CONCEN-TRATION CURVE FOR BARRETT NO. 4 EMULSIONS.

Frothing.—Pure emulsions, such as a Nujol emulsion, do not froth at all. Emulsions of an oil that is partly soluble do froth, but less than they would, for a given concentration, if the insoluble portion were absent. This is shown by comparison of the curves, Figs. 21 and 22, for the soluble and insoluble-plus-soluble portions of Barrett No. 4, and Figs. 23 and 24 for methyl salicylate solution and methyl salicylate-Nujol-
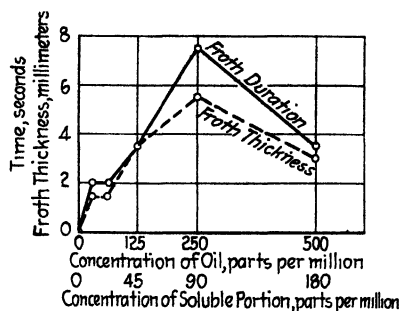
FIG. 21.—FROTHING OF SOLUTIONS OF SOLUBLE PART OF BARRETT NO. 4 CREOSOTE IN WATER.
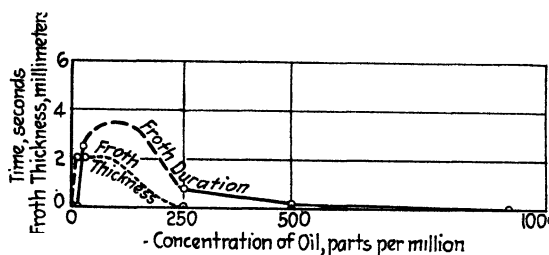
FIG. 22.—FROTHING OF EMULSIONS OF BARRETT NO. 4 CREOSOTE IN WATER.
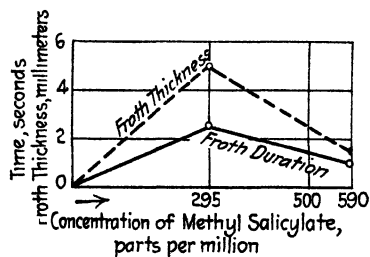
FIG. 23.—FROTHING OF SOLUTIONS OF METHYL SALICYLATE IN WATER.
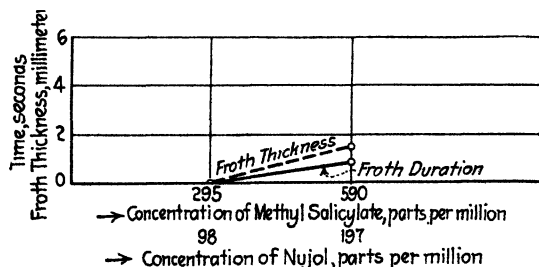
FIG. 24.—FROTHING OF MIXTURES OF METHYL SALICYLATE AND NUJOL EMULSIFIED WITH WATER.

water mixtures. An insoluble dispersed substance has a dampening effect on the frothing power of a solute. Note, however, that here the stabilizing effect of the undissolved oil is not considered. It may well be that this stabilization will give an appearance of greater frothing in the case of some oils with the undissolved portion present.

Conclusions as to Surface Tension and Adsorption of Soluble-insoluble Systems.—1. Insoluble oils do not affect surface tension and do not cause frothing in a system composed solely of water and the insoluble oil.

2. In a system composed of water, a water-soluble substance, and a water-insoluble substance, in which the latter two are not mutually soluble, the substance insoluble in water has no effect on the surface tension of the system.

3. In a system composed of water, a water-soluble substance, and a water-insoluble substance, in which system the latter two members are mutually soluble, the substance insoluble in water dissolves some of the substance soluble in water, thus preventing the latter from exerting its full effect on the surface tension of the water.

4. In such a system as the last mentioned, the substance insoluble in water acts as a dampener on the frothing power of the solution, particularly when the solution is dilute.

5. The soluble substance is dissolved in water and in the insoluble substance and is also adsorbed at the interface between these, establishing a chain of equilibria in which the water-insoluble substance acts as a self-adjusting reservoir for the water-soluble substance.

6. In the case of complex flotation oils, the undissolved portion may partly dissolve, if sufficiently diluted, in which case it acts as a reservoir to supply soluble oil to replace that removed by frothing.

Surface-tension Effect of Oleic Acid and Similar Substances.—Oleic acid and certain other higher fatty acids are reputed insoluble; yet, in minute amounts, many of them, including oleic, reduce the surface tension of water considerably. If a small amount of oleic acid is dissolved in benzene and a drop of the solution is placed on a large water surface, the drop spreads, the benzene evaporates and leaves the surface covered with oleic acid. If the surface is dusted with talcum powder and then blown, the talcum powder gathers in a certain definite area, indicating a film of oleic acid over that area; whereas, when no oleic acid is added and pure water is used, the powder can be blown to a negligible area. If the amount of oleic acid contained in the benzene drop is known and the number of molecules per gram-molecule is taken into account, computation shows that the minimum area on which the talcum powder can be condensed is approximately equal to the area that the oleic acid in the drop would cover, if spread to a monomolecular film. The word "approximately" is used here because of the implicit assumption that the molecules are equidimensional.

The surface tension of a water-surface to which oleic acid is added in this fashion is found to differ little from that of water, so long as the oleic acid is insufficient to cover the surface with a monomolecular film. As soon, however, as this point is reached, the surface tension begins to drop and quickly reaches a value near 50 dynes per cm., decreasing but slightly with further addition of oleic acid. The detailed procedure follows: An oleic-acid solution in benzene containing 1.95×10^{-3} mg. of oleic acid per drop of benzene was prepared. The area tested was 135 sq. cm. The height of the oleic-acid molecules, according to Langmuir,[8] being 10.8×10^{-8} cm., the weight of a film of oleic acid one molecule deep over this area is 12.75×10^{-3} mg. Between six and seven drops of the benzene solution were, therefore, required. As the benzene was dropped on the water, the oil spread instantly with iridescence and immediate evaporation, up to the sixth drop. The seventh drop spread with difficulty. The eighth drop did not spread but formed a large flat lens which resolved itself into several small drops as the benzene evaporated. From the ninth drop onward, the benzene solution remained on the surface as a globule for a considerable time, the benzene evaporating slowly. The surface-tension readings are condensed in Table 4.

TABLE 4.—*Effect of Oleic Acid on Surface Tension of Water*

Number of Drops of Benzene Solution Added	Weight of Oleic Acid Added, Mg. $\times 10^{-3}$	Per Cent. of Complete Monomolecular Film Present	Surface Tension, Dynes per Cm.	Depression of Surface Tension Below that of Water (72.8 Dynes per Cm.)
1	1.95	15.3	72.1	0.7
2	3.90	30.6	71.9	0.9
3	5.85	45.9	71.8	1.0
4	7.80	61.2	71.7	1.1
5	9.75	76.5	71.6	1.2
6	11.70	91.8	71.5	1.3
7	13.65	107.0	67.7	5.1
8	15.60	122.3	63.3	9.5
9	17.55	137.8	62.0	10.8

If a certain time is allowed to elapse after the evaporation of the benzene and before measuring the surface tension, the latter is found to increase. Results of observations are given in Table 5. The cause of this increase may be: (1) Some benzene is retained by the oleic acid and evaporates but slowly; (2) oleic acid evaporates; (3) oleic acid dissolves; (4) oleic acid changes chemically. Of these reasons, (2) and (3) seem to be the most plausible because the surface tension with any given number of added drops increases with time of standing at a rapidly decreasing rate and such curves as Fig. 25, showing the change in surface tension

[8] Fundamental Properties of Liquids and Solids, Part 2, *Jnl.* Am. Chem. Soc. (1917) **39**, 1848–1906.

TABLE 5.—*Effect of Time of Standing on Surface Tension of Oleic Acid-water Mixtures*

Amount of Oleic Acid-benzene Solution Added	Surface Tension Measured	Surface Tension, Dynes per Cm.
18 drops	Immediately after evaporation of benzene.......	54.3
	After standing 10 min..........................	59.0
	After standing 40 min..........................	63.5
24 drops	Immediately after evaporation of benzene.......	53.5
	After standing 2 hr...........................	64.0
	After standing 15 hr..........................	64.8
32 drops	Immediately after evaporation of benzene.......	48.4
	After standing 15 min.........................	51.5
	After standing 45 min.........................	55.2
	After standing 75 min.........................	58.7
	After standing 2 hr...........................	61.8
	After standing 3 hr...........................	62.9
	After standing 48 hr..........................	63.2

with time plotted from the last division of Table 5, and are typical of evaporation and solution, the curve being asymptotic to a line representing equilibrium under the existing conditions.
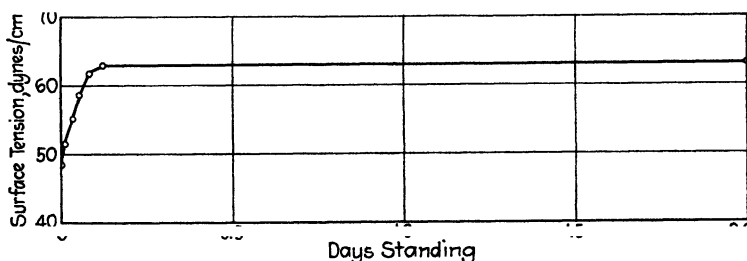
FIG. 25.—EFFECT OF TIME OF STANDING ON OLEIC ACID-WATER MIXTURES.

Because of this change in surface tension with time, and of the questionable purity of the benzene used, a slight modification was introduced in the experiment. The oleic acid, instead of being added as a solution in benzene, was added directly by dipping a looped platinum wire in the oil and then touching the water to be contaminated. The excess oil was first discarded by vigorously shaking the platinum dipper after taking the oil. The results are embodied in Fig. 26. Weights of oleic acid are estimated by determining the average area of film formed by the oil added by each touch of the dipper, and calculating weights from this figure and the numbers of dips, using the data in Table 4. Comparing this curve with the concentration-surface tension curve of a substance completely soluble in water, such as acetic acid or phenol, shows that a

monomolecular film of oleic acid decreases the surface tension by only
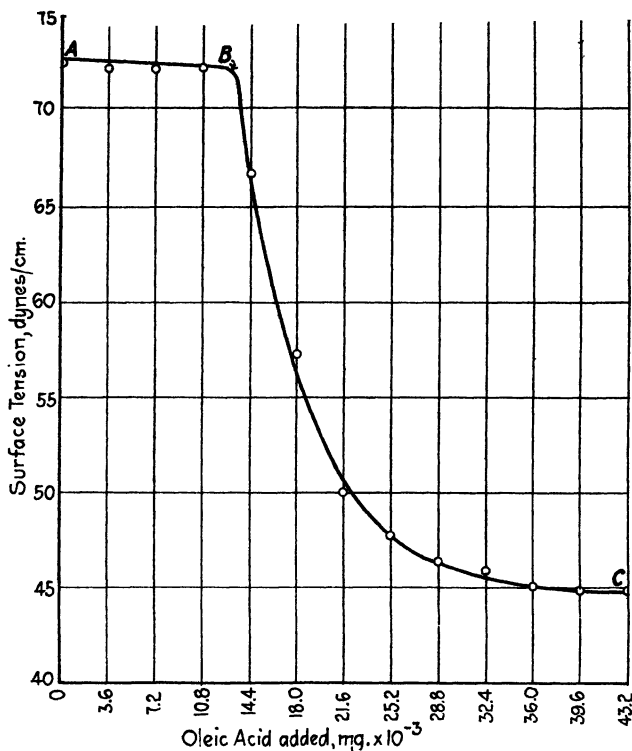
FIG. 26.—SURFACE TENSION-CONCENTRATION CURVE FOR OLEIC-ACID FILMS ON WATER.
AREA TESTED 135 SQ. CM.
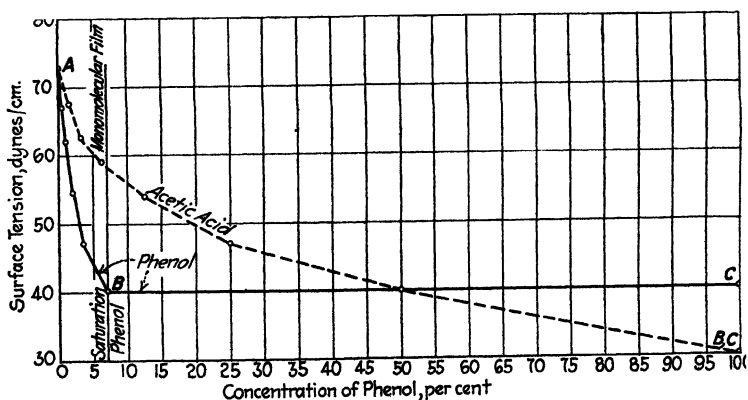
FIG. 27.—SURFACE TENSION-CONCENTRATION CURVES FOR WHOLLY SOLUBLE SUB-
STANCES IN WATER.

0.9 dyne per cm., while in the case of phenol or acetic acid the decrease
is much larger. The portion AB of the curve (Fig. 26) for oleic acid

corresponds to the portion AB (Fig. 27) for a completely soluble substance, inasmuch as in both cases B corresponds to a monomolecular film. B and C may coincide, as is the case with acetic acid; they may be on the same level but not coincident, as is substantially the case with cresol and phenol; and Marcelin[9] finds that there are a number of substances for which the depression in surface tension at saturation is intermediate between the depression caused by oleic acid and that caused by a highly soluble substance. This strongly points to the conclusion that the difference between oleic acid and these solutes is one of degree rather than of kind.

TABLE 6.—*Behavior of Various Flotation Oils at Water Surface*

Observations	Oleic Acid	Barrett No. 4	Pine Oil G. N. S. No. 5	Cresol U. S. P.
Vigor of spreading	Great	Much smaller	Slightly larger than Barret No. 4	Small
Decrease in size of drop on standing	No decrease observed	Decreases slowly	Decreases fairly rapidly	Vanishes in a few seconds
Number of drops to get "monomolecular-film" effect	One drop is just too much	9 to 10	About 30	25 drops cover but small fraction of available area

Another line of reasoning that leads to the same conclusions is based on the following experiment: A platinum wire with looped end was dipped in an oil and the drop was carefully placed on the surface of water in a clean photographic tray 5 by 9 in. The water was previously cleaned by dusting with talcum powder and crowding the surface film to one end of the tray by means of a strip of clean non-porous paper. The results are condensed in Table 6, which shows that the same phenomena occur with the four substances tested, although in different degrees; and increasing solubility of the substances named from left to right in the table is indicated.

ADSORPTION AT AIR-LIQUID SURFACES

Experiment.—Four 2000-c.c. portions of a solution containing 0.50 c.c. pine oil (0.023 per cent.) and 1 c.c. sulfuric acid per 2000 c.c. of water were obtained by a 10-min. agitation in the square glass jar apparatus, followed by filtering through 5 gm. of infusorial earth. The combined filtrate was placed in a Callow cell (laboratory size) and the air turned on. About 3000 c.c. of solution was collected as a froth. This was returned to the emptied cell and run again, 1800 c.c. of liquid being overflowed as

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[9] Marcelin, *Comptes rendus* (1921) **173**, 38–41, and 79–82.

froth. This was again returned and 800 c.c. of solution overflowed. Surface-tension determinations were made as follows:

Original solution................ 63.5 dynes per cm.
Residual solution from first run.. 65.3 dynes per cm.
Overflow from third run......... 62.2 dynes per cm.

Taking evaporation losses into consideration, these differences in surface tension clearly indicate that the concentration of the oil is greater in the froth than in the bulk of the solution, and indicate adsorption at the air-liquid surfaces.

Slightly different tests were conducted in the square glass jar agitation machine. While agitating 0.50 c.c. of pine oil with 1 c.c. of sulfuric acid in 2400 c.c. of water, the liquid was sampled by siphoning during agitation, while the beater revolved at about 2000 r.p.m. It was found that the surface tension of the liquid thus drawn off was appreciably lower than that of a sample taken similarly from the same body of liquid at rest. This result could be caused by one or more of the following phenomena: (1) Evaporation of part of the oil between the two sampling operations. (2) Collection of the oil at the top of the jar, at rest, causing corresponding decrease in concentration of the oil in bulk and thus in the sample taken at rest. (3) Adsorption of the oil on minute air bubbles during agitation, followed by selective drawing of these bubbles by the siphon.

To decide which of these was the real cause of the discrepancy, the following tests were carried out:

1. Concentration of pine oil, 0.019 per cent. on the water. (a) Surface tension of sample taken just before stopping an agitation lasting for 10 min. at 2000 r.p.m., 49.4 dynes per cm. (b) Surface tension of sample taken after cessation of agitation, 62.9 dynes per cm. (c) Surface tension of same sample after filtering twice through 10 gm. of infusorial earth, 62.5 dynes per cm.

2. Concentration of pine oil, 0.0076 per cent. on the water. (a) Surface tension of sample taken just before stopping an agitation lasting for 10 min. at 2000 r.p.m., 56.7 dynes per cm. (b) Surface tension of sample taken during agitation at 500 r.p.m., 64.0 dynes per cm. (c) Surface tension of sample taken after cessation of agitation from bottom of jar, 67.3 dynes per cm. (d) Surface tension of sample taken after agitation, from top of jar, 67.3 dynes per cm. (e) Surface tension of sample taken after agitation and filtering through 10 gm. of infusorial earth, 66.8 dynes per cm.

3. Concentration of pine oil, 0.019 per cent. on the water. (a) Surface tension of sample taken just before stopping an agitation lasting for 10 min. at 2000 r.p.m., 47.3 dynes per cm. (compare 49.4 dynes per cm. in first test.) (b) Surface tension of sample taken after cessation of agitation, 62.5 dynes per cm. (compare 62.9 dynes per cm. in first test). (c)

Surface tension of sample taken during a second agitation, after the above samples were taken, agitation at about 500 r.p.m., 56.0 dynes per cm. (*d*) Surface tension of sample taken just before the end of a second 10-min. period of agitation at 2000 r.p.m., 53.5 dynes per cm. (compare with 47.3 dynes per cm., test 3*a*).

These results, particularly those of the third test, show that samples taken from the body of the liquid during agitation contain considerably more soluble oil than is contained in the body of the liquid at rest. This cannot be ascribed to evaporation, inasmuch as the discrepancy existed whether the sample during the agitation was taken before or after the sample at rest. The fact that the surface tension of a sample from very near the top of the liquid in the jar at rest was exactly equal to that of a sample from the bottom of the jar, argues against an uneven distribution of the oil throughout the jar at rest. On the other hand, the presence of extremely numerous and minute air bubbles cannot be doubted, and while they increase the bulk of the emulsion only slightly, the larger bubbles formed by their coalescence in the siphon represent a considerable part of the bulk of the fluid therein. It is, therefore, easily conceivable that the siphon drew a greater proportion of air bubbles than was present in the bulk of the liquid, and that the sample thus obtained had the characteristics of a froth, that is, it was richer in oil than the bulk of the liquid.

ADSORPTION AT SOLID SURFACES

Manipulation and Calculations.—The adsorbents used were pure minerals and ores ground and sized. At first when small quantities only were desired, they were prepared by hand grinding in a porcelain mortar, but this proved a tedious task when several pounds of closely sized quartz averaging 0.1 mm. diameter were desired. Hand grinding was then replaced by a laboratory disk pulverizer with dry screening through Tyler standard screens. This introduced an appreciable amount of metallic iron but not enough to show an effect on the results.

In the first tests, the liquid was placed in a 250-c.c. Erlenmeyer flask and the adsorbent was added. Then the flask was agitated, usually one shake for each 2 gm. of adsorbent, and the suspension was allowed to settle. The supernatant liquid was decanted and its surface tension measured. Oftentimes so much adsorbent was floated that filtering was necessary, to insure accurate surface-tension determination. After considerable use this method was abandoned for the following: A 400-c.c. portion of a solution taken from a 2000 or 2500-c.c. preparation was placed in the square glass jar apparatus with the desired amount of adsorbent and agitated for 10 min. at 1700 r.p.m. This mixture was then filtered through a 5 to 10-gm. bed of infusorial earth, using a 25-in. vacuum. The surface tensions of the feed and product solutions were

measured. A blank test was then run with another 400 c.c. of feed solution as above, except that no adsorbent was added and surface-tension measurements were taken on the solution before and after treatment. If T_F be the depression for the feed solution in both cases, T_B the depression for the blank product solution, T_P the depression for the product solution, the corresponding concentrations are proportional (on the assumption of a straight-line law) or:

$$\frac{C_F}{T_F} = \frac{C_B}{T_B} = \frac{C_P}{T_P}$$

The reduction in concentration by adsorption, $C_B - C_P$, is then:

$$C_B - C_P = \frac{T_B - T_P}{T_F} \times C_F$$

The amount adsorbed, M, is:

$$M = \frac{T_B - T_P}{T_F} \times Q_F$$

Q_F being the amount of oil present in the original solution when the surface tension was measured. This, however, is not by any means the amount of oil introduced into the water when the feed solution was prepared. Part of this oil was removed by the filter as undissolved droplets, part was removed by evaporation during the preparation of the feed solution, and part was adsorbed by the infusorial earth. The actual amount of oil in solution can be either determined or estimated. The percentage lost by evaporation and adsorbed by the infusorial earth can be determined from the blank test as follows: Through this blank treatment the proportion of oil removed was

$$\frac{C_F - C_B}{C_F} = \frac{T_F - T_B}{T_F}$$

This applies to 400 c.c. Reckoned on the 2000 or 2400 c.c present at the beginning, the proportion is less; one-quarter would seem a fair figure to use. Q_F can then be computed from the amount of oil Q put into the jar with, say, 2000 c.c. of water as follows:

$$Q_F = Q \times \frac{400}{2000} \times \frac{T_F - T_B}{4\,T_F} \times S$$

where S = solubility of oil used, as independently determined.,

To determine the average thickness of the adsorbed film, it is necessary to know the surface of the adsorbent used. This cannot be accurately determined because the minerals do not break in regular shapes. However, most minerals break with a certain habit, galena fragments being generally cubical; sphalerite tetrahedral; calcite rhombohedral; pyrite and chalcopyrite granular and more or less equidimensional; chalcocite and·quartz, flattened and irregular. Galena fragments passing 100 mesh and staying on 150 mesh (Tyler) will have an average diameter approxi-

TABLE 7.—*Areas per Gram of Finely Ground Mineral Particles*

Mineral	Shape-correcting Multiplier	Specific Gravity	Approximate Area (sq. cm.) per Gram							
			−200-mesh, Tyler. Average Size			−150 +200 Mesh Tyler	−100 +200 Mesh Tyler	−100 +150 Mesh Tyler	−65 +150 Mesh Tyler	−65 +100 Mesh Tyler
			0.03 Mm.	0.035 Mm.	0.04 Mm.					
Galena..........	1.0	7.5	267			90	73	64	51	45
Quartz..........	1.6	2.5		1095		430	350	310	240	215
Chalcocite.......	1.4	5.7			370	165	133	118	93	83
Sphalerite........	1.2	4.0	600			200	165	145	115	100
Chalcopyrite......	1.3	4.2			465	210	170	150	120	105
Pyrite...........	1.2	5.0			360	160	130	115	92	81
Calcite..........	1.3	2.7		825		325	265	230	185	160

mately the mean between the openings of the two screens, or 0.125 mm. Assuming all the particles to be cubes of this size, the total area per gram is 64 sq. cm. For any other mineral, this number should be multiplied by the ratio of the specific gravity of galena to that of the mineral, and by a factor determined by the habit of the broken material. Table 7 presents the factors used and the approximate areas calculated as described. It will be noticed that for material below 200 mesh the column has been subdivided and the figures for various minerals have been put into columns headed by different figures for average size. This is done because the materials break differently, depending on the hardness, brittleness and breaking habit, and the average size will, therefore, vary according to the mineral. Table 7 does not take into account the possibility that some of the material may approach the colloidal state, which would greatly increase the surface. One per cent. of colloidal material averaging 0.1μ in diameter in the material finer than 200 mesh would increase the surface 350 per cent. This is, perhaps, a serious omission, but no exact data are at hand to correct it. On the other hand, with the pure minerals used in preparing solids for the adsorption tests, little if any colloidal material is formed in grinding and any error thus introduced probably affects all the results cited in much the same way and order, so that relative results are not seriously affected.

TABLE 8.—*Adsorption of Pine Oil from Emulsions by Solids*

Concentration of Oil (Pine Oil), Parts per Million	Adsorbent	Square Centimeters of Adsorbent Surface per Gram of Oil	Surface Tension		Difference in Surface Tension	Grams of Oil Adsorbed per Square Centimeter of Adsorbent Surface
			Before Agitation with Adsorbent	After Agitation with Adsorbent		
19.6	Quartz......	8.27×10^6	68.1	69.2	+1.1	2.8×10^{-8}
19.6	Galena......	5.75×10^6	66.4	71.7	+5.3	14.5×10^{-8}
19.6	Galena......	3.66×10^6	65.1	70.8	+5.7	20.2×10^{-8}

TABLE 9.—*Adsorption of Hardwood Creosote from Emulsions by Solids*

Concentration of Oil (Cleveland Cliffs), Parts per Million	Adsorbent	Square Centimeters of Adsorbent Surface per Gram of Oil	Surface Tension		Difference in Surface Tension	Grams of Oil Adsorbed per Square Centimeter of Adsorbent Surface
			Before Agitation with Adsorbent	After Agitation with Adsorbent		
42.4	Galena......	1.83×10^6	68.9	72.0	+3.1	62.0×10^{-8}
42.4	Quartz......	5.09×10^6	67.3	71.4	+4.1	14.5×10^{-8}
42.4	Galena......	0.55×10^6	68.7	71.4	+2.7	120.0×10^{-8}
42.4	Galena......	0.04×10^6	66.7	67.4	+0.7	303.0×10^{-8}
84.8	Galena......	0.85×10^6	66.7	70.4	+3.7	72.0×10^{-8}
84.8	Galena......	0.42×10^6	65.2	70.2	+5.0	156.0×10^{-8}
84.8	Galena......	0.21×10^6	65.5	66.9	+1.4	89.0×10^{-8}
84.8	Galena......	0.04×10^6	61.3	61.6	+0.3	68.0×10^{-8}
250.0	Pyrite	0.287×10^6	55.8	60.5	+4.7	97.0×10^{-8}
250.0	Galena......	0.287×10^6	55.8	61.1	+5.3	108.0×10^{-8}
250.0	Blende......	0.287×10^6	56.0	60.5	+4.5	94.0×10^{-8}
250.0	Quartz......	0.287×10^6	56.0	60.0	+4.0	84.0×10^{-8}
250.0	Chalcopyrite.	0.287×10^6	53.4	60.9	+7.5	136.0×10^{-8}
250.0	Chalcocite...	0.287×10^6	52.7	58.2	+5.5	94.0×10^{-8}

TABLE 10.—*Comparison of Adsorption Data by Fahrenwald with Results by Taggart and Gaudin*

Adsorbent	A. W. Fahrenwald's Data			Taggart and Gaudin Data		
	Surface Tension, Dynes per Cm.		Difference in Surface Tension	Surface Tension, Dynes per Cm.		Difference in Surface Tension
	Before Agitating with Adsorbent	After Agitating with Adsorbent		Before Agitating with Adsorbent	After Agitating with Adsorbent	
Galena	49	56.5	+ 7.5	57.0	57.5	+0.5
Sphalerite..........	49	52.0	+ 3.0	55.4	54.6	−0.8
Chalcopyrite	49	54.0	+ 5.0	55.4	56.4	+1.0
Chalcocite	49	54.5	+ 5.5	55.5	55.4	−0.1
Quartz	49	50.0	+ 1.0	55.4	54.5	−0.9
Total.............			+22.0			−0.3
Average difference..			4.4			−0.05

$$\text{a.d.} = \frac{\Sigma \text{a.d.}}{n} = \frac{3.0}{5} = 0.6$$

$$\text{A.D.} = \frac{\text{a.d.}}{2} = 0.3 \text{ dyne}$$

Amount of adsorbent used:

8 c.c. of −20-mesh, +28-mesh, freshly ground adsorbent, for every 100 c.c. of emulsion.

TABLE 11.—*Adsorption of Soluble Portion of Pine Oil at Sulfide Surface*

Concentration of Dissolved Portion of Pine Oil (G.N.S. 5), Parts per Million	Adsorbent	Square Centimeters of Adsorbent Surface per Gram of Oil	Surface Tension, Dynes per Cm.		Surface Tension Difference, Dynes per Cm.	Grams of Oil Adsorbed per Square Centimeter of Adsorbent Surface
			Before Agitating with Adsorbent	After Agitating with Adsorbent		
108.0	Galena......	1.23×10^6	61.3	63.9	+2.6	18.6×10^{-8}
108.0	Galena......	0.55×10^6	62.1	62.5	+0.4	7.2×10^{-8}
108.0	Galena......	3.70×10^6	61.1	65.1	+4.0	9.2×10^{-8}
54.0	Galena......	7.39×10^6	65.4	68.6	+3.2	4.6×10^{-8}
54.0	Galena......	1.54×10^6	65.6	67.3	+1.7	15.6×10^{-8}
54.0	Galena......	0.55×10^6	66.0	66.3	+0.3	8.0×10^{-8}
27.0	Galena......	2.46×10^6	69.3	70.0	+0.7	8.1×10^{-8}
27.0	Galena......	9.25×10^6	68.8	70.6	+1.8	4.9×10^{-8}
27.0	Galena......	1.10×10^6	69.0	69.6	+0.6	14.4×10^{-8}
13.5	Galena......	4.92×10^6	71.4	72.0	+0.6	8.7×10^{-8}
Average.	....	....	....	....	....	9.9×10^{-8}

Adsorption Tests.—The first tests were made with emulsions of pine oil (G.N.S. No. 5), Cleveland Cliffs No. 1 (hardwood creosote), and Barrett No. 4, (coal-tar creosote). These oils are typical of their classes. The results were erratic and unsatisfactory, caused, without doubt, by greasing of the knife edge in the surface-tension determinations on the original emulsion. The Erlenmeyer-flask adsorption method was used. The results, though poor, are included to give an idea of their erratic character and to impress the fact that it is useless with this apparatus to try to determine adsorption in emulsions. See Tables 8 and 9. An attempt was made to duplicate exactly Fahrenwald's tests[10] on adsorption with emulsions of hardwood creosote; our results disagree widely with his. Both sets are tabulated in Table 10, for comparison.

With the soluble portion of pine oil (G.N.S. No. 5) and the Erlenmeyer-flask method of determining adsorption more consistent results were obtained; these are condensed in Table 11.

It has been pointed out that as the work progressed refinements in procedure were introduced. The following summary of the plan developed may be useful to other experimenters:

1. Results with emulsions are erratic and unreliable; therefore work with solutions only.

2. Clean the knife edge of the surface-tension apparatus thoroughly with acetone and ether after every immersion and before weighing the recorded pull of the film.

3. Use dilute solutions only.

[10] *Loc. cit.*

4. The difference in surface tension between the solution and pure water should be as large as possible.

5. The solute should affect the surface tension of water considerably. The more the effect per unit of concentration, the better the results.

6. The results with solutes that are complex mixtures are subject to question, on account of the probability of differential adsorption.

7. If there is any evidence of chemical or physical change, as for instance a flocculent precipitate occurring after adsorption and filtering, the sample should be discarded.

8. The area of adsorbent per unit weight of solute should be as large as possible.

The fractions from the distillation of the bases and tar acids of the following coal-tar products, Barrett No. 4, International creosote, Reilly No. 19, and Lewis tar, were taken for the final adsorption tests. These tar acids and tar bases were prepared as follows: 500 c.c. of the oil was distilled in a 1000-c.c. distilling flask to separate the pitch. The liquid distillate was treated with a 200-c.c. portion of a solution of sodium hydroxide (specific gravity 1.10) by warming for several minutes, after which the mixture was poured into a separatory funnel and shaken; then allowed to settle and the sodium-hydroxide solution was drawn off. This process was repeated until the extracting solution was clear. The residual oil was treated with a sulfuric-acid solution (1 of acid, sp. gr. 1.84, to 4 of water by weight). The oil and acid were heated and shaken as above until the acid solution, after separation, was clear. The sodium-hydroxide solutions were then mixed in a large separatory funnel and extracted with successive 100-c.c. portions of benzol until the benzol separated clear. The sulfuric-acid solution was subjected to the same treatment, the purpose of which is to remove suspended tar. The tar acids were recovered by first boiling the washed sodium-hydroxide solutions vigorously and, after cooling, adding sulfuric acid until the solution was acid. The tar acids precipitated and were separated in a separatory funnel. The tar bases were similarly recovered from the sulfuric-acid solution, stick sodium hydroxide being added until the solution became alkaline.

The tar acids thus obtained were fractionally distilled in small lots in a 100-c.c. side-arm distilling flask. Fractions distilling below 190° C., from 190 to 205°, from 205 to 215°, from 215° to 225° or 230°, from 230° to 280°, and above 280°C., were collected. In all cases a tarry residue remained. The probable approximate composition of the fractions follows: Below 115°, water and some phenol; from 115° to 190°, phenol and ortho-cresol; from 190° to 205°, cresol (mostly meta- and para-) and ethyl phenols; from 205° to 215°, some cresols, ethyl phenols, xylenols; from 215° to 225°, xylenols; from 230° to 280°, homologs of phenol higher than xylenols or ethyl phenols; above 280° naphtols and their homologs.

TABLE 12.—*Adsorption at Quartz and Galena Surfaces from*

A	Solution of		Whole Tar Bases from International Creosote	Whole Tar Bases from International Creosote	Tar Bases from International Creosote			
					Fraction D (300°–360° C.)	Fraction C (260°–300° C.)	Fraction C (260°–300° C.)	Fraction B (100°–260° C.)
B	Concentration, parts per million		150	150	150	150	150	150
C	Original surface-tension depression, dynes per cm		8.3	8.3	11.8	9.5	9.5	4.2
D	Surface-tension depression after blank treatment, dynes per cm		4.2	4.2	5.5	5.8	5.8	2.9
E	Per cent. reduction in surface-tension depression through blank treatment on 400 c.c.		49	49	53	39	39	31
F	Estimated percentage of oil having evaporated before original surface tension was measured	$\frac{1}{4}E$	12	12	13	10	10	8
G	Estimated percentage of oil remaining undissolved		10	10	10	5	5	2
H	Per cent. of original oil present in water when original surface tension was measured	$10^{-2} \times (100-G) \times (100-F)$	79	79	78	85	85	90
J	Surface-tension depression after adsorption, dynes per cm		0.6	1.6	1.6	3.6	4.9	1.1
K	Reduction in surface-tension depression due to adsorption	$D-J$	3.6	2.6	3.9	2.2	0.9	1.8
L	Concentration present when original surface tension was measured, parts per million	$B \times H$	118	118	117	126	126	135
M	Per cent. reduction in original surface-tension depression through adsorption	$\frac{K}{C} \times 10^2$	43	31	33	23	9.5	43
N	Concentration removed through adsorption, grams per cubic centimeter	$L \times M \times 10^{-4} \times 10^{-2}$	51×10^{-6}	37×10^{-6}	39×10^{-6}	29×10^{-6}	12×10^{-6}	58×10^{-6}
O	Adsorbent		Minus 200-mesh quartz	Minus 200-mesh quartz	−100 +200- mesh quartz	−100 +200- mesh quartz	−100 +200- mesh quartz	Minus 200-mesh quartz
P	Adsorbent area per gram, from Table 7, square centimeters		1095	1095	350	350	350	1095
Q	Grams of adsorbent per cubic centimeter of solution		0.5	0.25	1.00	1.00	0.25	0.75
R	Area of adsorbent per cubic centimeter of solution, square centimeters	$P \times Q$	550	275	350	350	87	825
S	Amount of oil removed per square centimeter of adsorbent area, grams	$\frac{N}{R}$	9.2×10^{-8}	13.4×10^{-8}	11.1×10^{-8}	8.3×10^{-8}	13.7×10^{-8}	7.0×10^{-8}
T	Maximum percentage error that can be caused in S by an error of 0.2 dyne in each of the surface tensions measured (approximate)	$\left(\frac{0.4}{K} + \frac{0.4}{C-D}\right) \times 100$	21	25	17	29	55	53

The tar bases were similarly distilled, the temperatures at which the several fractions started and stopped being 105°, 260°, 300°, 360° C. The fraction distilling below 105° is, no doubt, composed mostly of water and pyridine; the fraction from 105° to 260° is probably composed of quinoline and its methyl derivatives; the fraction distilling between 260° and 300°, of higher homologs of quinoline; while the portion distilling

Solutions of Tar Acids and Tar Bases Derived from Coal-tar Products

Tar Acids from Reilly No. 19		Whole Tar Acids from Lewis Tar	Tar Acids from Lewis Tar, Fraction F (280°–350° C.)					
Fraction E (215°–225° C.)	Fraction E (215°–225° C.)							
200	200	200	200	200	200	200	200	200
2.9	2.9	5.1	13.8	13.8	13.8	14.3	14.3	14.3
1.6	1.6	2.9	11.2	11.2	11.2	8.1	8.1	8.1
45	45	43	19	19	19	43	43	43
11	11	11	5	5	5	11	11	11
0	0	10	30	30	30	22	22	22
89	89	80	67	67	67	69	69	69
1.3	1.5	1.0	2.4	6.9	4.8	1.0	4.8	3.7
0.3	0.1	1.9	8.8	4.3	6.4	7.1	3.3	4.4
178	178	160	134	134	134	138	138	138
10	3.5	37	64	31	46	50	23	31
17.8×10^{-6}	6.2×10^{-6}	59×10^{-6}	84×10^{-6}	42×10^{-6}	62×10^{-6}	69×10^{-6}	32×10^{-6}	43×10^{-6}
−100 +200-mesh quartz	−100 +200-mesh galena	Minus 200-mesh galena	Minus 200-mesh galena	Minus 200-mesh quartz	Minus 200-mesh galena	Minus 200-mesh quartz	Minus 200-mesh quartz	Minus 200-mesh galena
350	73	267	267	1095	267	1095	1095	267
1.0	2.7	3.0	3.0	1.0	1.0	1.5	0.5	1.5
392	196	800	800	1095	267	1645	550	400
4.6×10^{-6}	3.2×10^{-6}	7.4×10^{-6}	10.5×10^{-6}	3.8×10^{-6}	23.2×10^{-6}	4.2×10^{-6}	5.8×10^{-6}	10.7×10^{-6}
164	431	39	20	26	22	12	19	16

from 300° to 360° C. is composed of still higher homologs of quinoline and of phenanthridine.

The results of the adsorption tests with the tar-acid and tar-base fractions of the coal-tar creosotes are presented in Table 12, which carries the computation as far as determining the average thickness of the adsorbed film.

TABLE 13.—*Thickness of Adsorbed Films of Tar Acids and Tar Bases from Coal-tar Products*

Adsorbent	Size of Adsorbent, Mesh	Oil	Concentration, Parts per Million	Area of Adsorbent per cc. of Oil, Sq. Cm.	Thickness of Adsorbed Film, Centimeters
Quartz......	−200	Whole tar bases, from International creosote.	150	550	0.92 × 10⁻⁷
Quartz......	−200	Whole tar bases, from International creosote.	150	275	1.34 × 10⁻⁷
Quartz......	−100 +200	Tar bases from International creosote, fraction D (300°–360° C.).	150	350	1.11 × 10⁻⁷
Quartz......	−100 +200	Tar bases from International creosote, fraction C (260°–300° C.).	150	350	0.83 × 10⁻⁷
Quartz......	−100 +200	Tar bases from International creosote, fraction C (260°–300° C.).	150	87	1.37 × 10⁻⁷
Quartz......	−200	Tar bases from International creosote, fraction B (100°–260° C.).	150	825	0.70 × 10⁻⁷
Galena......	−200	Whole tar acids from Lewis tar.	200	800	0.74 × 10⁻⁷
Galena......	−200		200	800	1.05 × 10⁻⁷
Quartz......	−200		200	1095	0.38 × 10⁻⁷
Galena......	−200	Tar acids from Lewis tar, fraction F (280°–350° C.)	200	267	2.82 × 10⁻⁷
Quartz......	−200		200	1645	0.42 × 10⁻⁷
Quartz......	−200		200	550	0.58 × 10⁻⁷
Galena......	−200		200	400	1.07 × 10⁻⁷

TABLE 14.—*Length of Molecules of Various Organic Compounds; after Langmuir*

Name of Substance	Formula	Length of Adsorbed Molecules, or Thickness of Adsorbed Film, in cm.
Palmitic acid......................	$C_{15}H_{31}COOH$	2.4 × 10⁻⁷
Stearic acid.......................	$C_{17}H_{35}COOH$	2.5 × 10⁻⁷
Cerotic acid.......................	$C_{25}H_{51}COOH$	3.1 × 10⁻⁷
Tristearin.........................	$(C_{18}H_{35}O_2)_3C_3H_5$	2.5 × 10⁻⁷
Oleic acid.........................	$C_{17}H_{33}COOH$	1.12 × 10⁻⁷
Triolein...........................	$(C_{18}H_{33}O_2)_3C_3H_5$	1.3 × 10⁻⁷
Trielaidin.........................	$(C_{18}H_{33}O_2)_3C_3H_5$	1.36 × 10⁻⁷
Cetyl palmitate....................	$C_{15}H_{31}COOC_{16}H_{33}$	4.1 × 10⁻⁷
Mericyl alcohol....................	$C_{30}H_{61}OH$	4.1 × 10⁻⁷

Thickness of Adsorbed Films on Solids.—By the term, "thickness of adsorbed film" is meant the quotient of the amount of oil adsorbed divided by the area on which it is spread. As will be seen from Table 13, in which the results of Table 12 are condensed, as well as from Table 11, this average thickness is of the order of 10⁻⁷ cm. in the case of the material tested. This result is to be compared with the thickness of complete monomolecular films as given by Langmuir,[11] from whose paper Table 14 is abstracted.

[11] Constitution of Solids and Liquids. *Jnl. Am. Chem. Soc.* (Sept., 1917) 39, 1865.

The theoretical thickness of a complete monomolecular phenol film can be determined from the data given on page 20 as follows:

Let X = thickness; N = number of molecules per gram-molecule; M = molecular weight; p = specific gravity; and A = area occupied by each molecule. Then the volume of the gram-molecule is,

$$\frac{M}{p} = NXA$$

Solving for X,

$$X = \frac{M}{NAp} = \frac{94.1}{6.026 \times 10^{23} \times 27.9 \times 10^{-16} \times 1.07} = 0.52 \times 10^{-7} \text{ cm.}$$

For cresol $X = 0.63 \times 10^{-7}$ cm.; and for acetic acid, 0.39×10^{-7} cm.

All of these thicknesses are of the same order of magnitude and of the same order as those in the last column of Table 13. They indicate that the adsorbed films on mineral surfaces are monomolecular.

Table 13 shows that the adsorption of the same solute by different minerals may be different and that it is generally greater at sulfide surfaces than at quartz surfaces. In the case of the tar acids from Lewis tar, the adsorption on galena is from three to four times larger than it is on quartz. This can only be interpreted as meaning that the adsorbed film on quartz is not complete but partial, whether the adsorbed film on galena is complete or partial.

If the adsorbed film on galena is complete, the adsorption in this case should be irreversible. The following experiment was devised to test the question of irreversibility. After adsorbing a tar acid on to two 500-gm. lots of galena, in the one case below 200 mesh, in the other between 200 and 100, the solid was drained by suction filtration and then placed in a Janney laboratory flotation machine with about 1500 c.c. of water and agitated at 2200 r.p.m. No froth at all was produced. Yet the amount of adsorbed oil present produced an appreciable froth when added anew even without solid material to enhance the viscosity. This indicates that adsorption on galena is not substantially reversible.

The adsorption of tar bases by quartz (Table 13) indicates that a greater film thickness, or, more properly, a more nearly complete monomolecular film, is probably formed than is the case with tar acids. The difference in the thickness of the adsorbed films of tar acids and tar bases at quartz surfaces may be generalized into the statement that adsorption of different solutes at a given surface varies with the solute.

Conclusions.—The conclusions that can be drawn from this study of adsorption at solid surfaces are:

1. Adsorption of dissolved molecules from solution onto solid surfaces occurs.

2. The adsorbed film tends to be monomolecular, and may be more or

less complete, depending on the nature of the adsorbent and of the solute to be adsorbed.

3. Adsorption may be irreversible.

4. Tar acids adsorb to the extent of a complete monomolecular film on galena, but only as a partial monomolecular film on quartz.

ORIENTATION OF MOLECULES IN SURFACE FILMS

Langmuir and others have shown that molecules are not equidimensional, but usually elongated, although they may sometimes be short and stocky. For lack of exact knowledge concerning the molecules herein considered, they have been assumed, in the calculations of this paper, to be cubical.

Molecules are not homogeneous throughout their body, inasmuch as different atoms or groups of atoms occupy different positions within the molecule. These groups of atoms or radicals often have fields of force extending beyond the limits of the molecule,[12] so that their behavior toward neighboring molecules of similar or different constitution differs according to the atoms or radicals concerned. It is, then, logically conceivable that, if different parts of a certain molecule exert forces of a different magnitude on another molecule, that molecule will orient itself in a definite way with regard to its neighbor.

In regard to the orientation of molecules of a substance in contact with water molecules it is necessary to discriminate between polar radicals[13] (i.e., those that are attracted by water) and non-polar radicals, which are not so attracted. Examples of polar radicals are $-OH$, $-COOH$, $=CO$, $=CN$, $-CONH_2$. The radical $-CH_3$ is typically non-polar. When a molecule is in solution in water, it, being surrounded by water, will not tend to orient itself in any given direction because the pull of the water molecules on it is the same in all directions. If, on the other hand, a molecule chances to come to the surface, the polar end will be drawn to the water while the non-polar end will tend to stick out. In their articles[14] Messrs. Harkins, Davies, Brown, Clark and King arrive at the conclusion that at any interface of two phases the molecules are always oriented so as to make interfacial potential energy a minimum, and this includes the case of a water surface above considered. Similar conclusions are arrived at by Langmuir in his previously quoted paper. If molecules are oriented at liquid-liquid and gas-liquid interfaces, it is reasonably probable that molecules adsorbed from solution onto solid surfaces are likewise oriented.

[12] W. D. Harkins and H. H. King: Jnl. Am. Chem. Soc. (1919) 41, 970–992.

[13] W. D. Harkins, F. E. Brown, E. C. H. Davies: Jnl. Am. Chem. Soc. (1917) 39, 354, 541.

[14] W. D. Harkins, F. E. Brown, E. C. H. Davies: Ibid.

W. D. Harkins, E. C. H. Davies, G. L. Clark: Ibid., p. 541–591.

W. D. Harkins and H. H. King: Jnl. Am. Chem. Soc. (1919) 41, 970–992.

At a water-air interface, molecules of a monomolecular film of oleic acid are oriented so that the polar carboxyl radical is toward the water and the hydrocarbon chain toward the air. Similarly at an oleic acid-water interface, as when a droplet of oleic acid is submerged in water, the molecules of oleic acid are oriented so as to have the −COOH radical toward the water and the hydrocarbon chain toward the body of the oleic acid. If, then, oleic acid is emulsified in water, each globule of the acid will have a surface layer composed of molecules oriented with the carboxyl radical toward the water, and gas should not, therefore, precipitate from solution in the water on to such oil globules. This agrees with experimental results. Adding mineral particles to such an emulsion will bring solid particles in contact with the oil globules and these globules, if small enough, may be insufficient to coat the surface of the mineral with a film more than one molecule thick. Such an oil globule approaches the solid with the carboxyl end of its molecules toward the solid. One should, therefore, expect this end of the molecules to be the one to come in contact with the mineral, unless the molecules turn around at the instant of contact; that is, after coating, the molecules should present the paraffin end to the water. If this is the case, such a surface should be especially favorable for gas precipitation. Such an expectation is confirmed by experiment. A more convincing proof is furnished by the following test, which was carried out for one of the writers by Doctor Langmuir: First a perfectly clean burnt platinum plate was brought down through a monomolecular film of oleic acid on water into water supersaturated with gas; no gas precipitated on the platinum. Next, the cleaned plate was placed in water, a monomolecular film of oleic acid was produced on the water surface, and the plate was drawn out through the surface; the plate was allowed to drain to apparent dryness, and was then re-immersed in clean supersaturated water, when gas precipitation occurred. This test indicates clearly that in the first case the passage of the platinum through the oleic-acid film did not produce contamination of the metal surface, while in the second case it did. In the first case the paraffin end of the acid molecules was presented to the plate; in the second case the carboxyl end. The conclusion would seem to be that, in the case of this particular metallic surface at least, there was distinct orientation of the molecules forming the film coating, and that the orientation resulted in the formation of a paraffin surface.

Air bubbles in an oleic acid-water emulsion become coated by oleic-acid molecules having the carboxyl radical toward the water and the paraffin radical toward the air.

Reasoning by analogy it would seem that a galena particle in a dilute solution of phenol should appear as shown in Fig. 28; that is, with the polar end of the adsorbed phenol molecules directed toward the galena. As such a particle approaches an air surface, the non-polar ends of the

phenol molecules adsorbed on the galena are oriented toward the air, just as the non-polar ends of phenol molecules adsorbed at an air-solution interface are oriented toward the air. When the galena particle reaches the air surface, it will tend to become interfacial because of its adsorbed phenol molecules, such movement resulting in a reduction of the total surface energy of the system (see Fig. 29).

There is no reason to expect the adsorbed molecules at the surface of a quartz particle to be oriented differently from those at the surface of a galena particle. In fact, evidence exists that quartz floats, although much less readily than galena, indicating that there is a difference of degree only between these minerals, insofar as surface energy is concerned. Accepting the orientation of the adsorbed molecules on quartz to be the same as on galena, the difference in completeness of the film on
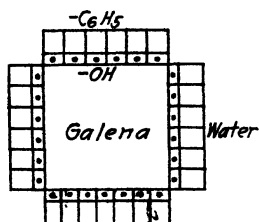
FIG. 28.—ADSORBED MOLECULES OF PHENOL ON GALENA.
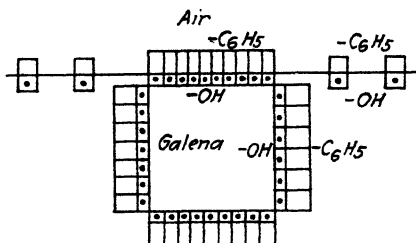
FIG. 29.—COATED GALENA PARTICLE AND PHENOL MOLECULES AT AIR-WATER SURFACE.

galena and quartz must cause a difference in the adherence to bubble films. Quartz showing an adsorbed film less complete than that on galena should and does show less tendency to adhere to air bubbles. Thus, because of adsorption of oil at the mineral surfaces, adherence to air surfaces already existent takes place, and tendency to develop new gas surfaces appears; that is, adsorption of flotation oils at mineral surfaces enhances gas precipitation. These properties are exhibited in different degrees by sulfides and gangue minerals as, for instance, galena and quartz in the presence of phenol.

A phenomenon illustrative of this fact is observed in the bubble column of a pneumatic flotation cell where gangue particles can be seen to drain readily between the air bubbles, while the sulfide particles drain much more slowly and show a decided tendency to linger at the air-liquid interfaces. This differential draining of the particles is what causes concentration of the more floatable (sulfide) minerals at the top of the froth.[15] In machines of the pulp-body type,[16] the effect of adsorp-

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[15] A. F. Taggart: "Manual of Flotation Processes." John Wiley and Sons, Inc 1921, 7–9.

[16] Ibid., 3–7.

tion is manifested by selective gas precipitation on the minerals having more complete adsorbed films, and results in forming aggregates or clusters of these minerals around small gas bubbles.

RELATION OF SURFACE TENSION TO FLOTATION

If great care is taken in placing a fragment of mineral heavier than water on an air-water surface, it will float, if small enough. By placing dry, unoiled mineral upon a glass dome (1-in. diameter) surrounded by water, the top of the dome being about $\frac{1}{4}$ in. above the water surface, the mineral can be caused to slide gently over the surface of the dome onto the surface of the water, and will either float or sink. Table 15 gives the results of such an experiment with galena; from this table, it appears that there is a critical size in the neighborhood of 1.17 mm.

TABLE 15.—*Size of Galena Grains in Skin Flotation*

Grain Size, Millimeters		Per Cent. Floated
Passes Screen, Millimeter Aperture	Stays on Screen, Millimeter Aperture	
2.36	1.65	0
1.65	1.17	1
1.17	0.83	85–90
0.83	0.59	100

The results in Table 15 are obtainable only when great care is taken in bringing the particles to the water surface. If any material agitation occurs, it is not possible to float any solid on pure water. When, on the other hand, a reagent is added to the water and is dissolved in it, it adsorbs either positively (in the surface), as cresol, or negatively (away from the surface), as common salt, and creates a difference between the surface tension of the liquid and the surface tension that the liquid would have if it could present a surface with the concentration present in the bulk. The latter surface tension is always the higher. This is the condition that promotes frothing and is also what causes froth flotation. If a mineral particle is introduced at the air-liquid interface of a solution, when the particle, due to the pull of gravity, extends the surface film, it creates a certain area of new surface having a tension nearer the hypothetical bulk surface tension than is the original, and always higher. This counteracts the pull of the mineral particle so that the film supports it. This equilibrium condition is stable as contrasted with the unstable equilibrium of the mineral in the case of flotation with pure liquid. There seems to be a definite relation between the maximum size of mineral that can be floated and the difference in surface tension between the bulk surface tension and the actual surface tension. This difference

is, in dilute solutions, practically equal to the depression in surface tension Δ below that of pure water.

The following series of flotation tests on sized and unsized porphyry copper ore was made in a Janney laboratory machine with tar-acid fractions from coal-tar oils as the sole reagents, in an attempt to quantify Δ. The amounts of oil varied from 0.20 to 0.40 lb. per ton of ore in the case of the fractions from Barrett No. 4, and from 0.19 to 0.48 lb. per ton for the fractions from Lewis tar. Similar tests were carried on with cresol, U.S.P., for comparison.

The results for the unsized feed are condensed in Table 16. The data show that:

TABLE 16.—*Results of Treatment of Unsized Porphyry Copper Ore with Tar-acid Fractions of Coal-tar Products*

Reagent	Pounds per Ton				Pounds per Ton			
	0.19	0.29	0.37	0.48	0.19	0.29	0.37	0.48
	Recovery, Per Cent.				Grade of Concentrate, Per Cent. Cu			
Tar acids from Lewis tar:								
Fraction B (115°–205° C.)	38.3	41.7		55.9	14.12	7.98		6.04
Fraction C (205°–215° C.)		54.0				7.72		
Fraction D (215°–225° C.)	47.6	66.4			10.10	6.30		
Fraction E (225°–280° C.)	59.1	78.4		83.6	8.89	8.58		4.72
Fraction F (tarry) (280°–350° C.)		41.9	(Heated) 72.1			9.04	(Heated) 18.18	
Tar acids from Barrett No. 4:								
Fraction A (170°–200° C.)	31.1		64.9		18.00		7.80	
Fraction B (200°–205° C.)	38.8		74.0		16.37		7.60	
Fraction C (205°–215° C.)	52.4		75.0		7.14		6.02	
Fraction D (215°–230° C.)	56.3		80.8		7.81		6.58	

Ore: Disseminated, primary copper sulfide; all passes 48 mesh, 72.2 per cent passes 200 mesh. Assay, 1.8 per cent. copper.

1. The higher the boiling point of the tar acid, and therefore the higher its place in the homologous series and the less its solubility, the greater the recovery, and, in general, the lower the grade of concentrate.

2. The greater the proportion of oil to ore (within the limits chosen), the greater the recovery and the lower the grade of concentrate.

3. The higher the recovery, the less the change in recovery for a given change in amount or quality of the oil.

The results for the sized feed are condensed in Table 17; from these data the following further conclusions are justified:

4. The finer the ore, the greater the recovery for a given proportion of oil to ore and a given quality of oil.

5. By increasing the proportion of oil to ore, with material too coarse

TABLE 17.—*Results of Treatment of Sized Porphyry Copper Ore with Tar-acid Fractions of Coal-tar Creosote*[8]

Reagent, Name	Lb. per Ton of Ore	Size of Feed, Mesh			Size of Feed, Mesh		
		−100, +150	−150, +200	−200	−100, +150	−150, +200	−200
		Grade of Concentrate			Recovery		
Tar acids from Lewis tar:							
Fraction B (115°–205° C.) .... {	0.29	10.35	21.37	6.74	1.7	10.4	84.4
	1.16ᵃ	4.03	10.67		14.2	35.0	
Fraction C (205°–215° C.)	0.29			6.17			88.6
Fraction D (215°–225° C.)	0.29			5.97			90.2
Fraction E (225°–280° C.) .... {	0.29	12.90	27.33	6.48	10.2	47.9	89.0
	0.87ᵃ	20.00	22.84		69.0	86.6	
Tar acids from Barrett No. 4:							
Fraction A (170°–200° C.)	0.37			6.53			91.4
Fraction B (200°–205° C.)	0.37			6.07			95.0
Fraction C (205°–215° C.)	0.37			5.82			94.1
Fraction D (215°–230° C.).. .... {	0.37		8.21	4.64		15.5	93.4
	1.14ᵃ		15.32			55.2	
Cresol, U. S. P.	0.37			6.33			92.4

ᵃ A sufficient further amount of the same reagent was added to make this total and a second 10⁻ min. run made. Grade of concentrate and recovery set against this amount are total for the two 10-min. runs. Feed, a disseminated, primary copper-sulfide ore, assaying 1.8 per cent. copper.

to give a good recovery with the smaller amount of oil, a much better recovery can be obtained.

6. The less soluble members of the tar-acid series, which are, for a given weight, the more powerful frothers, are most suitable for the recovery of coarse mineral and improve the recovery of fine mineral to a small, but distinct, extent.

The higher the boiling point of a tar acid, the less soluble it is and, therefore, the greater its effect on surface tension for a given amount of oil (see Table 1 for the products of Barrett No. 4); *i.e.*, the larger the surface-tension depression Δ. Bearing this fact in mind, conclusion 1 may be restated in terms of Δ as follows:

1. The greater Δ, the greater the recovery and the lower the grade of concentrate.

Increasing the proportion of oil means a proportional increase in Δ; hence conclusion 2 above becomes like conclusion 1.

2. The greater Δ, the higher the recovery and the lower the grade of concentrate.

The third conclusion may be stated as follows:

3. The more nearly perfect the recovery, the smaller the change for a given change in Δ.

The fourth conclusion becomes:

4. For a given Δ, the finer the ore the greater the recovery.

The fifth becomes:

5. Sufficient increase in Δ with a given ore will render floatable material previously too coarse to float.

Finally, conclusion 6 becomes:

6. Δ must be large for the recovery of coarse material and an increase in Δ improves the recovery of fine material. Recovery is, therefore, dependent on the magnitude of Δ and the size of the ore. Grade of concentrate varies in reverse fashion to the recovery, indicating that as Δ increases or size of ore dminishes, the proportion of gangue floated increases. Increase in Δ makes floatable the previously non-floatable gangue.

In order to establish a relation between size of ore and Δ, it is first necessary to know how much oil is removed from solution by adsorption on the mineral surfaces, and how much is lost by dilution and evaporation. For the purpose of the following calculation on this head it is assumed, based on data already presented, that oil is adsorbed on sulfides to the extent of a full film 10^{-7} cm. deep and on the gangue to one-third the same extent, and that the ore is composed of 6 per cent. of sulfides and 94 per cent. of gangue by weight. In 500 gm. of ore there are, then, 30 gm. of sulfide (chalcocite) and 470 gm. of gangue (quartz). With material finer than 200 mesh (0.074 mm.) there will be removed by adsorption (using Table 7): $(370 \times 30 \times 10^{-7}) + (1095 \times 470 \times \frac{1}{3} \times 10^{-7}) =$ 0.018 gm. of solute. The amount of oil adsorbed by sized ore of the same composition as the ore of the above illustrations, and the amounts of oil available to affect surface tension when two and four drops of tar acids are added, are presented in Table 18. The amount of water added to the Janney laboratory machine at the beginning of the flotation tests is about 1800 c.c. The concentration of the oil in the water after adsorption has taken place is, therefore, about 43 to 50 parts per million when adding four drops of oil. This presupposes that all of the oil has dissolved and that the amount which evaporates can be neglected. As the test advances, oil is removed selectively in the froth and the pulp in the cell is further diluted. The concentration of the oil thus decreases rapidly from the original 43 to 50 parts per million to, perhaps, less than 5 parts per million when the test ends.

The data relative to Δ have been obtained for concentrations greater than 44 parts per million, because with solutions as dilute as this, Δ is very small, and its determination difficult. The results in Table 19 are calculated from the experimental results by interpolation.

From the data in Tables 17, 18 and 19, the relation between size of ore particle and Δ seems to be of the type $a = K\Delta^m$, where $m < 1$, probably about $\frac{1}{2}$, a is the diameter of particle and K and m are constants for a given set of conditions. K and m will vary with the kind of solute, the kind of mineral, the viscosity of surface films of the liquid, the bunching of

TABLE 18.—*Amounts of Oil Adsorbed and Remaining in Solution in Flotation Tests on a Copper Ore*

Size of Ore		Amount of Oil Adsorbed by 500 Gm. of Adsorbent, Grams	Amount of Oil Available after Adsorption	
Passes	Stays on		Four Drops Added (0.096 Gm.)	Two Drops Added (0.048 Gm.)
0.074 mm. (200 mesh)		0.018	0.078	0.030
0.104 mm. (150 mesh)	0.074 mm. (200 mesh)	0.007	0.089	0.041
0.147 mm. (100 mesh)	0.104 mm. (150 mesh)	0.005	0.091	0.043
Unsized ore.....................		0.016	0.080	0.032

TABLE 19.—*Surface Tension Depression Δ in Laboratory Agitation-froth Tests on Porphyry Copper Ore*

Reagent	Concentration, Parts per Million	Δ: Corresponding Depression in Surface Tension	Computed Δ for Concentration of 44 Parts per Million, Dynes per Cm.
Cresol, U. S. P.......................	1250	7.0	0.24
Tar acids from Barrett No. 4:			
Fraction A (170°–200° C.)...........	100	0.4	0.17
Fraction B (200°–205° C.)..........	100	0.6	0.26
Fraction C (205°–215° C.)...........	100	1.9	0.84
Fraction D (215°–230° C.)..........	100	1.8	0.78
Tar acids from International creosote:			
Fraction C (205°–215° C.)..........	150	1.7	0.58
Fraction D (215°–225° C.)..........	150	1.4	0.48
Fraction E (225°–260° C.)..........	150	4.3	1.46

particles so as to form flocks, and the thickness of the bubble films. They probably change little, however, for solutes forming an homologous series, such as the tar acids on which the tests described above were carried out. Presence of very fine material together with coarser particles may change the viscosity of the surface film, so as to float otherwise non-floatable coarse particles.

SUMMARY

1. A form of apparatus for.the measurement of surface tension at gas-liquid interfaces by the Wilhelmy method is described.

·2. A number of surface-tension tests are presented which lead to the following conclusions:

(a) The members of an homologous series depress the surface tension of water in amounts that increase as the molecular weight increases and the solubility decreases.

(b) The concentration of solute and surface tension of the solution bear a linear relation in dilute solutions; they vary according to a logarithmic law in concentrated solutions; in solutions of intermediate concentration, the law of variation is variable and intermediate between the other two.

(c) The depression of surface tension due to the simultaneous addition of two or more solutes that are members of an homologous series is, in the main, the sum of the depressions caused by the individuals when present separately. This additive rule does not, necessarily, apply when the added substances are not chemical homologs.

(d) The addition to water of a substance (oil) insoluble therein has no effect on the surface tension of the water.

(e) The simultaneous addition to water of a substance (oil) insoluble therein and of another substance (oil) soluble in the water but not soluble in the first substance, produces only the effect on the surface tension of the water that would have been produced by the addition of the second substance alone.

(f) The simultaneous addition to water of a substance (oil) insoluble therein and of another substance (oil) soluble both in the water and in the first substance, produces an effect on the surface tension of the water less than that which would have been produced by the addition of the second substance alone. Surface tension-concentration curves for this type of mixtures change with concentration from convex to concave, indicating progressive increase in extraction of solute by the water.

(g) Oleic acid has but slight effect on the surface tension of water until the amount present is sufficient to and does form a monomolecular film. Beyond this point, surface tension drops rapidly until an amount of acid equivalent to a film about two-and-a-half molecules deep is reached, when another change occurs and further addition produces again but slight effect.

(h) Oleic acid is a limiting case in the series of soluble substances affecting surface tension.

3. The relation between the solubility of an oil with its consequent effect on the surface tension of water and its effect on frothing is found to be as follows:

(a) The addition to water of an oil insoluble therein has no effect on the frothing power of the water.

(b) The effect of the addition of members of an homologous series on the frothing power of the resultant water solution increases as the solubility of the member decreases.

(c) Maximum frothing effect occurs with the addition of amounts of

solute about one-half of that required to saturate the solution. The point of maximum frothing is that at which the difference between the static and dynamic surface tensions of the solution is greatest.

(d) Emulsions in water, in which no water solute is present, do not froth.

(e) Emulsions of oil in water, consisting in part of mechanically dispersed oil and in part of dissolved oil, froth less than does a solution containing an equal amount of solute alone.

(f) The undissolved oil of an emulsion acts as a reservoir for soluble oil so that the frothing power of the emulsion is not diminished as greatly by dilution with water as would be expected were the solute present alone.

4. Methods of making adsorption tests are described and results of adsorption tests are presented that lead to the following conclusions:

(a) At saturation, the concentration of a positively adsorbed substance at the gas-liquid interface of a solution is that of a complete monomolecular film.

(b) The extent of adsorption of solute increases with the concentration at a decreasing rate; this extent is greater the less the solubility of the solute.

(c) An oil insoluble in water, when present alone therein, will not adsorb at a water-air interface.

(d) Distinct concentration of soluble oil takes place in the froth in a flotation machine, with corresponding impoverishment of the water in the machine.

(e) Adsorption of flotation oils at solid surfaces from emulsions is not capable of measurement by the method employed.

(f) Soluble oils adsorb at solid surfaces in amounts of the order of monomolecular films. The adsorption is, in general, greater by two to three times at sulfide surfaces than at gangue surfaces.

(g) Adsorption of flotation oils from water solution at sulfide surfaces is an irreversible phenomenon.

(h) The amount of adsorption of different solutes at a given surface varies with the solute.

(i) Molecules in adsorbed films are definitely oriented with respect to the adsorbing surface. In the case of flotation oils the orientation is probably generally such that the hydrocarbon end of the molecule is away from the solid at solid-water interfaces, and away from the water at air-water and bulk oil-water interfaces.

(j) Selection of sulfide from gangue in flotation takes place by reason of the differential adsorption of flotation oils at sulfide and gangue surfaces. The difference in behavior of the coated and uncoated solid particles toward air takes place by reason of the orientation of the molecules in the adsorbed film.

(k) The difference in adsorbing power of sulfide and gangue minerals

is one of degree only and varies with different solutes; hence by proper regulation of conditions, variations in difference in flotability between the two can be controlled.

5. A relation between surface-tension depression and size of mineral grain that can be floated is established in a particular case and a general relation of the type $a = K\Delta^m$ is indicated, where a represents the maximum size of grain in the ore, K and m are constants for any given ore and reagent, and Δ is the surface-tension depression. The number m has a value near $\frac{1}{2}$. K and, to a lesser extent, m depend on such factors as kind of mineral, specific gravity of solid, kind of solute, thickness of bubble walls, size of bubbles, and viscosity of bubble films.

6. Relations between depression of surface tension and metallurgical results of a flotation operation are found to exist as follows:

(a) The larger the surface-tension depression at the beginning of a flotation operation, the greater the recovery and the poorer the grade of concentrate.

(b) The higher the recovery, the greater the possible change in surface-tension depression without resulting change in recovery.

(c) The finer the ore, the less the necessary depression in surface tension in order to effect a given recovery.

DISCUSSION

R. C. CANBY, Wallingford, Conn. (written discussion).—This paper, from a casual turning over of its pages, might seem to be so purely mathematical or so highly technical that it will be put aside for further study. The reader would thus fail to glean from its summary some important facts that from the title one might not expect the paper would contain.

While the authors do not so state, the work upon which the paper is based was largely done in connection with the flotation litigation, then in progress, in an effort to ascertain the underlying principles of flotation.

Many of the facts elicited are of broadly practical application suggesting valuable economic results which were not uppermost in mind in making the apparently purely theoretical bases of the experiments. Conclusion 5, for example, suggests that by a more accurate understanding of the surface-tension depression values of available frothing agents, and making the most suitable selection therefrom, one might attain improved results which would otherwise have appeared possible only through a remodeling of the crushing plant.

Personally, I cannot but regret that the facts of this paper, together with many other facts so well demonstrated by Professor Taggart and his assistants, which because of the settlement of the flotation litigation, for which this work was done, have become largely academic, could not have found the use for which they were intended. For example, conclu-

sion 2 (d) in the summary: "The addition to water of a substance (oil) insoluble therein has no effect on the surface tension of the water," challenges the validity of the so-called "soluble frothing agent" patent.

A. W. FAHRENWALD, Moscow, Ida. (written discussion*).—As possibly the pioneer in the application of surface-tension and adsorption measurements to flotation, I was greatly interested in this paper and am glad to find in it much that confirms my own earlier work.[17]

Quite unintentionally, I am sure, the authors, while crediting me with the development of the surface-tension apparatus that they used and have called it the "Fahrenwald apparatus," speak of it as a modification of Wilhelmy's method. Wilhelmy's method consisted in weighing a frame of given dimensions when suspended, from a balance, a given distance in the liquid to be tested; so mine is no more a modification of it than of the sphere-segment method used by Ferguson, neither of which measure the tension in a film and the balance used was either an analytical or a Jolly. Wilhelmy's method, like all others of similar principle, has serious objections to its use.

The authors modified my machine by using a smaller pulley wheel and claim that with the large wheel there is too much jarring of the apparatus when the film is being drawn out, because of the relatively larger vertical movement of the crystallization dish in the case of the large wheel; this objection is of importance only in the case of a flimsy poorly designed instrument. The writer has long used an instrument in which the stand supporting the crystallization dish and the mechanism for raising and lowering it is entirely separate from the balance. They claim also that greater sensitivity is gained by the use of a small counter-weighted pulley. In my first work a counterweight was tried, but proved unsatisfactory because the momentum of the rather heavy counterweight and swinging member gave it a tendency to swing through the film, or to rupture it, also because of the slowness with which the needle pointer came to rest and a lack of stability in general. They also claim that more accurate results can be obtained by weighing the pull of the film each time than by reading from a graduated dial, which they do by marking the stopping point of the needle on the dial and, after drying the knife edge, bringing it back to this point by proper weights. Such a procedure seems a bit awkward and time consuming, as well as open to error. The authors do not state the width of edge used to engage the liquid and to which the film 'is attached. If the knife edge is of appreciable thickness, results would be higher than the theoretical value by an amount equal to the weight of liquid driven from the plate when drying it and the error would be in relation to the width of the plate used.

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* Published by permission of the Director of the U. S. Bureau of Mines.
[17] Min. Sci. Press (Aug. 13, 1921) 123, 227–234.

The small pulley is a distinct disadvantage as slight irregularities in the design of the turning member reflect greater differences in the displacement of the swinging pointer than they would with a larger pulley.

In addition, the method of correcting for the capillary pull of the liquid on the horns of the knife edge seems open to possible error.   The method presupposes the existence of a zero contact angle of the liquid against the metal plate for all measurements and at all times; this will hardly be the case.   The contact angle, if it varies from zero, would affect the 5.5 per cent. of the total pull of the knife edge, which is the figure given by the authors for the "prong pull."   For pure water against a perfectly clean metal plate, the contact angle is no doubt zero; but on oily films it is hard to say what this angle would be.

The authors mention 0.2 dyne per cm. as the limit of accuracy for the modified apparatus used by them; I have narrowed this limit to 0.1 dyne per cm.

Under the heading Cleaning the Apparatus the authors say: "Early work showed that the surface tension of every preparation dwindled appreciably, often several dynes, with lapse of time" and that this dwindling was eliminated by cleaning the knife edge, after every dip in any solution or emulsion, by immersing it first in acetone and then in ether. This dwindling habit of emulsions and other preparations, with time, was one of the greatest disturbing factors in our early work.   We were unable to cope with it by simply cleaning the knife edge between measurements, although the method of cleaning the plate used by the authors was not tried.

The reason for the dwindling is as follows: All substances that lower the surface tension of water tend to concentrate in its surface; i.e., there is adsorption of the solute in the water surface.   The surface-concentration process is not an instantaneous one; for as the solute concentrates in the surface, the liquid layers immediately contiguous to the surface become less concentrated in the surface-tension lowering constituent and thus retard the rate at which the solute goes into the surface.   The surface tension continues to decrease slowly until the solute is uniformly distributed through the bulk of the solution below the surface film and until the number of molecules entering the surface film in a unit of time is equal to the number of molecules passing from the surface film into the bulk of the liquid.   It can be shown[18] that the dwindling in surface tension of solutions of organic substances and emulsions comes to a constant surface-tension value if sufficient time is allowed.   We found that the rate of adsorption of the constituents lowering surface tension could be greatly expedited by causing a gentle agitation of the solution or emulsion without disturbing its surface.   This was done by drawing the

[18] "Surface Tension in Flotation," *Min. & Sci. Press* (Nov. 5, 1921), 631.

emulsion into a 50 or 100-cc. pipette a number of times and allowing it
to discharge slowly beneath the surface; various preparations required
from two to two dozen treatments before equilibrium was reached. This
procedure simply served to renew the depleted layers under the sur-
face film.

In our work, much of which remains to be published, we found that
with the film method of measuring surface tension of liquids, the surface
tension of which decreased with age, the only reliable procedure was to
measure the tension in surfaces in which the adsorbed film had come to
equilibrium with the bulk of the liquid. As surface concentration or
adsorption processes in general, whether it be at an air-liquid or a liquid-
liquid interface, are not rapid, a surface tension method to be reliable
must measure the tension in (1) surfaces that have come to equilibrium
with the bulk of the solution, (2) surfaces of an equal age, or (3) surfaces
so fresh that no surface concentration has taken place. We found that,
if we ignored the time factor in working with emulsions, the drop
weight or the Jaeger capillary bubble method, which measures the tension
in relatively fresh surfaces and surfaces of uniform age, was more reliable
than the film method. Yet, I judge that the authors had no trouble, when
using the method of cleaning the plate between measurements, in obtain-
ing concordant values, on all ages of surfaces of a given liquid.

In the method of making adsorption tests of surface-tension lowering
substances described, it seems necessary, in the cases of such substances
as oleic acid or a pine oil, to make the following assumptions: (1) That
the oil is present in the water in two forms, *viz.* as molecules and as drop-
lets; (2) that the droplets, which may range in size from, say, groups of
two molecules to groups of as many as a thousand, have no effect what-
ever on surface tension; (3) that these droplets or molecule aggregates
are not adsorbed by minerals; (4) that the filter bed of infusorial earth
will retain as much soluble oil from a solution that has a surface tension
of say 65 dynes per cm. as it will from a sample of the same oil having a
surface tension of say 60 dynes, the difference being due to a previous
treatment with a large amount of mineral surface.

The first assumption is the only one that is justifiable. We are cer-
tainly not justified in assuming that molecule aggregates in water, of
say oleic acid, have no effect on the surface tension of water. It seems
that, in the case of a substance composed of a polar and a non-polar
group when the molecules group together to form a tiny droplet sur-
rounded by water, the surface of the droplet in contact with water
will be composed entirely of polar groups that tend to dissolve in the
water, and that the non-polar groups will point to the center of the
droplet. Such droplets, in all probability, will not behave as droplets
of ·a substance like benzene in water, which has no soluble portion in the

molecule. That such active droplets are not adsorbed at solid surfaces remains to be shown.

In Table 10, the authors compare some adsorption figures of their own for a pine oil with some figures previously given by the writer for the same oil. As they point out, there is no agreement; their figures show negative adsorption while the writer's figures indicate positive adsorption. We have never found a case of negative adsorption and it seems that such cases must be attributed to experimental error somewhere in the procedure of determining adsorption; one of the most likely sources of this error is probably in the neglect of the time factor in the value of surface tension.

Apparently the authors estimate the amount of solute adsorbed on a given amount of mineral surface on the assumption (my assumption No. 4 above) that a given bed of infusorial earth adsorbs as much of a given solute from, for example, a solution containing 10 parts per million as it does from a solution containing 20 parts per million. The authors give the reduction in concentration by adsorption (presumably referring to adsorption by the numeral only) as $C_B - C_P$, where C_B is the concentration of the blank solution after it has passed through the bed of infusorial earth and C_P is the concentration of the solution after it has been agitated with the mineral and then filtered through the bed of infusorial earth. If C_T is the concentration of the solute in the original solution, C_B the concentration of the solution after passing through the filter bed, x the amount of adsorption from mineral-treated solution by the infusorial earth, and C_P the concentration of the solution after treatment with mineral and after passing through the filter bed, the reduction in the concentration of the original solution by treatment with the mineral would be $C_T - (C_P + x)$. If $C_x = (C_P + x)$, the reduction in concentration would be expressed as $C_T - C_x$ instead of $C_B - C_P$ as given in the paper.

If the above reasoning is not faulty, and our experience has shown that the amount of a solute adsorbed by minerals (and this would no doubt be true for any other adsorbent) bears a relation to the concentration of the solute, a very grave error would enter into the method of calculating adsorption used by the authors. As a given bed of adsorbent (infusorial earth in this case) will adsorb more of the solute from the solution of higher concentration than it will from solutions of lower concentration, the values given for the amounts of solute adsorbed by a unit of mineral surface will, in general, be too low.

The authors show that the thickness of the adsorbed films on minerals are, in general, monomolecular, which is in line with Langmuir's theory that adsorbed films are, in general, one molecule deep. In the case of pure solutions, this is what one would probably expect to find; yet data that we have on such substances as heptylic acid and caprylic

acid as well as oils place the thickness of the adsorbed film at from 10 to 100 or more molecules deep. It is believed that the error in calculating the area of the mineral surface actually available for adsorption is much greater than that estimated.

A word more about the procedure of measuring the surface tension of various preparations, as used by Taggart and Gaudin, to overcome the dwindling in surface-tension effect, namely, by cleaning the plate with acetone and then ether between each dip in any liquid.

It is difficult to understand how, when measuring the surface tension of various preparations, cleaning the plate between each dip with acetone and then with ether will completely overcome the dwindling in surface-tension effect. Cleaning between dips gives a higher surface-tension value for a solution, that shows a decrease in tension with time, than would be obtained after the third or fourth dip without cleaning; however, the value that will be obtained even by this procedure will depend on the time the plate has stood dipped in the liquid before drawing out the film and on the depth to which the knife edge or plate was immersed in the liquid before being drawn out. The greater the length of time the plate stands immersed in the liquid the lower will be the surface-tension value; and the greater the depth of immersion, the higher will be the surface tension.

The British Columbia Batholith and Related Ore Deposits

BY PHILIP D. WILSON,* E. M., WARREN, ARIZ.

(San Francisco Meeting, September, 1922)

THE Province of British Columbia covers 382,000 sq. mi., about 250,000 sq. mi. of which have not been prospected. In fact, the coast country and the islands are so heavily timbered and the surface covered with so thick a mantle of soil, muskeg, and, over broad areas, glacial drift, that prospecting and even trail breaking are extremely difficult. It is only along the coast or stream channels, or above the timber line, or where snow or rock slides have scoured the surface, that rock outcrops can be found. Glaciers nestle in many of the cirques and fill the valleys in the higher ranges. Great ice sheets and perpetual snow persist in certain localities, thus reducing the exposed areas worth prospecting. Almost the entire country has been subjected to intense glaciation, so that the conspicuous oxidized outcrops of orebodies typical of the southwestern United States and Mexico do not exist and surface bodies of enriched or oxidized ore are rare.

With few exceptions, the mines[1] of the Province are working primary ore. Developments have proved that some of these orebodies are large and important, although usually comparatively low in grade. Besides the few mines that have produced most of the minerals, there are many small mines, but their total output is unimportant. Considering the number of mines and prospects and the widespread mineralization, it is strange that no really great mines, comparable in size and richness with those in the western United States, have been found.

The deposits that most closely approach the status of great mines are

probably the Hidden Creek at Anyox, the Britannia on Howe Sound, the Copper Mountain at Princeton, and the Sullivan at Kimberley. The first two have been producing on a moderate scale for some years and of late have furnished most of the copper produced in the Province. The Copper Mountain mine had just entered the ranks of the big producers in the fall of 1920, when it was shut down. These three mines are similar in that the ore, which is low grade, primary sulfide and usually amenable to concentration, occurs in isolated, irregular, lenticular bodies in steeply dipping fracture zones. Some of the bodies outcrop and some of the ore can be mined cheaply by glory-hole methods. Most of it will eventually have to be handled underground by shrinkage or untimbered open stopes. The uncertain distribution of the orebodies, the consequent high cost of mining such low-grade ore, and the comparatively small developed tonnages militate against large profits or a large return on the necessarily big initial investment. The Sullivan mine has developed a large replacement deposit of silver-bearing zinc-lead-iron sulfide ore. It furnishes most of the zinc and a large part of the lead production of British Columbia and, now that the metallurgical problems incident to the treatment of the ore have been successfully solved, it will probably take a foremost place among the big mines of the Province.

In point of value copper has taken, in recent years, first place in the metal production of British Columbia; but during 1919 and 1920 it yielded first place in quantity of production to zinc. The development of the Sullivan mine has greatly increased the production of both zinc and lead, while the output of gold in the Province is gradually declining. Silver has remained about stationary. Table 1 shows the production in the Province for the years 1918, 1919, and 1920 with the total value of the products to date.

The economically most important structural feature of British Columbia is the great Coast Range, which follows the Pacific Ocean a little west of north from the international boundary to the Yukon. This is part of the North American Cordillera, which is over 400 mi. (645 km.) wide at the border. The Cordillera in British Columbia is characterized by parallel north-south ranges, separated by long, narrow, lake valleys and river channels, which form an almost continuous belt of rough, mountainous country parallel to the international line and a direct continuation of the great ranges of Washington and Idaho. The Coast Range is 80 or 90 mi. wide in southern British Columbia, which width gradually decreases to 50 or 60 mi. (80 to 96 km.) at the Yukon boundary and finally merges into the Interior Plateau near the Yukon-Alaska line. It is almost wholly composed of coarse, granitoid rocks, which vary from gabbro to granite in composition, but are in large part granodiorite. These rocks are so closely related that the mass may be considered as one gigantic batholith, intruded during a long-continued period of igneous activity in late Juras-

TABLE 1.—*Mineral Productions of British Columbia²*

Material	1918		1919		1920		Total to Date Value
	Quantity	Value	Quantity	Value	Quantity	Value	
Gold, placer, ounces................	16,000	$ 320,000	14,325	$ 286,500	11,080	$ 221,600	$ 75,944,203
Gold, lode, ounces................	164,674	3,403,812	152,426	3,150,645	120,048	2,481,392	102,753,823
Silver, ounces................	3,498,172	3,215,870	3,403,119	3,592,673	3,377,849	3,235,980	53,668,294
Lead, pounds................	43,899,661	2,928,107	29,475,968	1,526,855	39,331,218	2,816,115	46,637,221
Copper, pounds................	61,483,754	15,143,449	42,459,339	7,939,896	44,887,676	7,832,899	161,513,864
Zinc, pounds................	41,772,916	2,899,040	56,737,651	3,540,429	47,208,288	3,077,979	19,896,466
Coal and coke tons (2240 lb.).....	2,491,212	12,833,994	2,358,680	11,975,671	2,662,917	13,450,169	212,573,492
Miscellaneous products.........		1,038,202		1,283,644		2,428,950	33,205,625
		$41,782,474		$33,296,313		$35,543,084	$706,192,978

² Adapted from report of Minister of Mines of British Columbia for 1920.

FIG. 1.—MINERAL DEPOSITS OF BRITISH COLUMBIA.

symmetry is lost. The deep-seated intrusives of Coast Range age and relationship are smaller and scattered haphazard throughout the belt. Consequently mineralization is widespread and mines are well distributed throughout the entire area. Railroads and lake navigation render this region easily accessible, although parts of it are extremely rough. The mineral possibilities of this border country, because of its greater accessibility and its proximity to the United States, have received more attention than those of any other part of British Columbia. On the other hand, the coastal area can be reached only by ocean-going vessels. The rugged, rock-bound coast with cliffs rising sheer from the water's edge to a height of 3000 ft. (915 m.) is more picturesque than hospitable. Timber is rare on the steeper slopes but abundant on the gentler, while the forest undergrowth is so dense as to be almost impassable until a trail has been cut. There are few trails and fewer roads to inland points, but the Provincial Government will build them to all properties possessing reasonable merit. In many places, a comparatively narrow band following the water's edge is all that can be considered accessible. If it were not that the coast is indented by many long, narrow, deep, navigable fiords, which in many instances penetrate entirely through the Coast Range barrier, the good mines and promising prospects along the interior contact of the range could not be worked.

These fiords belong to two definite systems: one parallel to the general northwest trend of the Coast Range and the other at right angles to it. With their sheer, lofty, parallel walls, the deep water with no beaches along their shores and their U-shaped cross-section, they are typically the result of glacial erosion of pre-existing river channels. The multitudes of tributary, gorge-like, U-shaped hanging and through valleys, and the polished, grooved surfaces of the rock in the walls offer abundant evidence of the intense glacial erosion of the country. Glaciers still feed many of the streams that empty into the fiords. The older and larger rivers have built up of the finely comminuted glacial silt great deltas —extensive mud flats that are usually submerged during high tide. The limits of these deltas are marked by an abrupt change from shallow to deep water conditions.

Ore Deposits in Eastern Contact Belt

There are many points of similarity between the ore occurrences in what may be called the inner contact belt of the Coast Range, of which the Alice Arm, Salmon River, and Atlin Districts are most important and may be considered typical. The dolly Varden, Premier, and Engineer orebodies, and the showings in the neighboring prospects are definitely associated with quartz-filled fissures, cutting the Mesozoic series of argillaceous sedimentary rocks and associated volcanics, largely

andesite tuff and breccia, that lie east of the Coast Range batholith. Numerous dikes and small bodies of intrusive rock related to the batholithic intrusion invade this earlier series and the final emanations from the same dying magma are probably responsible for most of the economically important mineralization. In the Dolly Varden there is silver with virtually no gold; in the Premier, the value of the gold and silver production is about equal; in the Engineer, nothing but gold is found. Secondary enrichment has been important in concentrating near the surface the silver in the two former mines. In the Dolly Varden, in particular, phenomenally rich native, silver, argentite, and pyrargyrite ore has been mined directly beneath the thin surface layer of muskeg and glacial debris.

Alice Arm, on the delta of the Kitsault River, is only 50 mi. south of Hyder at the mouth of the Salmon River, but it is separated from Portland Canal, into which the Salmon empties, by an impassable glacier-capped mountain range. Their comparative proximity may account for the marked similarity in the geology and ore occurrence in the two districts. Two systems of mineralized fissures, one varying in strike a few degrees from east and west and the other a few degrees from north and south, are characteristic of both. In the Premier, as in the Dolly Varden, the intersections of the two localize the primary and, to a still greater extent, the secondary deposits. Enriched ore has been developed to a depth of at least 200 ft. (60 m.) below the surface in both mines. The primary minerals in both instances are pyrite, sphalerite, and galena with minor amounts of argentite, proustite, pyrargyrite, stephanite, and rarely chalcopyrite. Large masses of quartz and great persistent veins are found in both districts in which these primary minerals are abundantly distributed through the quartz. Three conditions seem to be prerequisite to the occurrence of commercially valuable orebodies in these districts. A strong silicified fissured zone with low-grade primary mineralization must be cut by cross fractures of sufficient size and strength to permit secondary enrichment and the resulting enriched ore must have been protected from the erosional action of moving ice which has, in most instances, planed off the oxidized and enriched portions of orebodies throughout the northwest.

The ore in the Premier[3] occurs as small irregular lenses in a nearly

[3] R. G. McConnell: Portland Canal and Salmon River Districts. Can. Geol. Sur. *Memoir* 32, Parts I and II.

J. J. O'Neill: Salmon River District. Can. Geol. Sur. *Report*, 1920.

E. E. Campbell: Mineral Occurrences in the Stewart District. Can. Min. Inst. *Bull.* (March, 1920).

Victor H. Wilhelm: The Geology of the Portland Canal District. *Min. & Sci. Pr.* (1921) 122, 95.

Charles E. Prior: Notes on the Salmon River Mining District. *Min. & Sci. Pr.* (1920) 121, 518.

Charles Bunting: The Premier Gold Mine. *Can. Min. Jnl.* (1919) 40, 761.

east-west steeply dipping fissure zone about 80 ft. wide in schistose quartz porphyry, known locally as greenstone. A system of west-dipping faults, trending a few degrees east or west of north, approximately with the schistosity, cuts this sheared zone. The quartz mineralization follows both systems with the highest grade ore near the fissures and at the intersections, and lower grade complex sulfide milling ore in the blocks between the faults. As developed on the upper, or No. 1, level, the lenticular orebodies are in a block 750 ft. (230 m.) long and 80 ft. (25 m.) wide. Prospecting has been largely confined to following fissures of the two systems so that the openings are usually in good ore. In Tunnel No. 2, 248 ft. below level No. 1, a new pyritic oreshoot, lower in silver but higher in gold, was encountered in the summer of 1920 about 500 ft. (150 m.) west of the most westerly known high-grade orebody. It is probably associated with the developed fissure zone and if so gives it a total length of over 1200 ft. with ore developed at intervals. Oxidation is local but quite marked in the vicinity of the orebodies, particularly along the north-south faults. There is little doubt that the grade of the ore has been substantially increased by secondary enrichment, but there is a good chance that the primary ore will have sufficient value to show a profit with a mill on the ground.

The Dolly Varden[4] is working a badly faulted east-west quartz vein from 10 to 30 ft. (3 to 9 m.) in width with a dip of from 45° to 70° to the north. The vein, which traverses andesite, usually highly altered to greenstone, has been opened in six faulted blocks. The complex faulting has made development costly and difficult but has been largely responsible for the secondary enrichment, which has given the ore much of its value. The vein is developed for a length of 480 ft. to an average depth of 200 ft. Its eastern end is on a large fault while the other end is split into a series of unimportant stringers. Native silver and argentite are particularly abundant near the surface while more pyrargyrite is found at greater depth.

The Taylor Mining Co., which owns the Dolly Varden[4] mine and the 17-mi. narrow-gage railway to Alice Arm, also owns the Wolf Group 2 mi. north of the Dolly Varden. Here a large tonnage of low-grade highly siliceous ore has been developed by diamond drilling in two strong parallel veins striking about N. 20° E. with conspicuous outcrops from 10 to 50 ft. in width. Proustite, pyrargyrite, argentite, and some native silver can be found here and there in the outcrops but cross fracturing is absent and

[4] J. M. Turnbull: Alice Arm District. Minister of Mines British Columbia *Report* (1916).

A. W. Davis: The Dolly Varden Mine. Can. Min. Inst. *Bull.* (Feb., 1920).

A. J. T. Taylor: The Dolly Varden Mine. *Min. & Sci. Pr.* (1920) 120, 631.

Robert Dunn: The Dolly Varden Mine. *Eng. & Min. Jnl.* (1920) 109, 643.

enrichment is unimportant. Many strong veins in the district have good deposits, but they are rarely concentrated into workable orebodies.

The Engineer,[5] which is 300 mi. north of Alice Arm, just south of the Yukon on the shore of Taku Arm of Lake Tagish, is not a developed mine; it is remarkable for the shoots of rich gold ore that occur as small plums in the many tiny quartz fissure veins that cut dark gray, fine-textured slate and shale. The area of sedimentary rocks is close to a great mass of the Coast Range intrusive rock, largely granodiorite, and is invaded by andesite dikes. The gold deposits seem to be favorably influenced by the presence of the intrusive andesite. The veins cut the dikes and, where andesite forms both walls, are narrow and tight, but the vicinity of the intrusive rock, especially where the vein is cut by cross fractures, is the most favorable place for rich ore. The gold seems to occur exclusively as the native metal in small flakes, crystals, and dendritic forms, often symmetrically surrounded by a dark green aureole of chlorite, but also intergrown with quartz or as leaves on the cleavages or crystal faces. Its association is typically primary. The fact that it occurs with undiminished value at a depth of 200 ft. (60 m.) in the bottom of the main shaft on the only vein that has been consistently developed corroborates this assumption. In this intensely glaciated country oxidation is only superficial. The distribution of the gold in the vein is most irregular and erratic. Small lenticular masses are lavishly impregnated with visible metal but separated from one another by long stretches of barren vein material. Some of the veins consist wholly of quartz, others contain much calcite and crushed seamed slate. Chloritic alteration is characteristic of some but entirely absent in others. A little pyrite, usually in tiny crystals, is quite universally distributed through the veins and the immediately adjacent country rock. Fine needles and specks of stibnite and possibly native antimony are occasionally found. The rare mineral, allemontite, occurs in reniform masses in one of the veins. The presence of antimony minerals in the vein seems to be a good indication of gold.

About 100 mi. (160 km.) farther north, near Whitehorse, is a contact-metamorphic belt about 12 mi. (19 km.) long and 1 mi. (1.6 km.) wide from which a substantial tonnage of copper ore has been shipped. While this is also in the zone along the eastern contact of the Coast Range batholith, the deposits are entirely dissimilar to those described, resembling rather the small copper occurrences on the islands of the west coast. The orebodies are all of contact-metamorphic origin in limestone at or near its contact with large intrusive granite masses. The copper minerals are associated with magnetite, or with specularite, hematite, and limonite, or with garnet and tremolite. Some rich lenses are found, the belt is extensive and has many attractive showings, but all the properties

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[5] D. D. Cairnes: Atlin District. Can. Geol. Sur., *Memoir* 37 (1913).

have the same history, common to most of those in which contact-metamorphic copper orebodies occur—spasmodic operations on a small and costly scale, mining good ore but showing little or no ultimate profit.

ORE DEPOSITS IN WESTERN CONTACT BELT

Near Juneau, on both Douglas Island and the mainland, situated on the Pacific side of the Coast Range batholith, which at this point is about 50 mi. wide, are gold mines[6] famous for their large-scale low-cost operations. The Alaska-Gastineau,[7] Alaska-Juneau, and Alaska-Ebner have developed large but very low-grade gold-bearing deposits in a steeply dipping, mineralized, shattered zone about 200 ft. wide in graphitic slate cut by metagabbro dikes. Although dislocated by several large faults, the zone has a general N 25° W. trend and the mineralization may be traced for several miles between albite mica schist on the east and green slate, greenstone and amphibolite on the west. The fracture zone is mineralized with a network of reticulating quartz veinlets with associated pyrite, sphalerite, galena and free gold. The gold distribution is irregular so that selective mining is difficult, if not impossible. The orebodies are often 100 ft. wide, the value gradually decreasing but continuing in the weaker disseminated sulfide mineralization far into the hanging wall. In the Alaska-Gastineau, the lodes in the ore zone have a flat rake to the southeast.

Across Gastineau Channel, on Douglas Island, the famous Treadwell mines have been working, for almost 40 years, a similar but wider zone in which albite diorite dikes, mineralized with quartz, metallic sulfides and gold, are intrusive into black graphitic slate. The deposits are largely confined to the dike material with its network of quartz veinlets but the orebodies of commercial grade are sometimes very large. As in the Juneau deposits, selective mining is impracticable, but the average grade of the Douglas Island ore has been higher than that of the mainland bodies and the mines have been very profitable. Other low-grade gold mines, such as the Jualin and the Kensington, and the smaller higher grade veins of Chichagoff and other islands, are also found in the coast belt west of the Coast Range batholith. Even more typical of the mineralization in this belt are the multitude of limestone-copper deposits of the contact-metamorphic type, exemplified by the Marble Bay mine[8] on Texada Island and several on Vancouver, Quadra, Queen Charlotte, and Prince of Wales Islands. The aggregate production from this source

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[6] A. C. Spencer: The Juneau Gold Belt. U. S. Geol. Sur. *Bull.* 287.

Henry M. Eakin: Mining in the Juneau Region. U. S. Geol. Sur. *Bull.* 622.

G. F. Becker: Reconnaissance of the Gold Fields of Southern Alaska with Some Notes on the General Geology. U. S. Geol. Sur., *Ann. Rep. Dir.* (XVIII), Part III.

[7] G. T. Jackson: Mining Methods of Alaska Gastineau Mining Co. *Trans.* (1920) **63**, 464.

[8] Victor Dolmage: The Marble Bay Mine. *Econ. Geol.* (1921), **16**, 6.

has been substantial but the orebodies are irregular and it is doubtful if any great mines will be found among them.

ORE DEPOSITS IN INCLUDED BLOCKS WITHIN BATHOLITH

The two most important copper mines in British Columbia, the Hidden Creek at Anyox and the Britannia on Howe Sound near Vancouver, have been developed in included blocks or "roof pendants" within the Coast Range batholith. The Hidden Creek orebodies[9] occur as great lenses at or near the contact of a large intrusive greenstone mass with highly altered slate and argillite. The greenstone and slate area is a block 5 mi. wide and 10 mi. long included in the more recent batholith, by the granitoid rocks with which it is completely surrounded. Recent dikes of many types, but probably late phases of the Coast Range intrusion, cut the ore-bearing formations and the orebodies themselves. The slates are intensely twisted and contorted, especially near the greenstone contact where they are often sheared and altered to a mica schist. The mineralization, as shown by development, seems to have been most important where the greenstone forms embayments in the slate. On the eastern edge of a major embayment, 3000 ft. (915 m.) across, heavy shearing and intense silicification have taken place in a zone 1000 ft. wide and 3000 ft. long. In this zone are six distinct lenticular orebodies, the largest of which is from 100 to 300 ft. wide, 1500 ft. long, and extends from the surface to a depth of at least 1000 ft., but with a very irregular outline. The other orebodies are smaller but of substantial size. Only three lenses outcrop; the others have no surface expression and were found by diamond drilling. The ore consists of a primary combination of pyrite, chalcopyrite, and pyrrhotite with a little magnetite and varying amount of silica. Three of the bodies are fairly massive sulfide ore with 30 per cent. sulfur and 10 to 20 per cent. silica. The others carry up to 40 per cent. silica and cannot be smelted directly. A flotation mill is being designed for this more siliceous material. About 11,000,000 tons of 2.24 per cent. copper ore carrying about 60 cents in gold and silver and a large additional tonnage of about 1 per cent. copper are reported developed.

The inclusion in the Coast Range batholith in which the Britannia orebodies[10] are found is roughly 2 mi. wide and 7 mi. long with its axis trending in an east-west direction. It is made of black argillite with intruded sills of igneous rock, usually quartz porphyry, which conformably

[9] R. G. McConnell: Observatory Inlet. Can. Geol. Sur. *Memoir* 32, Part IV.
Wakely A. Williams: The Granby Mine at Anyox, B. C. *Eng. & Min. Jnl.* (1917), **104**, 629, 707.
Donald G. Campbell: Hidden Creek Mine and Smeltery. *Eng. & Min. Jnl.* (1917), **103**, 274.
[10] S. J. Schofield, Britannia Map Area. Can. Geol. Sur., Summary Report, 1918, B.

follow the bedding. All have been tilted to a steeply dipping position. Mineralization is confined to a strong, persistent shear zone in the quartz porphyry of what is known as the Fairview sill. The sill itself is 2500 ft. wide and the maximum width of the shear zone is close to 1000 ft. (300 m.). Both can be traced for 5 mi. and probably extend for an even greater distance. In the shear zone the quartz porphyry becomes a fissile quartz-mica schist, in which great lens-shaped masses of chalcopyrite, pyrite, a little sphalerite, and galena with variable amounts of quartz are found. The sulfide minerals occur disseminated in and replacing the schist and also frequently as roughly tabular veins following the schistosity. The ore-bodies are localized almost without exception along the axes of large, steeply dipping folds in the shear zone. At intervals in the same shear zone are veins and lenses of both gypsum and native sulfur, which have not so far been found associated with each other or with the copper mineralization; these minerals, however, are of scientific rather than economic interest.

The developed copper orebodies are four in number and a fifth is indicated by recent work. They are included in the 7000-ft. section along the strike of the zone, which has been more than superficially prospected. The largest body and the most important producer is the Fairview, the maximum dimensions of which are 500 ft. by 1500 ft. with ore developed to a depth of 2200 ft. The ore that can be mined in this great lens is confined to twelve parallel veins from 10 to 70 ft. in width with varying amounts of low-grade or barren schist between. The veins pinch and swell; some frequently join or the mass between is sufficiently mineralized to permit several veins to be mined as one body; on the other hand veins often split or become too low grade to be mined profitably. The other lenses are smaller and separated from one another and from the Fairview by low-grade or barren sheared porphyry. In one place oxidation extended to a depth of about 200 ft. In this zone some oxidized and secondarily enriched ore was found. Elsewhere oxidation has been superficial and there has been no appreciable enrichment of the primary sulfides. About 10,000,000 tons are estimated to average 1.9 per cent. copper and about 25 cents per ton in precious metals. The extremely rugged topography permits the extraction through tunnels of even the deepest ore. Unlike much of the Hidden Creek production, this ore can be concentrated. The 2200-ton concentrator at the beach, 3 mi. away, was recently destroyed by fire, so that the Britannia is closed. A new steel-and-concrete 2500-ton plant is now being constructed.

ORE DEPOSITS IN SOUTHERN BORDER BELT

The third large copper mine of the province is that brought to the point of production in the fall of 1920 by the Canada Copper Corpn.

at Copper Mountain[11] in the Similkameen district, near Princeton. It is about 30 mi. north of the international boundary and 180 mi. east of Vancouver. In this district, granitoid rocks with small included masses of highly altered argillite, limestone, and quartzite are traversed by recent light-colored porphyry dikes. It is the granitoid rocks and the occasional sedimentary blocks that are mineralized. The earliest of the former is an augite-monzonite that has been invaded by great masses of granodiorite and still later by pegmatite and granite. The intrusion of the granodiorite and granite and the subsequent mineralization are probably related to the intrusion of the Coast Range batholith, which has been so productive of mineralization throughout the northwest. Fracture zones, usually steeply dipping, cut all these rocks and the mineralization is intimately connected with them. The recent porphyry dikes trend with the major structure, N. 10°–30° W., but are not affected by the fissuring and in some instances cut sharply through the orebodies. Mineralization in the igneous complex is widespread. In the fracture zones the rock is often highly altered to a dense, green, cherty material. Pyrite, chalcopyrite, bornite, and pyrrhotite occur both as disseminated crystals in the rock mass and in tiny veinlets and seams with quartz, calcite, biotite, and feldspar. Magnetite and specular hematite, often intergrown with sulfide minerals, are plentiful in the northern and eastern parts of the district. Superficial oxidation has imparted a rusty color to the great mineralized zones. Although copper carbonates are found in some places near the surface and limonite occurs along open fissures to a depth of 100 ft., primary sulfides invariably come to within a few inches of the surface and nowhere has secondary enrichment been important.

At the Copper Mountain mine, eleven orebodies, lenticular in plan, have been developed by extensive diamond drilling in the central 3300 ft. of a fracture zone over 1 mi. long and, in places, nearly 1000 ft. wide; the average width of this zone is, perhaps, 200 ft. The ore lenses grade from the Sunset, which is 750 ft. long, 725 ft. deep, and 300 ft. wide, to small outlying orebodies 100 ft. long, 100 ft. deep, and 50 ft. wide. Only five lenses outcrop. The others were found by diamond drilling. The major axes of the lenses trend N. 30° W., the strike of the fracture zone, but parallel orebodies occur along east-west fractures. The intersection of the two systems seems to be important in localizing the ore. The depth of the lenses is normally the same as their length. While the lens having the greatest vertical dimension is 725 ft. in depth, ore has been developed to the bottom of some of the deepest drill holes, more than 1000 ft. below the surface. The value of the deposits decreases toward the walls so that the ore limits are commercial, except where a porphyry dike forms one or both walls, when the line of demarcation is sharp and the

contact often carries a little clay gange. The developed ore, of which there is roughly the same tonnage as in the Hidden Creek and Britannia mines, averages about 1.74 per cent. copper and about 30 cents in recoverable gold and silver per ton. The deep gorge-like Similkameen Valley, which approximately parallels the ore-bearing zone, offered an ideal opportunity for developing the mine by tunnels. An 8-mi. railroad connects the ore-bins at the lower tunnel mouth with the 2000-ton concentrator at Allenby.

A little over 100 mi. southeast of Princeton are the Rossland[12] and Boundary[13] districts, in which for many years the mines at Rossland, Phoenix and Greenwood have produced a substantial annual tonnage of copper-gold ore. The deposits at Phoenix and Greenwood are now virtually exhausted but the Rossland Camp will produce for many years. The Consolidated Mining and Smelting Co. of Canada, which owns mines at Rossland and the Trail Smelter, also owns the Sullivan mine,[14] 100 mi. to the east near Kimberley in the East Kootenay district and near the boundary of the Province of Alberta. Here a large replacement deposit of silver-bearing lead-zinc ore has been developed in a great thickness of pre-Cambrian quartzites, which has been invaded by sills of gabbro. The whole series has been greatly folded and fractured. The orebodies in the East Kootenay district occur both in veins and as replacement deposits in the lower quartzite beds. A little garnet and occasionally other contact-metamorphic minerals are intimately associated with the ore minerals, indicating the high-temperature origin of the deposits. The gabbro is older than the ore and is often mineralized. The source of the mineralization is doubtful as no other igneous rocks are found, but is probably some deep-seated unexposed intrusive.

The primary minerals in the Sullivan deposit are pyrite, pyrrhotite, sphalerite and galena, usually but not always intergrown. Shoots are found in which one mineral occurs almost to the exclusion of the others and attention was first confined to the bodies of comparatively zinc-free lead ore. Recent metallurgical progress, however, has rendered the hitherto refractory lead-zinc-iron sulfide ore amenable to economic separation and it is on this type of material that the Sullivan mine will largely depend for its future production. A single bedded mineralized zone in the quartzite series has been developed for about 3000 ft. along its north-south strike and for about 1600 ft. along its average dip of 23° to the east. In this zone, the quartzite is almost completely replaced

[12] Charles Wales Drysdale: Geology and Ore Deposits of Rossland. Can. Geol. Sur., *Memoir* 77.

[13] O. E. LeRoy: Mother Lode and Sunset Mines, Boundary District. Can. Geol. Sur., *Memoir* 21.

[14] S. J. Schofield: Geology of the Cranbrook Map-Area. Can. Geol. Sur., *Memoir* 76.

by massive sulfide minerals for an irregular thickness, which varies from 10 to over 100 ft. The bedded vein, so called, has been opened by two long tunnels and further prospected by a large number of diamond-drill holes.

The local concentration or segregation of the primary minerals in this great sulfide bed is noteworthy. When the upper tunnel was first driven, it appeared that the vein as exposed could be roughly divided mineralogically into three sections along its strike. In the south section nearest the portal, a preponderance of lead was found; in the central section, the mineralization was almost pure pyrite; while in the northerly third, zinc predominated and the iron was almost entirely in the form of pyrrhotite. Recent developments seem to have proved that what at first appeared to be a broad zonal arrangement of the minerals was only a local phenomenon. In the lower tunnel, however, at a greater vertical depth of about 700 ft. the intergrown iron sulfide is pyrite instead of pyrrhotite. While the entire zone is continuously mineralized, the ore occurs in irregular shoots in much more abundant pyrrhotite or pyrite, often with a greater proportion of lead in the heart of the shoot, grading into a more zincy part and then into a shell of iron. Local rolls along both strike and dip sometimes form great dome-shaped orebodies 100 ft. or more thick, although the average thickness of the mineralized bed is between 25 and 30 ft. A large tonnage of ore carrying enough combined zinc, lead, and silver to show a substantial profit with reasonable metal prices has been developed. A site near the mine for a mill with a probable initial capacity of 2000 tons a day has been selected. The results obtained in the company's experimental mill at Trail have been so encouraging that preferential flotation will probably be used for the separation and the zinc product handled in the electrolytic zinc plant at Trail. The mine is destined to be an important producer for many years.

The St. Eugene mine[15] at Moyie, 25 mi. south of the Sullivan, is typical of the vein deposits in the quartzite; now approaching exhaustion, it has produced over 1,000,000 tons of lead-zinc-silver ore of a gross value of more than $10,000,000 from two quartz veins within a vertical range of 2600 ft. The veins are steeply dipping, parallel, and about 600 ft. apart. Intersecting fissures have localized the oreshoots, which in many cases are irregular replacement deposits up to 30 ft. thick in the purer quartzite beds. The veins themselves showed only about 5 ft. of ore where they cut the more favorable beds and pinched to small dimensions where the more argillaceous quartzites formed the walls. In this southern portion of the arc-shaped mineralized belt of British Columbia are many little mines but few of them are as important economically as the St. Eugene.

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[15] S. J. Schofield: Geology of the Cranbrook Map-Area. Can. Geol. Sur. *Memoir*, 76.

Within this great curved zone, extending for 1500 mi. from the Alberta boundary in the southeast to beyond the Yukon line in the northwest, it is almost certain that more large mines will be found. There are great stretches along the eastern contact of the Coast Range batholith that have not been explored. With the exception of the comparatively narrow band made accessible by the navigable fiords, Portland Canal and Observatory Inlet, and the Grand Trunk Railway along the Skeena River, the inner contact from Lillooet for 800 mi. north to the Taku River may be considered virgin, almost unprospected, territory. There is also every reason to believe that other included blocks of ore-bearing rocks similar to those at Anyox and Howe Sound will be found within the great granite range. When more trails are built and the heart and eastern slopes of the mountain chain become more accessible, this curving coastal backbone is sure to develop into an ever increasingly important asset to the mineral wealth of the Province and of the Dominion.

DISCUSSION

Hugh R. Van Wagenen,[*] Allenby, B. C. (written discussion).—It is strange that some really large copper orebodies have not been discovered, because the evidence of and the proper conditions for the presence of the metal are widespread. Of course, large portions of the area have not been closely prospected for it requires strong financial backing to investigate rock in a place that is hidden away and guarded by the factors mentioned. Eventually very much copper will be won in British Columbia, I believe.

In discussing the Copper Mountain mine the author says that the ore is limited (other than by dikes) by commercial walls. This is correct, but not in the sense of the commercial limitations to deposits in limestones, or other expressions of metasomatism. At Copper Mountain, as the mineralization is infiltration filling fractures there is an end to the ore where the fractures are insufficient to bring the whole up to grade, and this transition frequently occurs sharply. Occasionally the wall is a "teaser," carrying the usual good grade of bornite and chalcopyrite in fractures spaced just far enough apart to make the whole below pay, yet leading into other more thoroughly fractured areas only a few feet distant.

[*] General Manager, Canada Copper Corpn. Ltd.

Deposition of Copper Carbonate from Mine Water

By Philip D. Wilson,* E. M., Warren, Ariz.

The genesis of some orebodies has been explained by the mingling and chemical interaction of water solutions of different compositions and the consequent precipitation of the mineral load of one or both solutions at their confluence. While this process can be proved in the laboratory, rarely can a positive demonstration be found in nature, for the conditions at the time an ore deposit is being worked are usually entirely different from those that existed when the orebody was formed. For this reason, the present deposition of copper carbonate in the southern end of the Briggs mine of the Calumet and Arizona Mining Co. near Bisbee, Ariz., is of great interest.

The developed orebodies in the Calumet and Arizona mines are roughly confined to two broad parallel zones separated at the north by a wedge of Copper Queen ground and at the south by an undeveloped block of Calumet and Arizona ground about 2000 ft. (610 m.) wide. The more westerly zone includes the Irish Mag, Oliver, P. and D., and Cole mines, none of which has been operated since 1919. Much of the water from this division is carried from the 1400-ft., or bottom, level at the southern end of the Cole mine in a long water drift through the undeveloped block to the 1300-ft. level of the Briggs mine at the southern end of the more easterly zone. Most of this water comes originally from outlying areas in the P. and D. mine, passing through parts of the lower levels of the Cole mine, which are heavily mineralized with iron and copper sulfides. This sulfide zone has been opened up and. slowly oxidizing for years. Originally a pure, potable water, the analyses of which are shown under A in Tables 1 and 2, on its way through the mineralized area in the Cole mine it gathers copper, calcium, magnesium, iron, and aluminum sulfates, a load of dissolved mineral salts almost four times as great as that which it formerly carried; analyses B

' Chief Geologist, Calumet and Arizona Mining Co.

represent this water. Its carbon dioxide has entirely disappeared, the carbonates having been changed to sulfates. While it contains no free acid and is neutral to methyl orange, it shows a decidedly acid reaction to phenol phthalein. It also has an extremely low chlorine content. The iron and copper sulfide zone in the Cole through which it has passed contains a small amount of galena, insignificant in percentage but substantial in the aggregate, so that the lead may have appropriated the chlorine and fixed it as insoluble lead chloride.

TABLE 1.—*Analyses of Residues in Percentages of Radicals or Ions and of Total Anhydrous Inorganic Solids*[1]

	A	B	C	D
SiO_2	4.16	1.44	4.18	2.13
SO_4	11.44	70.45	28.80	48.11
CO_3	39.84	*Nil*	27.42	13.70
Cl	10.62	0.89	7.64	6.73
Na	*Nil*	0.58	*Nil*	*Nil*
Ca	26.11	16.61	25.96	22.42
Mg	6.84	6.59	5.22	6.36
$Fe_2O_3Al_2O_3$	0.99	0.50	0.78	0.42
Cu	*Nil*	2.94	*Nil*	0.13
	100.00	100.00	100.00	100.00
Salinity, parts per million	362.51	1360.92	459.52	660.92
Gallons per minute	55	55	200	255

[1] These analyses were made by Henry S. Hollis, industrial chemist, mechanical department, Calumet and Arizona Mining Co., who has done excellent work in connection with the determination of the corrosive qualities of mine waters and their commercial treatment.

A. Fresh pure water from the 1250-ft. level, P. and D. mine; no work has been done in this country for years and virtually no ore has been found in it.
B. Same water after passing through Cole mine from the 1000-ft. to 1400-ft. level, where some of it seeps through zones mineralized with pyrite and chalcopyrite, which have been opened up and exposed to oxidizing influences for several years; the rest of it is carried in open ditches in the floor of drifts through these same zones.
C. Fresh pure water from the southern end of the Briggs mine, 1300-ft. level, from country in which little work has been done and no orebodies known.
D. Mixture of *B* and *C* in the ratio of 55 gal. of the former and 200 gal. of the latter per minute. Sample taken about 50 ft. below the point of confluence.

Where the mixture between this highly saline water and the fairly pure alkaline water from another outlying undeveloped part of the Briggs mine, the analysis of which is given under *C*, takes place a

bright green deposit of hydrous copper carbonate, essentially malachite in composition, is formed as a crust on the limestone in the ditch. That the reaction is almost instantaneous and quite complete is shown by the negligible percentage of copper in the residue analysis of the mixture D, in a sample taken close to the confluence of the two. The mixed water is carried in a drift for about 1300 ft. (395 m.) and then dropped through a 100-ft. (30 m.) raise to the level below. For this entire distance the rock in the ditch and in the raise is painted a brilliant green. An analysis of the water issuing from the foot of the raise did not show a trace of copper, but the suspended matter gave a definite qualitative test for it. The chemical reaction has apparently been completed and virtually all the copper precipitated long before the water reaches the raise. The copper precipitate, however, in a finely divided state is carried in the stream for a long distance and gradually deposited. The natural agitation the mixture receives while tumbling down the rough walls of the raise pretty well completes the deposition of the solid matter. The ditch that carries the water is stained for only 2 or 3 ft. beyond the foot of the raise.

TABLE 2.—*Analyses in Terms of Grains per United States Gallon*

	A	B	C	D
SiO_2................	0.88	1.14	1.12	0.82
$Fe_2O_3Al_2O_3$................	0.21	0.40	0.21	0.16
$CaCO_3$................	13.80	*Nil*	12.25	8.80
$CaCl_2$................	*Nil*	*Nil*	*Nil*	*Nil*
$CaSO_4$................	*Nil*	44.88	7.00	17.41
$MgCO_3$................	0.20	*Nil*	*Nil*	*Nil*
Na_2CO_3................	*Nil*	*Nil*	*Nil*	*Nil*
$MgSO_4$................	3.04	25.96	3.48	7.78
$MgCl_2$................	3.02	*Nil*	2.75	3.47
Na_2SO_4................	*Nil*	*Nil*	*Nil*	*Nil*
$NaCl$................	*Nil*	1.17	*Nil*	*Nil*
$CuSO_4$................	*Nil*	5.85	*Nil*	0.12
	21.15	79.40	26.81	38.56
Causticity,[2] (phenol phthalein).	Neutral	18 acidity	Neutral	3 acidity
Alkalinity,[2] (methyl orange)....	24	Neutral	21	15
Hardness[2]................	36.5	118	39	59

[2] The numbers used in these comparative classifications are arbitrary; they represent no definite standard chemical conditions, but comparisons for commercial purposes.

The relation between the water's capacity to carry dissolved copper and its acid or alkaline condition is clearly indicated by these analyses. Copper will be carried in solution by a normally neutral or alkaline (to

methyl orange) water just as long as the water shows a definite acid reaction with phenol phthalein. The water's copper-carrying capacity decreases directly with its phenol-phthalein acidity until it is practically nil, when it reacts neutrally, as in the case of the water emerging from the raise in the foregoing description. The water showed a strong alkaline reaction with methyl orange but was barely neutral with phenol phthalein. In other words, appreciable copper may be carried and iron corroded by water definitely classified as alkaline by the ordinary methyl-orange test. The "causticity," as shown by phenol phthalein, should be the significant reaction in commercial analyses for the purpose of determining the corrosive properties of mine waters.

The Ore Deposits of Sierra Mojada, Coahuila, Mexico

BY S. F. SHAW,* E. M., ESMERALDA, COAHUILA, MEX.

THE Sierra Mojada mining district is situated in western Coahuila, about 8 km. east of the Chihuahua state line, and about one-half way from the northern to the southern extremity of the state. It is at the terminus of the Mexican Northern R. R., 125 km. northeast of Escalón, Chihuahua; also about 240 km. due west of Monclova, Coahuila. The altitude is approximately 1500 m. above sea level.

The Sierra Mojada valley is about 10 km. long, from east to west, with a width of 1 to 2 km. It is approached from the east through the Puerto del Oro, where the Mexican Northern R. R. enters the valley; the slope from west to east is gentle. The Sierra Mojada, of which about 300 m. is an almost vertical wall, bounds the valley on the south side; on the north side, the slope to the crest of Sierra Planchada is more gradual, rising only 200 or 300 m. above the valley.

Water is relatively scarce in the district, and there are no running streams. The rainy season, usually beginning at the middle of June, lasts through July, and occasionally into August or September. In some years the rains fail entirely. The slope of the ground is such that nearly all the water immediately runs off. The arroyos along the flanks of the mountains are not very deep, but in heavy rains, sometimes approaching cloudbursts, a tremendous volume of water rushes down them, carrying large boulders.

A certain amount of seepage collects in the agglomerate, lying below the conglomerate, and is retained in the local fractures. Some shallow wells have been sunk in this agglomerate, and one well over 150 m. deep was sunk on the Volcan ranch, supplying about 76 l. (20 gal.) per min. A considerable supply of water is collected in large earth reservoirs in and below the town of Esmeralda. In ordinary years this supply will last until January or February, after which the Volcan well has to be depended on until the next rainy season.

Vegetation is scarce, aside from scanty scrub oak and other brush in the mountains, and a few mezquite, locust, pepper, and cottonwood trees.

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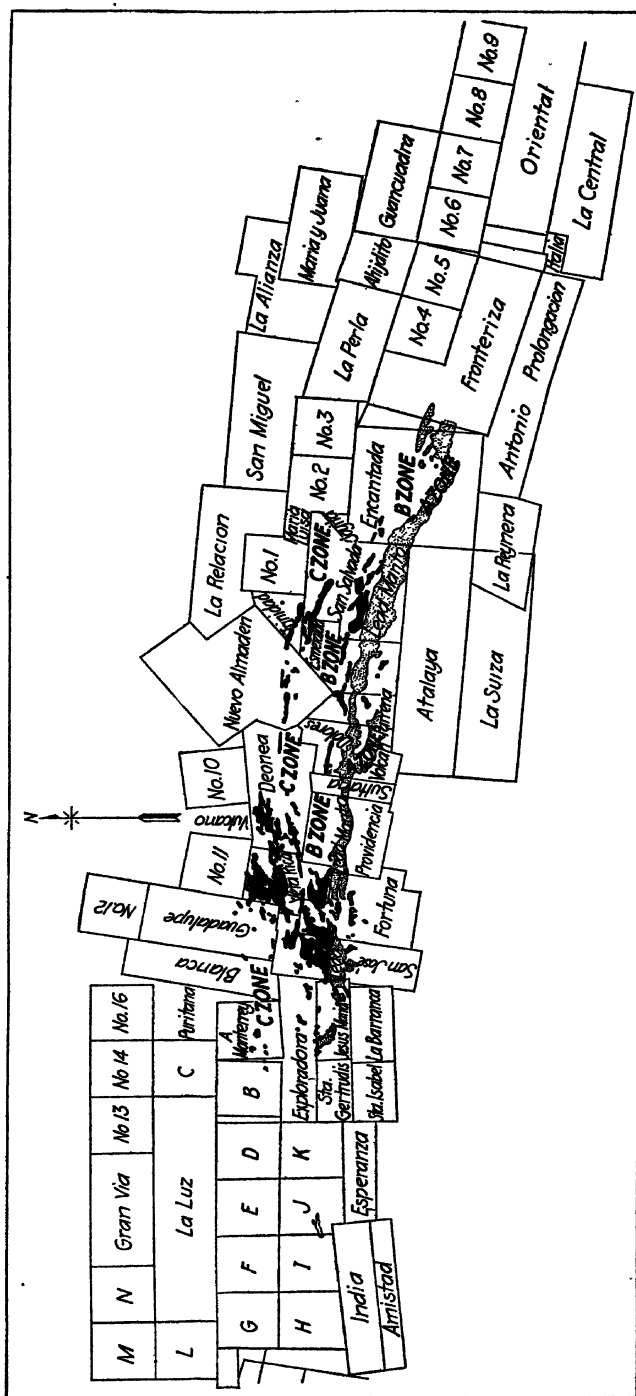
FIG. 1.—CLAIM MAP, SIERRA MOJADA DISTRICT, COAHUILA, SHOWING ORE ZONES.

The climate is delightful throughout the year, with few cloudy days. The maximum range of temperature for a whole year was from 45° F. in December to 84° in June.

In the early days of Sierra Mojada, the ore was transported by wagons or carts to Escalón or to Monclova. Since 1891, when the Mexican Northern was completed, this has been the only rail communication for the district. A standard-gage railroad from Monclova was started about 1892, but was completed only as far as Cuatro Ciénegas, 67 km.; this road is now being extended by the Mexican Government to Sierra Mojada, and will join the Mexican Northern 14 km. from that point, making the total distance from Monclova to Sierra Mojada 240 km. This new road will shorten the distance to the Monterrey smelters and to the border, via Monclova and Eagle Pass, and will enable the operators to obtain comparatively cheap coal from the Sabinas coal fields.

GEOLOGY

Rocks

The three principal rocks in the district are: Limestone, agglomerate, and conglomerate or alluvium.

Limestone.—The oldest rock in the district is a Cretaceous limestone, which is exposed in all the higher points and in the surrounding hills. In the valley, it is covered by agglomerate and conglomerate.

On the surface the rock is usually light colored. It forms abrupt slopes and almost perpendicular precipices on the Sierra Mojada or south side of the valley, but the slope of the northern side is more gentle. The beds slope fairly regularly to the east-southeast with a dip of from 15° to 20°. They show a thickness of at least 3000 ft. (914 m.) along the Sierra Mojada, although their base is not exposed at any point. Diamond-drill holes put down in the valley to a depth of 1000 ft. (305 m.) or more are said to have penetrated shale, quartzite, and even igneous rock, but the records appear to present some uncertainty.

The beds are composed mainly of uncrystallized and semi-crystallized or re-crystallized limestones. In the vicinity of the orebodies, and near the surface, there are relatively small quantities of altered limestone, designated by Garrey as sheared limestone, leached limestone, and land-slide limestone. The rock has been classified as Cretaceous although only imperfect fossils have been discovered; however, practically all of the table land in this portion of Mexico has been found to consist of Cretaceous limestone.

Except in the vicinity of the ore deposits, the limestone is almost free of silica and iron. Certain of the beds contain considerable magnesia, and it has been suggested that the magnesian limestone has been more favorable for replacement by the metal-bearing solutions; subsequent iavestigations have not borne out this theory.
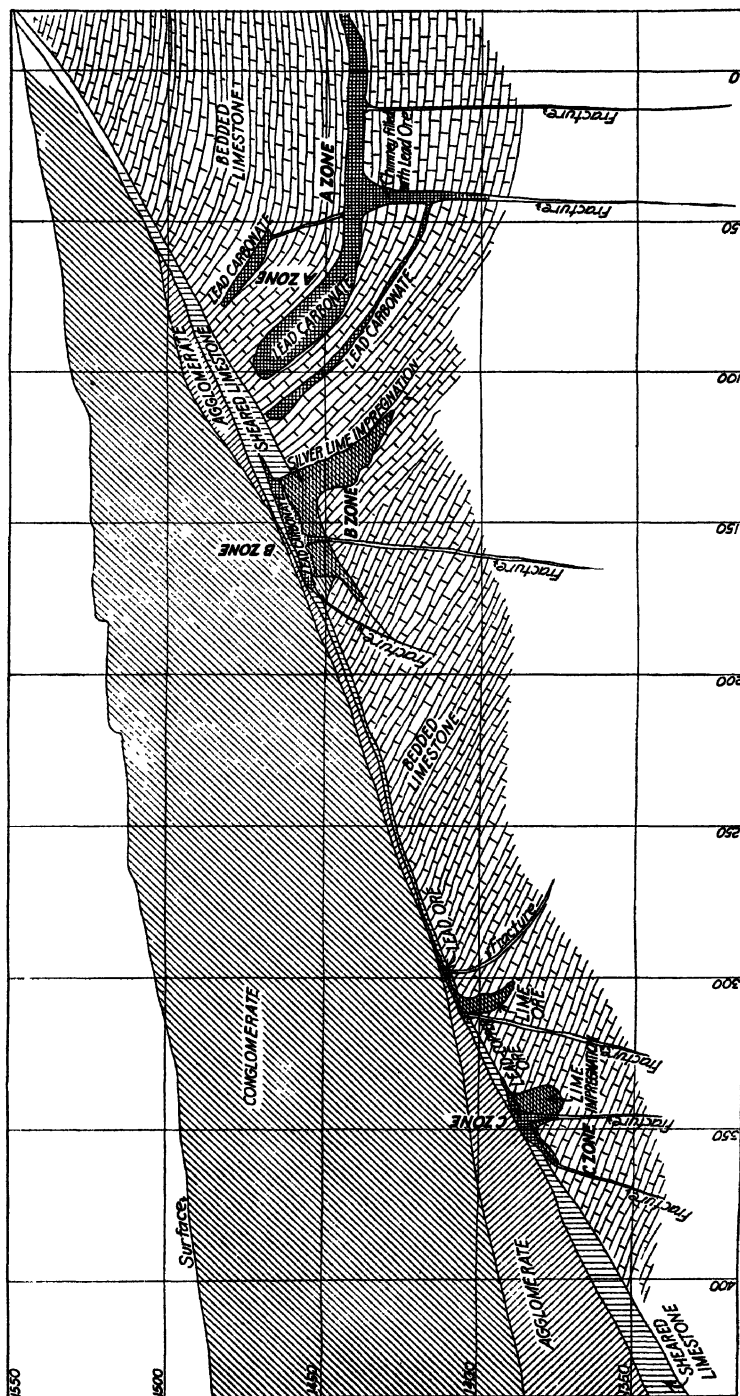
FIG. 2.—IDEAL N-S SECTION, SIERRA MOJADA, COAHUILA.

Agglomerate.—The valley and the flanks of the mountains on both sides have been covered with a volcanic breccia designated by Garrey as agglomerate. The source of this rock is unknown. It may have originated from a vent to the west of the mineralized district, flowing into the valley and filling it to a much greater depth than appears at present. Another theory is that the line of weakness presented by the east-west axis of the limestone anticline, now occupied by the valley, is probably the source of the agglomerate flow. The agglomerate, wherever visible, is much altered; along the flanks of the mountains it has weathered to a rust-colored material, embedded in which may be seen fragments of limestone and of igneous material different in character from the matrix. Nearer the middle of the valley, where No. 10 shaft penetrated the agglomerate, it is found comparatively fresh, showing fragments of limestone and granite embedded in the volcanic material.

The original mass was evidently quite thick. Two hills of this material on the Blanca and "C" claims, which have been protected by a capping of cemented limestone, indicate a thickness of approximately 1000 ft. (305 m.) or more; the original thickness was doubtless greater than it is at present.

Conglomerate (Alluvium).—The flanks of the mountains are covered, in places, by a blanket of cemented debris, consisting of limestone boulders and pebbles, which extends over the valley, burying the agglomerate to a known depth of 510 ft. (155 m.) on the Volcan ranch. The conglomerate is quite porous, and water drains through it rapidly until it reaches the agglomerate.

Structure

The limestone strata in the Sierra Mojada dip 15° to 20° to the southeast, while those in the Sierra Planchada dip to the north at about the same angle. It appears likely, as mentioned by Chism,[1] that the present valley was once the site of an anticline, the axis of which had a strike to the east-southeast.

Minor faulting is observed in practically all of the mines, but the throw, as a rule, is small. There is a fault having a north-south strike just west of the India claim and passing through the village of Sierra Mojada. The agglomerate capping is missing directly west of this fault, but at a distance of 1 km. to the north of the India mine the contact is seen to continue to the west of the fault, and a mineralized area is found in the San Francisco mine.

A series of small faults with horizontal throws 10 to 50 ft. (3–15 m.), and striking N. 10° W., is seen in the Jesús María mine, where the agglomerate is displaced, indicating that the faulting occurred after the agglomerate was laid down; these faults extend into the Exploradora mine to the north. To the northeast of the Jesús María faults, another

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[1] *Trans.* (1886) **15**, 547.
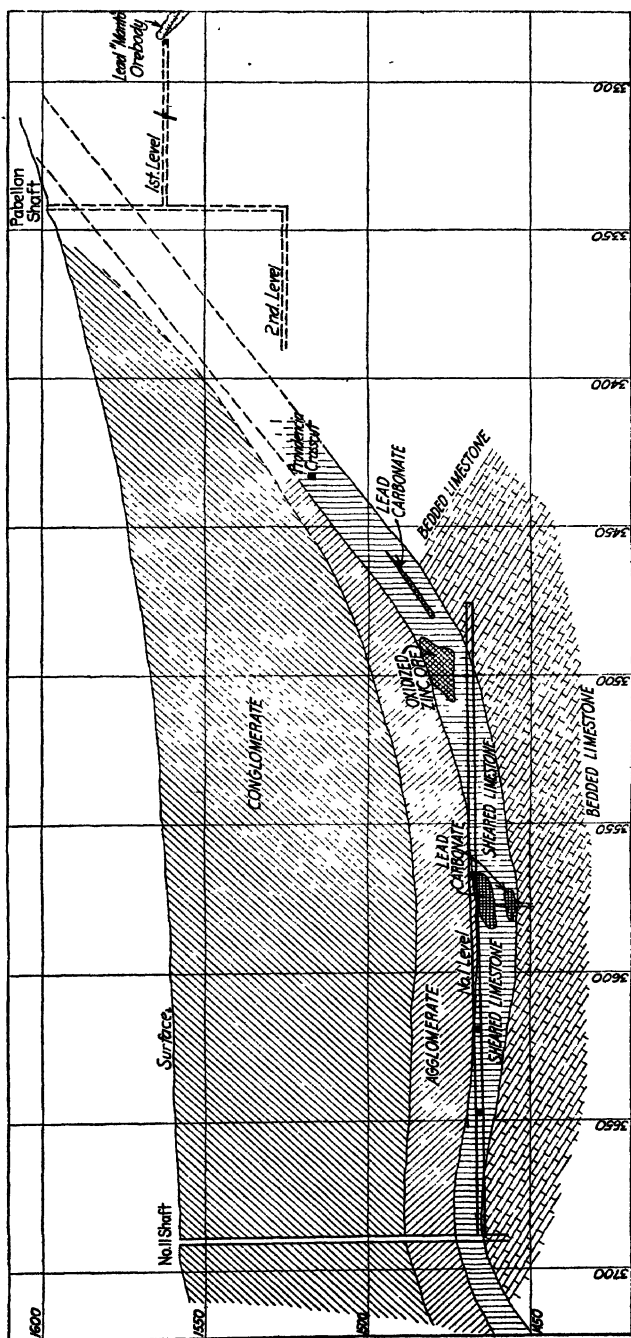
Fig. 3.—Section across zinc stope, Doenea and Providencia mines, Sierra Mojada, Coahuila.

fault is found striking approximately N. 25° W. which can be traced through the "A" (Monterrey) and Exploradora claims for a distance of 350 m.   The orebodies and contact have been displaced for a few meters by this fault.

Still farther east, in the Fortuna mine, a fault has been followed by a drift along a strike of N. 40° E. for a distance of 175 m., and it is believed that a somewhat less prominent fault found in the Veta Rica mine is the extension of the Fortuna fault to the northeast.

In the San Salvador mine, there are two faults less marked than those mentioned, called the San Juan and the Santa Gertrudis.   These faults strike N. 45° E. and N. 20° E., respectively; both of them displace the orebodies for short distances.   A series of local minor displacements occur in the San Salvador mine with strikes from north-south to east-west, which produce displacements in the orebodies of a few inches to a few feet.

The most prominent and important displacement so far observed in the district is the movement of the agglomerate to the north along a usual dip of 15° to 45°.   This movement of the agglomerate over the edges of the bedded limestone has produced a zone of sheared limestone with strike and dip paralleling the plane of the agglomerate contact, and having a thickness of a few inches to 50 or 75 ft. (23 m.) as a maximum.

Geological History

The sequence of events appears to have been about as follows: The stratified limestone was laid down in the sea, elevated to approximately its present position, then folded by horizontal stresses.   An anticline once occupied the present Sierra Mojada district, the erosion of which resulted in the present valley.   This period of erosion and denudation was followed by expulsion of the agglomerate, which was afterward partly, and in places entirely, covered by further erosion from the limestone cliffs and from the upper part of the agglomerate itself; this covering, in time, was cemented into the present conglomerate.   At some period after the expulsion of the agglomerate and the partial covering by debris, there was pronounced faulting resulting in a downward movement of the agglomerate to the north, the fault plane having a general east-west strike and a dip to the north of 15° to 45°.   This faulting involved movement of the conglomerate as well as of the agglomerate, and in many places the hanging wall is seen to be conglomerate with a foot wall of sheared limestone.   The movement of this enormous weight over the edges of the limestone beds produced the sheared limestone that is now seen nearly everywhere between the limestone beds and the agglomerate, or the conglomerate, as the case may be.

Several zones of fracturing resulted either from the expulsion of the agglomerate or from the movement of the superencumbent mass of rock, or both, the principal fractures, and apparently the deepest seated, being

those with a general east-west strike. North-south fractures of limited extent are also seen, but appear to have no relation to the origin of the ore. The metal-bearing solutions rose through the east-west fracture zones, of which three principal ones have already been recognized, and deposited their contents as replacements in the limestone, along fractures, along the contacts of sheared limestone and agglomerate, or within sheared limestone beds as well as in the movement planes within the sheared limestone.

THE ORE DEPOSITS

The ore deposits may be classified as follows:

1. Oxidized argentiferous lead ore, containing considerable iron.
2. Contact orebodies, of the following types: (a) silver-copper ore; (b) siliceous silver ores; (c) siliceous silver-lead ore.
3. Silver impregnations in the limestone.
4. Oxidized zinc ore.

The oxidized argentiferous lead ore is found as a replacement of one or more strata of limestone along a general east-west zone of fracturing, apparently related in no way to the agglomerate-limestone contact. This ore also occurs in lesser amounts as the filling of fractures having a general east-west strike and lying to the north of the main lead orebody, under or close to the sheared limestone-agglomerate contact. The main lead *manto* was the first important orebody mined in the camp, and extraction has continued from the discovery of the camp in 1878 until the present, although since 1905 the tonnage recovered has not been large. Complete records of production are not available. In 1896, the production is stated by Chism to have been 1000 tons per week. In 1893, the American Consul reported that the San Salvador mine, which was the most important, was producing 400 tons per day. Malcolmson states that, in 1900, the production of lead ore was about 120,000 tons per year.

Silver-copper orebodies have been mined from the contact in the A & B (Monterrey), San José, Fortuna, Veta Rica, Deonea, Guadalupe, Parreña, Esmeralda, Trinidad, and San Salvador mines. They occur as a replacement in the sheared limestone, generally at or close to the agglomerate contact. A large body of copper ore was found in the San José claim, which was very siliceous, and appeared to be a replacement of the agglomerate rather than of the sheared limestone. Siliceous silver ore has been mined in the Jesús María, San José, Providencia, and San Salvador mines; this ore has been of much importance. Siliceous silver-lead ore has been mined in nearly all of the principal mines of the district, from the sheared limestone at or close to the contact. A considerable tonnage of this class of ore has been contributed to the production of the camp.

Large quantities of silver-lime impregnations have been mined from

nearly all of the principal mines where the limestone-agglomerate contact is represented. In 1886, this ore had not been discovered. In 1889, Malcolmson mentions the discovery of the San José copper orebody, which was soon followed by discoveries of limy silver ore. In 1900, 25,000 tons of limy ore and 52,000 tons of siliceous limy ore were mined. In 1900, the limy ores were beginning to be the mainstay of the camp, and since that time the relative importance of this class of ore has increased, until at present but little ore of any other class is produced.

The oxidized zinc ores were of no commercial value until 1915, when a considerable quantity was produced from the San Salvador mine; since then, important quantities have been extracted from the Deonea, Esmeralda, San Salvador, and Atalaya mines.

Origin of Ore Deposits

Three series of fracture zones have been recognized, extending in a general east-west direction almost continuously from one end of the camp to the other. They were probably formed partly during the folding of the limestone, partly during the expulsion of the agglomerate, and partly by movement of the agglomerate over the edges of the limestone beds. It seems that subsequent to the movement of the agglomerate the mineralizers that formed the contact deposits rose along the two fracture zones lying to the north, deposited their contents to some extent along the fractures, and on reaching the contact spread through the sheared limestone just under the agglomerate. The agglomerate itself appears to have been impervious to the solutions. In places, the solutions followed upward along planes in the sheared limestone before reaching the contact. These two fracture zones have been designated B and C for the sake of reference. It has not yet been determined whether the lead ore in the main fracture zone, which we have designated as the A zone, ascended previous to the deposition of the ores along zones B and C. At any rate, the quantity of solution rising along the A zone was much greater than that which formed the combined B and C zones. The former carried lead, zinc, and iron sulfides, with silver, but no gold.

North-south and northeast-southwest fractures are seen in many parts of the camp, in all of the zones, but these appear to have had no connection with the genesis of the ores. The lead manto in the eastern portion of the Encantada mine appears to make an offset to the north, which may have been caused by a fault; but, if so, the fault seems to have had no bearing on the origin of the ore. Fractures striking north and northeast are seen in the Encantada, San Salvador, Esmeralda, and Parreña mines, under the lead manto. In the San Salvador and Encantada mines, the intersections of the north-south fractures with the east-west fractures are sometimes the locations of large iron-filled chimneys.

North-south fractures in the San Salvador mine have displaced the mineralized fractures in the *B* zone for short distances.

The Ore Zones

A Zone.—The solutions rising through the fractures along the *A* zone replaced certain beds of limestone for a width of from 30 m. to a maximum, in the San Salvador mine, of 150 m., and for a thickness of 2 to 30 and even 40 m. It extends for an unbroken length of over 4000 m. from east to west along the fracture zone. In the Fortuna mine, the replacements have occurred along three beds following the fracture zone, but in the San José, the San Salvador, and the Encantada mines, along two limestone beds; the question arises as to whether, at depths below those to which mining has yet progressed, other beds may not have been replaced by the rising solutions. On encountering the limestone beds chemically or structurally favorable for the deposition of metals, the solutions worked out both north and south of the fractures, replacing the limestone in both directions. Locally the orebodies are somewhat irregular in shape, leaving "horses" of limestone in the center of a highly mineralized mass. The outlines of the edges are irregular, as seen on the old maps, but this irregularity may have been due to impoverishment in grade or to diminishing thickness, rather than to actual absence of mineralization.

Cross-cuts under the lead *manto* in the Parreña, Esmeralda and San Salvador mines have cut iron-filled fractures that have a strike parallel to the lead orebody, and appear to be the path through which the solutions ascended. No work has been done under the lead orebodies in other mines of the district, wherefore other similar fractures have not been found.

The lead orebody has been worked in an unbroken line from the Jesús María mine in the west through San José, Fortuna, Providencia, Sultana, Volcán Dolores, Parreña, Esmeralda, San Salvador, Encantada and Fronteriza claims, to the Oriental claims, at which point the orebody practically pinches out. The working area in the Fronteriza is much narrower than in the San Salvador, and the eastern limit of mineralized area appears to have been reached. There is a faulted section in the approximate center of the Jesús María mine, to the west of which the lead ore has not been found in the *A* zone, and investigations so far have not determined whether an important mineralized area extended to the west beyond this point.

The deposits in the *A* zone are practically limited to argentiferous lead ore, now entirely oxidized; considerable iron pyrite was present, which aided the oxidation of lead and zinc sulfides. The lead sulfide was oxidized for the most part to lead carbonate which remained, *in situ*, together with considerable iron; while nearly all of the zinc, large quantities of iron, and probably considerable silver migrated to more distant points. Extensive chimneys of irony material, with lead and silver contents decreasing with depth, are found in the Esmeralda and San Salvador mines, presumably at

the intersection of east-west with north-south fractures.   In the Esmeralda mine, the Jarocho chimney gradually gave way to hard limestone, and at the lowest point reached, 140 m. below the lead *manto*, were found two narrow fractures containing oxidized ore assaying up to 40 per cent. zinc. So far as the records go, no silver was found in the lower part of the chimney, and one wonders whether at a still lower point silver-lime impregnations may not be found.   The Porvenir chimney in the San Salvador mine reached a depth of 105 m. below the main lead *manto*, without finding the limit of the irony material; the lead and silver contents had, however, almost disappeared.   On the fourth level, or 80 m. below the *manto*, a northeast fracture extending out from the Porvenir chimney was found to be filled with zinc carbonate assaying about 40 per cent. zinc.   On the third level of the San Salvador mine, a cross-cut was driven for about 30 m. to the south in a body of ferruginous material, which showed an almost total absence of lead and silver.   The southern limit of this iron body had not been reached when work was suspended at that point.

In the Esmeralda mine, there is a body of oxidized zinc ore on the southern edge of the *A* zone, which appears to be a replacement of the same bed of limestone that is replaced by the lead ore farther to the north. The limestone beds here change from their south dip to a slightly northerly dip, on the slope of which the zinc ore was found.   This ore is entirely oxidized, and assays 22 per cent. zinc and 1 per cent. lead, with low silver.

In the Encantada mine is a large body of oxidized zinc ore assaying about 18 to 22 per cent. zinc, which appears to be a replacement of the limestone bed, lying under and parallel with the main lead *manto*.   There is said to be upwards of 1,000,000 tons of this low-grade zinc ore in this mine, which in the western part of the property extends into the San Salvador and Atalaya mines.

In the *A* zone, oxidized lead ores, mixed with considerable free sulfur, have been found at various points.   There now remains a small body of this ore in the Esmeralda mine which assays approximately 350 gm. silver, 15 per cent. lead, and 50 per cent. sulfur.   Attempts have been made to sublime the sulfur with varying degrees of success.

B Zone.—The *B* zone lies to the north of the *A* zone, paralleling it from east to west; in the Encantada and San Salvador mines, the mineralization along its southern edge almost reaches the northern limit of the *A* zone.   The orebodies in this zone are found along the contacts or in the sheared limestone dipping to the north.   Lying in places immediately under the contact, and in other places at a considerable distance below the contact, are found silver impregnations in the limestone.   As a rule siliceous silver-lead ore is found along the contact, or along planes in the sheared limestone paralleling the contact, while beneath the siliceous lead ore are found the silver impregnations either in the sheared limestone, or in the bedded limestone under it.   In the San Salvador mine, the Hallazgo silver-lime stope extends to a depth of 50 m. below the

contact; as a rule, however, the silver-lime seldom extends deeper than 25 or 30 m. below the contact. In the Esmeralda and Volcán Dolores mines, the orebodies that possibly existed in the B zone close to the A zone have been partly eroded, leaving only the northern portion of the ore, which is therefore separated 40 to 60 m. from the A zone orebodies. Farther to the west in the Veta Rica, Fortuna, San José, and Jesús María mines, the northern edge of the A zone and southern edge of the B zone are very close; and in places the B zone overlaps the A zone. No mining has been done in the B zone in the Sultana and Providencia mines. The ore in the Sultana may possibly have been eroded but in the Providencia mine it would appear likely that ore should be discovered by properly directed prospecting.

Practically all of the ore in the B zone is oxidized. In the Encantada mine near the San Salvador line, a small amount of primary sulfide is found. Lead-zinc sulfide is also found in small patches in the Veta Rica above the fourth level, while in the Jesús María mine a fracture is seen filled with primary iron-lead-zinc sulfide which has a thickness of about 0.5 m. and has been followed for 50 m. in length. These sulfide orebodies are always small and seldom of economic importance. In the Veta Rica and San Salvador mines, there are extensive areas containing fractures filled with barite, accompanied in some places by sphalerite, but without any appreciable amount of iron. This ore yielded less readily to the oxidizing influences than the sulfides containing much iron.

In the Deonea mine is a considerable body of oxidized zinc ore, assaying 30 to 35 per cent. zinc, with low silver and lead contents. This ore occurs on the south slope near the low point in the trough at, or below, the contact, and has been mined to a considerable extent during the past two years. Calcination raises the grade to approximately 40 per cent. zinc.

There are two fractures with northeast strike in the I-mine through which solutions reached the contact and deposited their contents. The fractures themselves have been worked for lead and zinc ores, both of which are oxidized, and in places are found mingled to such an extent as to require admixture of clean lead ore to reduce the smelter penalty on zinc. The fracture in the Jesús María mine, filled in one section with primary sulfide, is probably one of the sources of mineralization along the B zone in this mine. This fracture extends eastward through the San José mine, where it is known as the San José fissure, and through the Fortuna into the Veta Rica mine where the Camilo Orozco y Maria stopes were worked. The Providencia to the east has not been sufficiently explored to determine whether this fracture continues eastward beyond the Veta Rica mine. The Washington fracture in the Veta Rica mine, extending westerly into the San José, and easterly into the Deonea mine, is probably the source of part of the mineralization along the B zone in the San José, Fortuna and Deonea mines, while the southernmost portion of the zone in these mines has been filled from the San Vicente and the Orozco-San Juan

fractures.   The San Vicente fracture extends east into the Veta Rica and west into the San·José mine; in the Fortuna and Veta Rica mines it contains large lead orebodies along its walls.   It is clearly seen to extend up to the agglomerate contact, which it follows to the surface, though it does not produce ores at the surface.

Fractures of the *B* zone in the Esmeralda mine are not clearly defined in connection with the contact orebodies.   In the San Salvador mine, however, the Santa Gertrudis and the Old Find fractures are distinct. The Santa Gertrudis fracture can be followed underground by stopes and exploratory workings across almost the full length of the San Salvador mine.   This fracture is again seen in the western part of the Encantada mine, 125 m. below the surface, where small patches of primary sulfides are in evidence.

Practically all of the fractures in the *B* zone dip to the north at angles of 30° to 60°, and in some cases become almost vertical.   As a rule, they pass through the sheared limestone up to the agglomerate contact where they flatten to conform to the contact dip.

Along the *B* zone there is a fracture in the agglomerate in the Veta Rica mine, from which some argentiferous lead ore has been mined, but the quantity and grade were too low to make this of any importance. The first workings in this mine are said to have been driven on this agglomerate fracture.

The *B* zone explorations have been carried to the east only a short distance in the Encantada mine.   It remains to be seen whether the eastern portion of the Encantada mine, as well as the Fronteriza mine farther east, will yield ore along the contact as far east as the lead ore *manto*, or *A* zone, that is, to the Oriental claim.   At least this area appears worthy of being prospected.   Work on the *B* zone has been carried to the west through about one-half the length of the Exploradora claim, and ore has been extracted for this distance.   Farther west there is an area practically unprospected until the I-mine is reached, where important limy silver orebodies have been worked.

C Zone.—This lies about 125 m. to the north of the *B* zone in the San Salvador and Esmeralda mines.   It is represented by the North San Salvador and the Trinidad channels in the San Salvador mines, and by the Norte Oriente stopes, a continuation of the North San Salvador channel westward into the Esmeralda mine.   In the Deonea, Veta Rica, and San José mines, this zone is wider than in the San Salvador, and to the south reaches practically to the *B* zone.   The relations in the Deonea and Veta Rica mines are such that it is difficult to distinguish definitely between the two zones, as they appear to be connected by diagonal fractures.

The orebodies of the *C* zone, as in all of the contact deposits, while locally irregular, are quite persistent over the zone as a whole.   The outlines of the stopes are very crooked, because of the irregular manner in

which secondary deposition of silver, and in some cases copper, has taken place in the sheared limestone and in the bedded limestone below the original contact orebodies. The outside limits of the stopes do not necessarily represent the absence of mineralization.

No work has been done on this zone in the Encantada or Fronteriza claims to the eastward, but as the zone is so persistent through the central portion of the district, paralleling in a general way the A and B zones, there would appear to be good prospective value in the San Salvador, Encantada, and Fronteriza claims approximately as far eastward as the A zone has been worked. The C zone has been worked in the Nuevo Almadén and Deonea claims, but there is an area of unprospected ground in the western part of the Nuevo Almadén claim, and a large area in the eastern section of the Deonea claim which appears to be very inviting. The Eleno, Angel and Juárez stopes along this zone in the western part of the Deonea claim were very productive.

The Jacobino-Liberales, San Francisco, Novios, Alvarez and other stopes along this zone in the Veta Rica mine produced large quantities of rich ore. The San Francisco and Novios stopes extend westward into the Guadalupe mine. There is an area of either barren or insufficiently explored ground in the western part of the Guadalupe and Blanca claims, where no ore of consequence has been extracted. Farther to the west the extension of the C zone is found in the A and B claims of the Monterrey mines, beyond which point explorations have not been carried.

A considerable body of oxidized zinc ore has been extracted from the Rubén stope of the Deonea mine along the line of the Eleno fracture in the C zone. This ore assayed about 30 per cent. zinc as mined, and was calcined to 38 or 40 per cent. zinc for shipment. This appears to be a body of ore which has migrated from the original deposition of lead-zinc sulfides higher in the Eleno stope. A fracture is found in the San José mine which appears to have mineralized the stopes along the C zone. Well-defined fractures have not as yet been found in the Guadalupe claim, along the C zone. In the Veta Rica mine the Jacobino-Liberales fracture has been followed on the sixth level to the east under the mineralized zone, and is found to enter the Deonea mine, where it is known as the Eleno fracture. This fracture occurs at the lowest point of an east-west trough in the contact in the Veta Rica mine, where on the sixth level it is well defined. The lower part seen here is filled with oxidized iron and lead minerals, containing little silver. There appears to be a possibility of discovering secondary silver ore below the fracture at this point. Along the same zone in the Deonea mine may be seen the Angel and Victoria, or Juárez, fractures, which have produced large and important orebodies. All of these fractures have been lost in the Deonea mine at about the center of the property, and little work has been done in the eastern section to relocate them.

Farther east, in the Nuevo Almadén mine, two fractures are seen, the southernmost of which dips about 45° south, practically following the bedding planes of the limestone, and filled with lead ore. This fracture has not been followed upward as far as the sheared limestone or the contact; therefore, we have no knowledge of the deposits along that horizon. This fracture can be followed into the Trinidad claim where large stopes of lead and copper ore were extracted. The northerly fracture in the Nuevo Almadén has produced both lead ore and limy ore; and it appears to connect with the Trinidad stopes, although it has not been followed to the east the entire distance.

A fracture is found in the Norte Oriente orebody, and the mineralization continues to the west into the Nuevo Almadén mine, though it is not very important, and to the east into the large lead and lime orebodies along the North San Salvador channel. At the interior shaft in the San Salvador mine, there is a fracture in the North San Salvador channel, which dips to the south and is filled with oxidized lead ore. In one or two northeast-southwest fractures crossing this channel a small tonnage of ore assaying approximately 40 per cent. zinc has been mined. There is a south-dipping fracture in the eastern part of the San Salvador mine, reached from the east drift of the No. 1 shaft, below which some oxidized silver-lead ore has been mined. This fracture is in the supposed prolongation of the Trinidad channel to the east, and it would be interesting to know whether orebodies will be encountered near the contact where the fracture is followed.

Small amounts of ore have been mined from the points north of the generally accepted limits of the C zone, in the Vulcano, No. 10, and Trinidad mines. The exploration work to the north of the C zone has been of a negative character, but the small orebodies found indicate that this area is not entirely barren of mineralization.

The fracturing along the B zone follows closely the variations of strike in the A zone, and where prospecting has been done along the B zone, contact orebodies and lime impregnations have been discovered. The ground along this zone as far as the east-west limits of the A zone, where not already worked, would appear to have good prospective value. The same observations may be made as to the C zone, which parallels approximately the A and B zones, and has produced ore in nearly all places where it has been well prospected. The unprospected ground along this zone would therefore appear to justify exploration.

Character of Ore

The ore mined from the lead *manto* in the first 10 years of production was undoubtedly selected, and probably assayed approximately as shown in the accompanying table. The character had changed somewhat

at the time of Malcolmson's paper,[2] the average assay for 1900 being given in the table.

	Silver, Grams	Lead, Per Cent.	Iron, Per Cent.	CaO Per Cent.	Insoluble, Per Cent.	Copper, Per Cent.
First 10 years..........	600	25	35	5	15	
1900 average...........	290	15	36	5	16	
Lime ore, average......	800	1	3	35	10	
Copper ore, average	1000	1	8	18	25	3

PRODUCTION

Records of production have not been carefully kept, especially during the early years. In 1893, the American Consul stated that up to Oct. 1, 1892, the production had amounted to approximately 500,000 tons of ore (probably almost entirely lead ore) and 85,000 tons of lead bullion had been smelted in the district. Malcolmson states that between 1890 and 1900 the tonnage averaged approximately 200,000 tons per year, and gives an estimate of 3,000,000 tons as the production between 1886 and 1900. The total production of lead ore to the end of 1920 has probably been between 3,000,000 and 3,500,000 tons, and of copper and silver-lime ore, 1,500,000 to 2,000,000 tons, the combined value of which totaled between $50,000,000 and $75,000,000 (U. S. Cy.).

PROSPECTING

The lead ore in the *A* zone was comparatively easy to follow, and no special ability was required to discover the orebodies and extract them as far as profitable to do so. The contact ores have been less easy to follow, owing to their comparative irregularity, but by following the contact or the planes in the sheared limestone within the areas of the zone limits, the prospecting work can be reduced to a minimum. For many years this work was carried out in a most haphazard fashion, owing to the lack of knowledge regarding the occurrence of the ore, but it can now be done with greater intelligence. The limy silver ores were also found by prospecting in a hit-or-miss fashion, but are now prospected by confining work within the limits of the zone areas, working within the sheared limestone or in the bedded limestone for a short distance below the former. For the most part, these lime impregnations have seldom been found to extend more than 25 m. below the contact, or below the sheared limestone; wherefore prospecting should be confined to a block lying horizontally within the zone limits, and vertically within 25 m. of

the contact. Lime stopes in a few cases have been extended 50 m. or more below the contact, but the upper portions will nearly always be found within 25 m. of the contact.

Diamond drilling is not suitable for prospecting the lead fractures because of loss of water and caving of holes. Within the limestone, where lime impregnations are being searched for, there is no reason why the entire area being prospected cannot be diamond drilled, after drifts have been extended lengthwise of the zones. In the I-mine, several such holes were drilled in 1910, whereby at least one orebody was discovered.

FUTURE OF DISTRICT

Observations at the present time lead to the conclusion that the future of the district lies in exploiting the contact ore zones, together with underlying silver-lime impregnations. It may also become profitable to mine the low-grade oxidized zinc ore, of which a large tonnage is now available in the Encantada and San Salvador mines, but this can be done profitably only after some means of concentrating the ore locally has been developed. There are considerable areas in the western part of the camp along the B and C zones that have been imperfectly worked, also in the Providencia, Deonea, Nuevo Almadén, Esmeralda, San Salvador, Encantada and Fronteriza mines, in which it is reasonable to expect to find considerable tonnages of siliceous-lead and limy silver ores, and also possibly silver-copper ores. There is considerable tonnage of low-grade limy ore in the Esmeralda and San Salvador mines which it might pay to extract if mined in large quantity, at low cost. Tests have shown that some of this ore is amenable to cyanidation.

Prospecting north of the C zone, both underground and by diamond drills, has given negative results. It is not conclusive, however, that other orebodies will not be found north of any orebodies so far exploited.

The extensive lead *manto*, once the principal mainstay of the camp, is practically exhausted. There appears to be no geological reason why another bed of limestone favorable for deposition of lead ore may not be found below the present worked bed, and diamond drilling could well be employed to determine this point. The migration of the silver from the extensive lead *manto* to lower points could well be investigated. The lead-bearing orebody was so extensive that very large quantities of silver may have been carried off, and if deposited in a concentrated form would generate large and extensive ore deposits. No exploration work of consequence has been done to the south of the main lead *manto*, although there seems to be no reason to preclude the possibility of another fracture zone south of the A zone.

Some lead and zinc ore has been mined from local fractures in the Atalaya claim, south of the generally accepted ore limits, which leads to interesting speculation as to the existence of orebodies at some depth

below the present surface, or possibly to the former existence of orebodies now eroded.

DISCUSSION

W. W. SHELBY, Esmeralda, Coah., Mexico (written discussion).— Some more recent work done in the virgin ground of eastern Deonea mine has disclosed a rather unique orebody. This work consists of the prospecting of the principal east-west fracture of the C zone. The fracture has a well-defined hanging wall, but no definite foot wall. The C zone fractures generally form silver-lime orebodies of considerable size. They are found from the contact to about 25 m. below, but more often do not extend more than 10 m. below.

In the 190 m. of drift that has been run on this fracture, a practically continuous orebody has been found. This ore is a silver-bearing lead carbonate with more or less iron. As the iron increases the lead and silver decrease in value. It occurs as a nearly horizontal oreshoot, paralleling the east-west dip of the agglomerate contact, but 30 m. below the contact. In the several raises that have developed the fracture above this oreshoot, no lime ore has been found. There is no apparent reason why the lead ore was deposited at this distance below the contact with barren lime above it. In places there are chimneys of a low-grade irony lead extending down from the ore horizon, but they are too low grade to be mined. These chimneys carry about 45 per cent. iron.

The unimportance of the post-mineral faults is again shown. A north-south fault displaces the fracture only 2 meters.

BASIL PRESCOTT, El Paso, Texas (written discussion).—This paper summarizes the situation in that camp in a clear concise manner. The division of the mineralized area into three zones, A, B, and C, while somewhat arbitrary, is useful and will undoubtedly become part of the terminology of the district. The conclusion that thorough and systematic prospecting, confined to the zones indicated, will yield a maximum result is undoubtedly correct, though the zones might be subdivided in many instances and the area warranting intensive prospecting reduced to definite parts of the zones along certain definite fissures.

Other factors, not especially emphasized in this paper, enter into the localization of the ore deposits. Throughout the central part of the district, the trace of the limestone bedding planes on the east-west fissures is approximately parallel to the contact, so that the preferential action of certain beds over others is not particularly obvious; but toward the eastern end of the district, it becomes apparent that the position of the orebodies on the fissures is controlled by the character of the sedimentaries. This applies only to the lead deposits of the A zone as neither B nor the C, as the author points out, have been followed to the extreme eastern end of the camp.

The position of an orebody in space must be defined in three directions. The position along a north-south coördinate is controlled, at Sierra Mojada, by those east-west fissures that the author has grouped in zones. The position along the strike is controlled by those obscure and sometimes undeterminable factors that govern the localization of oreshoots along fissures and veins. The vertical component is controlled chiefly by the temperature and pressure at the time of deposition and, to a minor extent, within the limits of favorable physical factors, by the character of the country rock. It is with regard to this third component, particularly to the range of physical conditions and the controlling factors, that Sierra Mojada presents features of interest to the engineer and economic geologist.

The lower, or northern, limit of the *C* zone marks the approximate line of intersection between the top of an underlying limestone and shale series and the agglomerate. This underlying formation, where exposed in the mine workings, is composed of limestone that would not be judged unfavorable for replacement by mineralizing solutions, with exceptionally heavy selvages or shale layers along the bedding planes. These shale layers, in some instances, have an exposed thickness of over 1 m., though in general the series is composed of thinly bedded limestones, alternating with heavy selvages. These beds, through the west central part of the camp, probably lie with the average dip slightly to the south, though at many points they are rather intricately faulted and are contorted into sharply compressed folds which, while maintaining generally an east-west strike, yield any desired dip. This change in the character of the country rock was not beneficial to mineralization and the contact, below the point where it is first found, is tight and, as a rule, economically so uninteresting that little investigating has been done in the area. Probably it is this changed country rock that controlled the limits of mineralization to the north, rather than that the northern edge of the zone of east-west fissuring has been reached, for in several places fissures apparently as well developed as those that are orebearing farther up the contact can be found north of the economic ore.

There is a slightly developed tendency toward the same shaly lime beds in the San Francisco cañon nearby at a higher horizon in the mine workings; and there is an outcrop in which single layers of the shales reach a thickness of 20 ft. Probably this formation indicates a transition with depth to a shale series, but there is no data as to the thickness of the transitional beds nor certainty of the existence of the shales below. It is said that diamond-drill holes showed shales below the limestone and also a distinct underlying formation; but as there are no records of that work no reliance can be placed on such reports. On the other hand, while relatively few exposures of the shaly limestone series are to be found in the immediate district and there is a dearth of knowledge of the conditions

below, such transitions with depth are not unexpected on the Mexican plateau and are often attended by unfavorable effects on the ore deposits.

The occurrence of the ore above the agglomerate and under the lime breccia that caps La Blanca Hill is noteworthy, as it lies at least 200 m. north of the northern limits of the C zone. La Blanca Hill is a remnant of a previous erosion surface composed of agglomerate up to within a few meters of its top, capped by a limestone breccia derived from the main sierra to the south, with an ore deposit, said to have been the original discovery of the camp, between the agglomerate and the overlying reworked limestone. This ore apparently lies along the same fissuring that yielded a small amount of ore in the lowermost workings of the volcano, along the contact between the limestone and the overlying agglomerate, about 1000 ft. (305 m.) below the top of La Blanca Hill. In this case, it seems apparent that the release of temperature and pressure was not sufficient at the lower contact between the tight lime and shale series and the agglomerate to cause heavy deposition, nor was this accomplished in the agglomerate itself, the mineralization apparently staying in solution until the upper contact was reached. The possibility of the mineralizer passing along a fissure in the agglomerate, about 1500 ft. south of this point and east of the Veta Rica claim, on a higher part of the contact but at about the same approximate elevation as the La Blanca occurrence, is supported by the fact that an east-west fissure was mineralized in the agglomerate for at least 100 ft. above the contact and the ore outcrops on the surface in an arroyo. This occurrence seldom reached economic width and grade but attempts to stope it at various points have been made.

Along the lower edge of the C zone, the east-west fissures reaching the contact between the shale-lime beds and the agglomerate, which are cut and faulted by cross fractures extending into the agglomerate, appear to have been mineralized farther to the north than those not affected by cross breaks; and it seems probable that the lower edge of the C zone, when worked out, will be more irregular than is now apparent.

The southern limit of the A zone has already shown some irregularity and it cannot be said that the non-existence of ore-bearing fissures in the south has been established. Toward the east, there is evidence that an upper ore-bearing horizon that lay south of the present limits of the A zone has been eroded; in the Fortuna ground, the indications on upper horizons to the extreme south are still favorable; and well into the heart of the sierra, east-west fissures outcrop on the surface though no mineral has been discovered along them. It would seem, therefore, that while the three zones established group in a concise manner the present known ore deposits, it would be inadvisable to assume that these necessarily mark the extreme limits of the ore-bearing ground.

As the paper states, the mineralizing solutions rose along the east-west fissures and deposited on the contact and in the limestone usually only a short distance below it. This can be established beyond question at a number of widely scattered points. The fissures of other courses, often very prominent, are seldom of interest and then only as faults or channels of secondary action; the supposition that the mineral was introduced upward along the contact from some point farther north under the valley has little to recommend and less to support it, and development carried out with this hypothesis as a basis has yielded poor results. In many instances, however, as the east-west fissures approach the sheared limestone and the contact, the solutions followed this zone for some distance, depositing in bodies parallel to the contact.

In most limestone deposits, many fissures are found, often with a most marked effect on the distribution of the ore, which show no movement or dislocation of their walls; in fact, in many instances, the absence of these features is a characteristic of the premineral fissures. At Sierra Mojada, the important premineral fissures seem to show at least minor displacement and evidence of movement, and in this particular the district differs from most limestone camps of Mexico.

From the orebodies found along the contact, from La Blanca occurrence, and from the ore deposit in the fissure in the agglomerate, it is apparent that the volcanic agglomerate antedates the mineralization; and a study of the surface readily establishes that the topography and the surface at the time of deposition could not have been far different from that now existing. It follows that the ore deposition took place at an extremely shallow depth below the surface; this is one of the unique features of the ore deposits of Sierra Mojada, and one in which it differs from all other well-known limestone deposits in Mexico. The valley between the Sierra Mojada and the Sierra Planchada has suffered some erosion since mineralization took place, but it is doubtful whether the surface at that time was far above the top of La Blanca Hill. No strong faulting is now known parallel to the orebodies and the range that could seriously confuse or vitiate this conception, so we are forced to conclude that the deposition took place, in many instances, at a depth not greater than 1000 to 1500 ft. and may have taken place, in certain instances, at a shallower depth.

The total vertical range of mineralization, as exposed, is extremely small for such extensive orebodies. From east to west, there is a considerable difference in elevation, possibly 1500 ft., the eastern end being the lower; but on any one north-south section, the total range will probably not exceed 800 ft.; and at any single point, the range of economic primary ore will probably not exceed 300 ft., with the usual maximum more nearly one-half that amount. This condition is what would be expected, if the conclusion regarding the position of the surface at the

time of mineralization be correct. In this, Sierra Mojada is again unique among the limestone deposits of Mexico, many of which have a proved vertical extent of 2000 or even 3000 ft. without great change in size or character of the ore.

Within this limited vertical range, there were deposited three distinct types of primary mineralization:

1. A silver-lead-zinc-iron sulfide, usually with a low silica gangue. This is now almost completely oxidized, but occasional specimens and the character of the derived oxidation products give ready means of reconstructing the primary ore. It occurs chiefly in typical replacement deposits in limestone, to a minor extent in the contact and in fissures in the agglomerate 50 or 100 ft. from contact.

2. A silver-copper-zinc sulfide with silica and barite gangue, occasionally with some lead and sometimes practically without zinc. Small tonnages of these ores are found in a semi-oxidized condition, chiefly along the contact and short distances below it.

3. A complex silver sulfide accompanied by minor amounts of copper and sometimes traces of cobalt with a barite gangue and sometimes calcite and rhodochrosite. In recent years, considerable of these ores have been mined in an unoxidized state. They are found along the contact in the sheared lime, but chiefly along the fissures to a depth of 100 ft. below the contact, and are accompanied by impregnations in the wall rock that are responsible for much of the so-called secondary silver-lime ores. The distribution of these ores with relation to the fissures is quite irregular and generalities are dangerous.

Most of the silver-lead-zinc-iron ores were deposited along the fissures designated as the A zone; in which neither of the other types is prevalent. In the B and C zones, all three types are encountered; and it is not uncommon to find along a single fissure successive stopes representative of the distinct mineralization.

As would be expected, the demarcations between the three types are not always sharp, particularly between the silver-copper ores when the zinc is absent or low and the gangue heavy in barite, and the silver-barite ores when they carry some copper. On the other hand, intermingling is not usual, nor is a complete series of intergradations developed. The mineralization appears to have occurred at different periods and the mineralizer to have varied in composition, perhaps progressively. No evidence can be obtained in the present oxidized condition of the orebodies as to the sequence of the mineralization, though it seems probable that the silver sulfide, with some copper and the barite gangue, was the last to be formed.

We have then in this extremely limited vertical range, three mineralizations that normally would require quite different physical conditions for their deposition; and in this, Sierra Mojada is unusual among Mexican

replacement deposits, though all four of the distinct features here empha-
sized—the absence of premineral fissures unaccompanied by faulting,
the shallowness at which deposition took place, the limited vertical range
of the deposits, and the complexity and variation in the ores—are all
dependent on one another; and after establishing any of these features,
the others would not be entirely unexpected.

From the foregoing, it would seem that the drop in the physical con-
ditions required to cause deposition took place above a shale-lime series,
the argillaceous beds acting as a blanket to retain it, perhaps abnormally,
below that point. The release took place, in many instances, on or near
the contact or in the sheared lime just under the contact; and in this
upper horizon of the camp, apparently the general physical condition
being favorable, selective action of the different beds was possible and
even quite prevalent.

If the conclusions as to the effect of the underlying formation on the
deposition and as to the typography at the time of deposition are correct
and accepted, the known deposits of the camp, of economic importance,
may be said to occur in a triangular block between the shaly lime series,
the agglomerate contact, and a southern boundary that probably will be
north of the high cliffs of the Sierra Mojada range.

As the author has pointed out, the principal production in recent
years has come from bodies of limestone impregnated with silver. It
has been assumed that these ores were the results of migration, during
oxidation, of the silver originally contained in the primary sulfide bodies,
the silver being redeposited in the limestone along the channels of the
percolating waters; but occurrences that justify such an hypothesis are
not numerous nor have they yielded great tonnages.

The quantity of such limestone ores found in the *A* zone below the
silver-lead carbonates has been negligible and there are no indications
that appreciable quantities of such ores exist. The question is often
asked "where is the silver that (it is presumed) was leached from the
orebody during oxidation and migrated with the circulating waters?"
In all probability, no appreciable amount of silver was leached from the
lead *manto* of the *A* zone.

Limestones, so impregnated with secondary silver minerals leached
from primary orebodies during oxidation as to form important commercial
deposits are rare in any district, and require most unusual circumstances
for their formation; where the original orebody carried a high lead con-
tent, they are practically unknown. The question involves the behavior
of many compounds found in the primary ore during the various stages
of the process resulting in the bodies of carbonates and oxides that are
later mined. While these processes are quite complex and variable, the
final results recorded in many different occurrences are fairly uniform.

During the oxidation of a silver-lead-zinc-iron sulfide, the zinc and

iron sulfides are the first to break down, the process perhaps being started by the iron pyrite. The zinc goes into solution and does not redeposit as a sulfide, except perhaps as a mineralogic curiosity, and is carried (depending on the volume of solution, the position of the ground water, and the character of the limestone) from a few feet to several hundred feet. When redeposited, it occurs either as a carbonate or a silicate, often quite pure though sometimes accompanied by considerable iron oxide, usually hematite. The silver content of such zinc ores is only a fraction of that contained in the oxidized body that was its source. The iron, often the most abundant metal, remains chiefly at the site of the original sulfide, but large quantities are leached downward along the oxidizing channels and occur, usually in diminishing quantities as depth is attained, for some distance below the orebody. In some instances, the iron reaches continuously to the point at which the oxidized zinc has deposited. The lead typically oxidizes in place and its migrations seldom extend far beyond the limits of the original orebody, even though the channels may appear to be most suitable for such a result. In general, without taking into consideration the minor chemical processes that may take place or the different lead compounds that may be formed, the original galena reaches its final state of a carbonate in one of two ways: the lead taken in solution on the oxidation of the galena may react directly with the limestone to form a carbonate, or it may reprecipitate as secondary lead sulfide which later breaks down through the sulfate to the carbonate. Examples of commercial orebodies that appear to have been formed by the precipitation of anglesite without the intermediate steps of the secondary galena are not unknown, but are quite rare.

As a result of these processes, in a typical *manto* orebody there are the lead carbonates accumulated in the original site of the sulfide deposition, somewhat enriched by the subtraction of the zinc, sulfur and some iron; below, toward the bottom of the original orebody, a heavy accumulation of iron extends along open channels into the underlying limestone; and at varying depths below this latter, there are deposits of secondary zinc carbonates or silicates. The silver follows both the lead and iron. When the carbonate of lead is formed by direct reaction without intermediate steps through the secondary lead sulfide, it is usually low in silver; but the secondary galena, the anglesite, and, to a less degree, the carbonate derived from it have a much higher silver content. The underlying iron in the typical case carries a normal silver content for some distance below the point at which lead is an important factor, but in most cases the silver content becomes negligible with depth long before the iron gives out.

These reactions and conditions have been observed in so many points and under such widely varying conditions as to the grade of ore, ratio of metals, character of the country rock, and form of the deposit, that they

may be accepted as standard; most unique conditions would be required to cause any great variations. There seems to be no tendency shown at any stage for the silver compounds to separate and migrate independently as does the zinc; and both iron and (when present) manganese travel farther and penetrate farther than the silver.

At Sierra Mojada, large open fissures beneath the lead *manto* carry the iron and zinc leachings from the orebodies above, the silver decreasing rapidly long before the iron diminishes appreciably; the occurrence in no way differs from those observed elsewhere in such oxidized replacement deposits. The lime in places is discolored to pink and red with black blotches, from the effect of iron and manganese solutions, but no appreciable silver results are obtained from the most thorough and painstaking sampling. In this connection, no mention need be made of the pipes hanging below the orebody, for they are clearly of different origin and nature.

Just above the sixth level of the Veta Rica mine, the contact that has been followed from the surface on its normal dip slowly flattens out and, after practically reaching the sixth level, rises gradually to the north to a minimum of perhaps 10 m. before continuing its normal dip. Along this basin are found several strong east-west fissures and along their intersection with the sheared lime, or the contact, and spreading nearly over the entire basin were found rich productive stopes from which several million dollars of ores have been taken. The ores are oxidized silver-copper-iron sulfides with very little zinc or lead, and a meter of the agglomerate, and perhaps 2 or 3 m. of the foot-wall limestone were mineable. The situation is almost ideal for secondary limestone impregnations, the long slopes ending in the basin, the high-grade ore without lead and the strong fissures cutting the orebodies; but although crosscuts pass under these bodies, at various points from 2 to 30 m. below them, and drifts follow the fissures and several diamond drill-holes cross the area, no silver-lime ores were found below or beyond the limits of the primary orebodies.

A long list of limestone deposits, covering nearly every conceivable variation in metal content, form of orebody, and character of the country rock, could be given—all of which are oxidized but none of which produce these limestone ores. In all these deposits, moreover, included limestone fragments or blocks are found in the orebodies that have not been impregnated with secondary silver to form commercial ore. This fact is sufficient proof that such lime ores are not a normal result of the oxidation of a silver ore in a limestone replacement deposit.

At Santa Eulalia, a large tonnage of limestone impregnated with silver has been mined, but practically all has come from a limited part of a single mine. It is all on the border in contact with the oxidized replacement ores; none has been produced from the vicinity of the high-lead bodies,

except in one case where such a body was intersected by a silver-bearing vein-like fissure of a different age than the main body. The only economically important tonnage of such limestone ores occurred under unusual conditions, allowing a possible doubt as to the correctness of the assumption of a secondary origin. One of the most continuous, and probably the richest, *manto* of the camp produced only a few tons of such limestone ores although a most careful search was made for them, and the mine was sampled freely. That body averaged less than 5 per cent. lead, often running about 1 per cent. to the kilogram of silver; but even under these favorable conditions, the silver remained almost entirely within the walls of the original orebody, principally with the iron, though at certain points it is impregnated in masses of gypsum that were included in the orebody.

It would appear, therefore, that Santa Eulalia, one of the few districts that produces limestone ores, supports the conception that impregnations of limestone by secondary silver-bearing solutions to form commercial orebodies are rare.

The largest body of lime ore being worked in northern Mexico occurs in the Descubridora mine near Chihuahua City. This is a gold-silver impregnation independent of and many miles removed from any known replacement deposit, so that there can be no doubt as to the primary origin of the body, though in its present oxidized state it exhibits practically all and only those characteristics of the so-called secondary silver-lime orebodies.

At Sierra Mojada, in the B and C zones, silver-lime ores are being mined from bodies occurring in the limestone well removed from both the contact and the replacement ores, and at points where it is exceedingly difficult to account for the source from which or the channels through which secondary solutions must have traveled, if the bodies were the result of such action; also it is necessary to conceive that these secondary solutions carried their burden of silver through favorable limestone without dropping their content en route.

The greater part of all such ore is now oxidized, but at several large stopes much of the ore is still unoxidized and unaltered. Special attention should be given observations made at such points, for the difficulty of distinguishing ore deposited from an ascending primary solution from that deposited from a descending secondary solution is increased by oxidation and by the minor readjustments and movements of the metallic values that take place during that process. The typical unoxidized silver-lime impregnation at Sierra Mojada is cut by east-west barite stringers frozen tight in the limestone, the stringers showing in different. specimens, massive chalcocite, argentite, proustite, pyrargyrite, stephanite, and sometimes bornite and pyrite, and sulfides intricately mixed with the barite. It is not intended to convey the impression that such

barite stringers, which usually measure from 2 to 8 in. or even 1 ft. in width, are always heavy sulfides, but rather that, without a special effort, specimens that show a considerable proportion of sulfide can be obtained. The limestone is usually impregnated with silver sulfides for a distance not often exceeding 10 to 15 ft. from some such veinlet, the entire mass being of sufficient value to pay heavy mining and treatment costs.

These ores may occur for a vertical distance of from 20 to 150 ft. and very continuously for several hundred feet along the strike of the fissures; but irregularities in the walls are noticeable and the grade varies sharply while the limits of the stope are usually marked by the approach to slips and cross fractures which, were the ores the result of secondary action, would be the very channels for the introduction of the metal content.

In many instances, the barite occurs in flat thin beds of limited extent and even in apparently isolated patches throughout the pay line, while in one stope on the northern edge of the C zone in the Veta Rica mine where this type of silver mineralization occurs in the lime-shale series, the beds of barite attain a maximum thickness of several meters, with the same silver sulfide scattered through the mass.

When unoxidized, the impregnated limestones present no marked difference, under casual inspection, from the ordinary limestones; and even when oxidized they are not conspicuous. It is a well-established fact that the limestones which are most heavily stained with iron and manganese and most shattered by small fractures along which are found copper stains, all criteria of secondary action, are seldom of a grade that will pay for extraction.

There can be no doubt that the silver sulfides found in the stringers and blankets were deposited from the solution that brought in the barite; and it is not conceivable that the silver sulfide found impregnating the limestone adjacent to these fissures should have a different origin than the same sulfide found in the fissures. Therefore, in these particular cases, the question may be attacked from the standpoint of the origin of the barite. This barite in no way differs in its physical characteristics from that accompanying the admittedly primary ores along the contact nearby, and there is no evidence nor reason to believe that the conditions existing at the time of the oxidation of the orebodies were such as to permit the solution and migration of such a relatively stable mineral as barite. Certainly the tightly frozen veinlets in the limestone unaccompanied by other oxidation products do not point toward the secondary origin for the barite.

The following conclusions have, therefore, been reached:

1. Silver-lime orebodies of economic importance, formed by the impregnation of the limestone with silver leached from replacement deposits, are a rare occurrence.

2. There is adequate and substantial evidence that at least some of such ores are formed by an ascending primary mineralization of a definite type and period.

3. It is believed that, in many instances, where the origin of the silver-lime ores has been attributed to secondary descending solutions, a re-examination of the evidence will indicate the oxidation approximately in place of a primary mineralization.

IVAN RAGAZ, Andeer, Switzerland (written discussion).—This paper shows development since 1901, when J. W. Malcolmson made an exhaustive report on the district for the Mexico City meeting of the Institute.

This development was mostly in the northern part of the camp parallel to the old lead orebody in ore zones B, C, and D, and future development no doubt will show additional orebodies and prove that the chain of ore-bodies will extend east and west, the same distance as the old lead *manto;* part of this has been proved by the valuable orebody found by the author for the Constancia Mining Co. in its Diaz shaft west of North Almaden mine.

Malcolmson and Shaw report accurately the occurrence of the big lead orebody (which extends about 5 km. from east to west with a width of 30 to 150 m. and a height of 1 to 60 m.) except that they have neglected to mention that the richest orebodies had, as a rule, the agglomerate as the hanging wall or roof. This has been shown in San José, San Salvador, and Fronteriza mines. Only in isolated places is this shown, as other-wise lime rock is always the hanging wall. It seems, that when the ascending mineral-carrying solution struck the impermeable hanging wall of agglomerate, the deposition of the orebodies was concentrated and the orebodies eastward on the dip enriched. This was noticeable in the San Salvador, Encantada and Fronteriza mines; there was a gradual drop in the silver content of the orebody all through San Salvador, Encantada, and Fronteriza in going east, until in Fronteriza it was enriched again by the large contact lead orebody north of the Fronteriza shaft close to the Encantada line. I do not claim that the mineral-carry-ing solution came up only under the agglomerate cap, but at the other places the solutions entered the lime hanging wall and the valuable minerals (especially silver) were scattered over a large territory, whereas the impermeable agglomerate concentrated the deposition to certain strata.

The value given for the ore mined for the first 20 years (1880 to 1900) is too low. In 1888, the El Paso smelter was finished and received ore from Sierra Mojada; the first contract read as follow. "No ores received with less than 28 per cent. Pb; no payments made for lead contents." As the freight from Sierra Mojada to Escalon alone was 14 pesos per ton and silver was the only content of the ore paid for, it required lead ore

with over 1 kg. of silver to pay expenses. The average shipment contained from 1½ to 3 kg. silver and 30 per cent. lead; also the slags of the many adobe furnaces averaged 150 to 300 gm. of silver and 10 to 14 per cent. lead, showing that the ore had been exceedingly rich or the extraction, considering the barren silica and lime flux, less than 50 per cent.

The chief drawback of the camp has been the high freight rates and the leasing system in vogue, which made selective mining much more necessary, as for instance in Mapimi, Santa Eulalia, and Velardeña. A large tonnage of low-grade lead ore has been left in some of the mines.

The freight on the Mexican Northern Ry. was 6 pesos per ton for a haul of 125 km. The lowest freight rate to the nearest smelter at Torreon is 5 to 10 pesos per ton, according to the grade; on 6000 to 10,000 tons a month, this is a heavy burden on the camp.

The only way to avoid this heavy charge would be to establish small smelters at the mines; the extraction of silver and lead would, of course, be considerably less, but 25 to 30 per cent. of the saving in freight alone would make up for the difference and give the miner from 3 to 6 pesos per ton more for his ore. It is said that the revenue of the Mexican Northern Ry. paid for the road construction before it reached Sierra Mojada. It was built in about 1½ years and was considered the only truly "gold mine" of the camp.

The leasing system had the disadvantage that the lessor had little interest in spending large amounts of money on exploration work that would require years, although some lessors worked the mines much more thoroughly than some of the owners.

The royalties were rather high too. The Consolidated Kansas City Smelting and Refining Co. paid 50 per cent. royalty for the San Salvador mine, the ore delivered free of all cost to the owners on the surface; over 1,000,000 tons were mined at that royalty from San Salvador. At the time this mine was taken over, it was estimated that the mine had 500,000 tons of lead ore in sight, with an average assay of 300 gm. silver and 25 per cent. lead, 20 per cent. iron and 10 per cent. silica. This mine has produced up to date about 1,500,000 tons of ore. For the last 20 years, the royalties for the well-known mines have been from 20 to 30 per cent.

The principal production of the camp today is lime ore, of which a large tonnage is of low grade. This ore was first mined and shipped from the Fortuna mine in 1891. Since then no doubt about 1,000,000 tons have been shipped to the different smelters.

I am quite well acquainted with the large mining camps of the western part of the United States and Mexico, but do not know of any camp that has had such large bodies of rich silver and lead ores as Sierra Mojada. The San José copper orebody, discovered accidentally in 1892, for 5 years produced monthly 3000 to 4500 tons of ore, which for the

first 3 years averaged 3 to 4 kg. silver and 6 to 10 per cent. copper. This revenue enabled the Consolidated Kansas City Smelting and Refining Co. to weather the financial storm of 1893 and 1894. A strip of land between San José and Fortuna, not much more than 100 ft. (31 m.) wide and 250 ft. (76 m.) on the dip, yielded $2,500,000. In the Veta Rica W. Hageman, mine superintendent of the Cia. Metalurgica Mexicana of San Luis Potosi, found the first rich copper orebody; in 1898, they had a face of ore about 300 ft. (92 m.) from east to west and from 10 to 45 ft. in height, that averaged 2½ to 3 kg. silver and 2 to 4 per cent. copper. The company shipped an average of 4000 tons a month of that grade for 5 years, when the grade dropped to 1½ kg. silver.

In the Dionea mine, Charles Dunham, superintendent for the Constancia Mining Co., sank a timber shaft 30 m. and accidently got into the Juarez orebody in ore zone D, which netted the Constancia Mining Co. over 1,000,000 pesos.

Farther east, in the San Salvador mine, W. B. Gates of the American Smelting and Refining Co., found in a crosscut to the north the continuation of the orebodies belonging to zone C in the San Salvador and in the Trinidad to zone D; both discoveries were of great importance and paid large dividends.

Future development will no doubt prove that ore zones C and D have large bodies of rich ore to the west and to the east.

Because of the common-sense mining law of Mexico and the equitableness of the men in charge, there was little or no litigation. The different trespasses were settled in a friendly way and never occasioned any friction; except one between two of the most prominent operators. They did not want to air their grievances in court, so took the dispute in regard to the value of a certain trespass to New York where Judge Hornblower was appointed as the referree. Both companies spent about four times more than the ore mined was worth, for lawyers, witnesses, etc., and received no satisfaction; so that Sierra Mojada is a poor camp from a lawyer's point of view.

WILBUR H. GRANT, San Francisco, Calif. (written discussion).—This paper gives a clear idea of the geology of that interesting camp and, on the whole, corroborates many of the conclusions drawn by George H. Garrey who, with five assistants, made a detailed geological survey of that camp in 1912–13, devoting a year to the field and underground work. As the author has frequently used the same nomenclature as Mr. Garrey, and occasionally cited his determinations, it is assumed that he has had access to that report.

A few statements, however, might be misunderstood without further explanation. In describing the limestone the author states, "Except in the vicinity of the ore deposits, the limestone is almost free of silica and

iron." This is true so far as introduced silica by mineralizing solutions is concerned; there are, however, several limestone beds, or series of beds, that contain myriads of cherts, some of which are 5 ft. in length; these limestones contain a high percentage of silica and commonly are brown to black iron-stained where weathered.

In speaking of the agglomerate he says, "The source of the agglomerate is unknown." As is commonly known among geologists, agglomerate is formed in the neck of volcanoes. Where sluggish or intermittent activity permits crust to form on the molten lava and solidify on the walls, later pulsations break these solidified crusts and sometimes the country rock (in this case limestone) into angular fragments. The numerous volcanic activities that follow produce the intricately complex mass of agglomerate in the volcanic neck and may force it out over the rim of the crater. There is enough evidence at Sierra Mojada to warrant the conclusion that a volcanic neck existed in the valley and the agglomerate formed in the usual way. The thickness of the agglomerate therefore increases toward the central neck, where it will be of indefinite depth.

Some of the statements regarding the "geological history" are indefinite. The author says that the limestones were "folded by horizontal stresses." Our detailed study showed quiet conclusively that the distortion of the limestones from a horizontal position was by a cause similar to that which has produced so many isolated mountains in the uplands of north central Mexico. Numerous examples show that many isolated peaks and ranges were caused by intrusions of igneous rocks from magmatic centers below; the applied force, therefore, is probably vertical rather than horizontal. Furthermore, the structure could more precisely be called a dome than an anticline with the long axis east-west. This dome has been eroded until a valley has been carved out along the long axis and the flanks of the dome now form ridges. During the process of erosion, limestone landslides fell from the cliffs of Sierra Mojada and now, on account of their greater resistance to erosion than the agglomerate, form caps to outliers such as those near Jesús María.

There are several interesting geological facts and conclusions of commercial importance that could amplify some of the statements made to which the author has not referred. The company for whom Mr. Garrey made his detailed survey is still interested in that camp; it would, therefore, be unethical for the writer to discuss these commercial geological questions without permission from that company. It is permissible, however, to state that Sierra Mojada is a camp where a detailed geological survey is essential for the understanding of ore controls. Such an understanding of Sierra Mojada geology will greatly reduce the cost of development and mining of the ore.

Non-metallic Mineral-filler Industry in the Southeastern States*

By W. M. Weigel,† Tuscaloosa, Ala.

The rapid advance, during recent years, in the manufacture of articles that have been in common use for generations and the development of new materials entering into appliances and devices unheard of fifty years ago have created a demand for some of the more common non-metallic minerals. To supply this demand there has quietly developed a branch of the mining industry of which little is known by most mining engineers —the mineral-filler industry. In this paper, and generally in the trade, the term *mineral filler* is applied to a mineral used in its natural state after mechanical preparation only (such as grinding, milling and washing), without undergoing any chemical change and without exerting any chemical action on the substance with which it is incorporated.

The consumption of mineral fillers is confined chiefly to large manufacturing centers in the Eastern and Northern States. Where available, materials originating near the place of consumption are used, as most fillers bring a comparatively low price and, frequently, the freight may equal the cost of the finished filler at the mine. The Southern States have abundant deposits of non-metallic minerals suitable for fillers, which supply the manufacturing industries in adjacent states. Many of these deposits are equal to, or superior, in quality to those mined in the North, but they can not compete with them because of the freight rates.

The mineral-filler industry has felt the present depression, but it has suffered less than most mining industries, for as a rule the mills are simple, the overhead expenses are low, and the labor is entirely local and can revert to farm work when the plant closes down. Many other deposits in the southern district will undoubtedly be developed when business revives and freight rates are adjusted.

Most of the non-metallic mines are small and the tonnage handled is low in comparison with the tonnage involved in modern large-scale mining of coal, iron, copper, etc. On the other hand, at nearly all non-metallic

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mines, practically all of the material mined is marketed, very little being eliminated as waste. Nearly all mining is by open pits, cuts, and quarries. The smaller deposits are worked with pick, shovel, and wheelbarrow; but where conditions justify a larger investment, cable excavators and steam shovels are used. The mills are simple in design and often temporary in character. The short mild winters of the South make better building construction unnecessary; and where a deposit may be worked out in a few years, an elaborate outlay is not justified. At many of the mines there is, however, room for great improvement in milling methods.

The common mineral fillers are white clay, barite, mica, talc, whiting, ocher, silica, graphite, and slate. These will be discussed separately as the methods of mining and preparation are quite different.

WHITE CLAYS

Sedimentary or transported white clays are the only clays used to any extent as fillers; the residual white clays, or kaolins, go to the ceramic industry. The principal centers of the clay-mining industry in the southern district are in the vicinity of Langley, S. C., and in the central part of western Georgia. The clay belt in Georgia extends entirely across the state, from Augusta to Columbus, but the greatest development is from Dry Branch to Columbus." Putnam and Lake counties, Fla., are also producers. Tennessee, northeastern Mississippi, North Carolina, and Alabama have considerable clay resources, but for filler uses the development in these states is small. The extensive development of residual clay, or kaolin, in North Carolina is not considered here.

The clay beds lie approximately horizontal and vary from 5 to 20 ft. (1.5 to 6 m.) thick. In some parts of the bed the clay may be stained or off color; this is sent to the waste dump. The overburden is usually red and brown clay, sand, and gravel and may vary in thickness from a few inches to a depth too great for stripping. It is considered profitable to remove 3 ft. of overburden to recover 1 ft. of clay. The thickest cover known to be stripped is 60 ft. Thin overburden is removed by teams and drags or slip scrapers; when this method is too costly or too slow, mechanical excavators are used. In South Carolina, cable excavators are generally used. The clay is mined by pick and shovel, loaded into cars holding 1 to 1½ tons, and hauled to the drying shed or washing plant. Occasionally charges of low-grade dynamite are used. At a few mines, the stripping is done by steam shovels; but as a certain amount of sorting to eliminate boulders and discolored clay is done at the face, mechanical mining and loading do not seem feasible.

If the clay is sufficiently clean and free from sand so that washing may be omitted, it is piled on racks under cover to air dry, or is crushed or shredded to about 1-in. size and passed through an indirect-heat rotary dryer, and to bins. The dried clay is pulverized and packed in

paper or jute sacks. Two types of pulverizers are used, the Raymond roller mill and a high-speed impact pulverizer; with both types the fine particles drop into a cyclone collector. As the circuit is closed, the fine grit is eventually ground fine enough to pass over with the clay.

Raw clay that contains too much sand or mica must be washed. The clay is passed through a crusher, into a blunger or pug mill, where sufficient water is added to make a thin slip or cream. This slip passes through a short series of troughs, where the coarse material settles; through a screen of 100 to 120 mesh, which removes most of the mica; and then at lower velocity through a series of troughs in which the finer sand and mica settle. If much coarse material is present, a wheel which scoops out the coarse sand that immediately settled and discharges it to one side is placed directly after the blunger. From the mica troughs, the slip flows into one of a series of tanks. When this tank is full, the feed is diverted to the next tank. After the clay has settled sufficiently, the clean water is siphoned off, and the thick mud pumped to filter presses of the usual type. The press cake is dried in air or on steam coils, and pulverized or shipped in lump form. This method of treatment has been used for many years but it is slow and inefficient. In addition, considerable good clay settles out in the troughs and is lost; irregularities in the clay feed cause variations in the density of the slip and in its power for transporting the larger grains; and there is not much chance for blending or making a uniform average product in order to overcome variations in the quality of the feed.

Some clays do not stay in suspension well and make separate settlement of the sand difficult; this can often be remedied by the addition of very small amounts of caustic soda or soda ash to the slip. In order to settle the clay previous to filtration, it is necessary to neutralize the soda by the addition of sulfuric acid. Some clays with extremely "colloidal" properties give considerable difficulty in filtering, making so impervious a cake that filtration stops; this may be remedied by the addition of acid.

It has always been considered impossible to filter clay slip on a continuous filter of the Oliver or Portland type, but recent experiments tend to show that under certain conditions this may be feasible. The problem of continuous washing with the making of a uniform product also seems near solution.

On account of the cost of washing and drying, the operator ships his clay raw whenever it is free enough from grit or mica, with only pulverizing for preparation. By this plan, in many cases, clay of inferior grade has been placed on the market, so that, in some cases, the consumer has discontinued using domestic clay, preferring imported china clay, which could be depended on for uniform quality. If properly prepared, the domestic clays are fully equal to the imported clays for filler purposes.

Clay as a filler is used largely in paper, rubber, oilcloth, paint, and, to

some extent, in phonograph records, polishes, textiles, matches, and soap. For rubber and phonograph records, the color is not important as long as it is light, but for all other uses whiteness and freedom from grit or quartz are the prime requisites. For the different uses, certain physical properties are desirable, but these two requirements are common to all. White clay is seldom used in white paints but is extensively used in colored paints.

BARITE

Barite is produced in Georgia, Tennessee, North Carolina, and Alabama; Georgia ranks first and Tennessee second in production. The Georgia district centers about Cartersville, Bartow County; Sweetwater is the center in Tennessee. The most important development in Alabama is near Angel, in the northeastern part of the state; in North Carolina, in the vicinity of Stackhouse and Hot Springs. Bulletins covering the origin and occurrence of these deposits have been published by the respective state geological surveys.[1]

In general, the barite deposits are derived from the weathering of limestone beds in which the barite formerly occurred as vein filling in the cavities and crevices. Deposits of sufficient concentration to justify mining consist of fragments of all sizes from sand to pieces weighing several hundred pounds, mixed with clay, pieces of rotten limestone, and sometimes quartzite. The barite fragments are rough, slightly rounded, and irregular. The fresh fracture is white to blue white; the weathered surfaces are stained brown to yellow. Mining is practically always by steam shovel in open cuts made in hillsides, or in pits. The average deposits yield 10 to 20 per cent. barium sulfate; 5 per cent. barium sulfate is considered about the lowest grade workable at a profit.

Methods of concentration or washing are fairly well standardized. The mine cars dump on a flat grizzly with rails spaced 4 to 6 in. All ore is worked through this, by sledging if necessary, and large pieces of limestone or other waste rock are thrown out. The ore drops either directly into the end of the log washer or onto a steel plate, from which it is washed into the log. The log washer is double, with 26 to 30-ft. (7.9 to 9 m.) logs making 18 to 20 r.p.m. The slope is 1 in. (2.5 cm.) per ft. The overflow is laundered to a tailing pond; the washed lumps are discharged through a trommel, 42 to 48 in. (1.1 to 1.2 m.) in diameter and 6 to 8 ft. (1.8 to 2.4 m.) long, covered with plate containing 3/4 to 1-in. (19 to 25 mm.) perforations. Occasionally two screens, 5/8 and 1 in., are

[1] C. H. Gordon: Barite Deposits of Sweetwater District of Eastern Tennessee. *Resources of Tenn.* (No. 1, 1918) 8, 48–82; Barite Deposits of Upper East Tennessee. Tennessee Geol. Sur. *Bull.* 23 (1920), 65–70.

J. P. D. Hull: Barytes Deposits of Georgia. Geol. Sur. of Georgia *Bull.* 36 (1920).

employed. The oversize goes to a picking belt where waste constituting 30 to 40 per cent. of the material on the belt is picked out, the barite going to the shipping bin. This material contains 90 to 97 per cent. barium sulfate.

The undersize from the trommel goes to Harz jigs, usually three-cell with ⅛-in. screen. The jig tailings go to the tailings pond and concentrates drawn from the screen go to the shipping bin; various methods are in use for disposing of the hutch discharge. This hutch material contains 50 to 70 per cent. $BaSO_4$; and whenever possible some of it is graded in with high-grade concentrates. Sometimes, after grinding, it is used as a low-grade filler where calcium carbonate or silica are not objectionable; at some plants, it is stacked for future treatment. A few plants are installing tables to improve the grade. Probably considerable. fine barite is lost in the tailings but the large amount of clay with which it is mixed makes its recovery hardly a commercial problem when concentrates are selling for $8 per ton. A double log washer has a capacity of 30 to 40 tons of ore per hour. Most of the barite is shipped out of the district as concentrates, but some is ground before shipment.

In preparing the natural barite for use as a filler, two methods are employed. Where a pure white grade is not required, the concentrates, if in lump form, are crushed and washed in a single log washer, dried, and ground in a roller mill with air classification. This gives a product 96 per cent. of which passes a 200-mesh screen. If a pure white product is desired, the second washing is followed by crushing to ⅛ in. or finer. The material is then leached with sulfuric acid and, after thorough washing, is ground, usually wet, and is water classified, making the so-called water-floated product. This is usually as fine as 98 per cent. through a 300-mesh screen. Blanc fixe, the precipitated barium sulfate, is not considered as a natural mineral filler.

Barite as a filler is used in paper, putty, textiles, and phonograph records, but mostly in the paint and rubber industries. In paper, it is used mainly where a high gloss or finish is desired in super-calendered paper, and in cardboard. On account of its high specific gravity, its use is limited to articles for which it is peculiarly adapted. For paint and paper, fine grinding and pure white color are demanded. For the other uses, fine grinding is desired, especially for rubber, but color is of secondary importance. The content of quartz grains should be reduced to a minimum. Much oxide of iron is very objectionable if used in rubber compounds.

GRAPHITE

Graphite occurs chiefly in Clay, Coosa, and Chilton counties, Ala., which state, since 1915, has produced over 60 per cent. of the domestic graphite; and this field has been the subject of numerous papers

and reports.[2]   There are some deposits in North Carolina and Georgia, but their development is small.

Graphite mining in Alabama consists of open-cut operations, usually with steam shovels, the pits being confined entirely to the upper or weathered parts of the schistose orebodies.   The value of the unweathered ore is yet to be determined.   As graphite has been mined extensively only during the past few years milling practice has not been standardized. Fine grinding is done in Hardinge mills, ball mills, rod mills, and rolls. Dry concentrating equipment used in early days has now been replaced in nearly all plants by oil-flotation machines representing five types.   The crude concentrate obtained from flotation must be refined to increase the carbon content.   Grinding the dried concentrate in burrstone mills and bolting on silk cloth is the most common method of purification, for the gangue material grinds finer than the graphite and is thus screened out. In some plants, the crude concentrates after drying are given a further concentration in a horizontal current of air, the higher grades being then screened without further grinding.   In others the crude concentrates are tabled, then dried and screened.   In one mill, the refining is done on a modified Huff electrostatic separator, with very good results.

Nearly all the mills make three grades of finished products.   No. 1 flake, the highest, which is retained on a 100-mesh screen; No. 2 flake, passing 100 mesh and retained on some finer mesh; and dust, which is screened from the No. 2 flake.   These sizes, though, are not fixed. Some mills retain their No. 1 flake on meshes varying from 85 to 120, and some mills put No. 2 flake and dust together, making one product.   No. 1 flake contains 85 per cent. or more graphitic carbon; and the lower grades, down to 60 per cent. carbon.

The uses of graphite as a filler are rather limited, largely because of its flaky structure, its color, the difficulty of grinding it to a fine powder and of obtaining a product free from grit, and its electrical conductivity.   The highest grade graphite, free from grit, is used as a filler in lubricants; the lower grades, containing 60 to 75 per cent. carbon, are used in paint, stove polish, and foundry facings.   Carbonaceous slates and schists with 10 to 15 per cent. amorphous graphite have occasionally been ground and used as foundry facing and for mortar colors.

MICA

Mica, due to its characteristic structure, is a valuable filler in some lines of manufacture.   North Carolina is the largest producer, but con-

    [2] J. D. Dub and F. G. Moses: Mining and Preparing Domestic Graphite for Crucibles.   Bureau of Mines. *Bull.* 112 (1920).

    H. S. Spence: Graphite.   Canadian Dept. of Mines, Mines Branch, Ottawa (1920).

    U. S. Geol. Sur. *Mineral Resources* for recent years.

siderable mining is also done in Georgia and Alabama. Operations are usually so small that one mine rarely supplies a grinding plant, so material for grinding, which consists of defective mica and scrap obtained from trimming the good sheet mica, is shipped to central grinding plants. The wet grinding method is slow, expensive, and necessitates drying the finely ground mica; it is used only for the highest grades. The scrap is passed through rolls, washed, and screened to remove clay, sand, and foreign substances. It is then charged into the grinder in batches of about 1000 lb. The grinder is a steel tank 10 ft. in diameter by 4 ft. deep, paved with wooden blocks placed on end; attached flexibly to a spider making 10 to 12 r.p.m. are three wooden rollers about 30 in. in diameter by 30 in. long. Enough water is added to the scrap to make a mush or paste, but not enough to cause the charge to flow. After the charge is ground for 24 hr., it is flushed into launders, where grit and heavy materials settle out. Further settling and washing follow; then the supernatant water, with the grease and wood fiber, is drawn off and the mica is removed to a drying floor heated by steam coils. The dry product is passed through a disintegrator and bolted through silk cloth, the oversize being returned to the grinders. The standard size is that which will pass a 140 or 150-mesh screen.

Dry grinding is cheaper and much more rapid, but the product does not command as high a price. Generally less care is used to clean the mica before grinding and the resulting product is often gritty and contains some clay. Ball mills have been tried but, so far, have not met with much favor. The machine generally used is a high-speed pulverizer with $\frac{1}{8}$-in. screen. The material passing through the screen is withdrawn by a fan, passed through a cyclone collector, and discharged into a bin. The mica is then screened into three sizes, from 20 to 100 mesh, in trommels fitted with wire cloth.

Wet-ground mica is used largely in the manufacture of wallpaper, as a dressing or dusting powder for high-grade rubber goods, such as casings and tubes, and in the compounding of rubber. Freedom from grit is essential, and the color must be nearly white and uniform. This is one of the highest priced fillers; the normal price is from 6 to 8 c. per lb. at the plant, but during 1920 it reached 14 c. per lb. The superiority of water-ground mica is said to be due to its better luster, its ability to stay longer in suspension in water, and its lightness or low apparent specific gravity. Dry-ground mica is used as a coating on ready roofing, in some lubricants, phonograph records, ornamental paints, dusting or dressing powder for rubber, pressed and molded goods, and artificial stone.

Within the past few years, the production of ground mica from schists has been developed, largely because of the rapid increase in the manufacture of ready roofing, which uses most of the low-grade material. A ground chlorite mica lately introduced bids fair to replace the more expen-

sive ground muscovite in the roofing trade. It is dry ground with high-speed pulverizers or ball mills. Some deposits provide material pure enough to be marketed direct without purification, while others require air or electrostatic concentration.

TALC

The talc industry of the south has been developed most extensively in southwestern North Carolina and northwestern Georgia. Some of the mining is carried on in open cuts but most of it is underground. Hand and machine drills are used. A minimum amount of explosives is used, as it is desired to obtain as large a number of sound pieces as possible. These are sawn into pencils, crayons, or other shapes and command a much higher price per ton than the ground talc. Scrap from the saws and material not suitable for sawing is ground for filler. Grinding is done in roller mills of the Raymond or Fuller-Lehigh type with air classification, or in tube mills followed by sizing on silk bolting cloth.

Ground talc is used as a filler in paper, paint, textiles, shoe polish, toilet powder, soap, artificial stone, and to some extent in rubber, oilcloth and other articles. It is used largely as a dressing or dusting powder for rubber and roofing, leather, cordage and other lines of manufactured goods, the cheaper off-color grades are used as foundry facing, alone or mixed with graphite. Paint and paper require white talc; for toilet powder only the extra white is used, very little of which is produced in the south. These higher grades must be free from grit; for the other uses, off-color grades with small percentages of impurities are acceptable. For roofing manufacture, 60-mesh talc may be used; but most of that used for other purposes is ground to pass 200 mesh.

WHITING

Whiting, the common name for finely ground marble or limestone is one of the most widely used fillers. True whiting is ground chalk; other ground calcareous materials are substitutes. Production is now confined to Georgia, Tennessee, and North Carolina. Waste materials are commonly utilized, marble-dust whiting being made from the unsound blocks and scrap from marble-working plants. Primary crushing is by jaw or gyratory crushers; secondary crushing, by rolls or hammer mills. Both wet and dry methods are used in final grinding. The dry method is the cheaper, as it does not involve drying the finished product, which is always an expensive operation. Roll mills with air classification, making a product about 98 per cent. of which will pass 200 mesh, is the most common dry-grinding equipment. Wet grinding is done in ball or tube mills followed by water classification, settling in tanks, and drying by spraying the thickened pulp on the outside of revolving steam-heated cylinders. After drying, it is necessary to pass the whiting through a pulverizer to

break up lumps formed during the drying process. Good water-floated whiting should run 98 to 99 per cent. through 300 mesh.

Both the dry- and the wet-ground materials are used in rubber, paint, a few grades of paper, linoleum, oilcloth, shoe polish, putty, tooth powder, roofing, artificial stone and asphalt paving. For use in rubber, linoleum, and paint the alkalinity should be very low. Freedom from grit is required for nearly all uses. A pure white color is essential for use in paint, shoe polish, and similar products. Low oil absorption is of greatest importance in the paint and linoleum industries. There are many grades of whiting, ranging from the amorphous material produced from true chalk to the highly crystalline material from marble. Different whitings possess different physical properties, often when quite similar in chemical composition, fineness, and appearance. Such differences are often difficult to understand. For example, of two whitings of similar chemical composition, grain size, and general appearance, one will make good putty while the other may not.

OCHER

Most of the southern ocher is produced near Cartersville, in northwestern Georgia, though promising undeveloped deposits occur in other states. Considerable amounts of red oxide of iron are produced, but this is generally used for its color and not as an inert filler. The mines in the Cartersville district have been worked for many years, and practice is fairly well standardized. Both open-cut and underground mining methods are employed. Underground workings consist of tunnels usually less than 100 ft. (30.5 m.) in length, projected into the hillsides to meet the orebodies. From the ends of the tunnels, drifts are made to right and left and the ore is removed by overhand stoping. The ground is heavy and requires much timbering. Very little drilling and blasting are necessary as most of the ore can be broken down with a pick. As the distances are short, tramming is done by hand or, occasionally, by mule. The ore, consisting of a mixture of quartzite fragments, clay, sand, and 30 to 40 per cent. ocher, is delivered on a coarse grizzly where all large rock is picked off and the fines washed through with a stream of water, passing to a single log washer. The coarse discharge, as it leaves the log, passes over a 12-mesh screen, the last of the ocher being removed by a water spray. The overflow of the log, holding the ocher in suspension, is conveyed through a long launder about 10 in. (25 cm.) square with a slope of ¼ in. in 16 ft., where the fine sand settles and is shoveled out by hand. From this point, the process varies at different plants. One method is to conduct the ocher-laden solution into a series of settling tanks. The water is siphoned off, and the sludge shoveled into shallow bins equipped with steam-drying coils. When dry, the ocher is pulverized and bagged or barreled for shipment. By another method, the

ocher-bearing water is passed to a Dorr thickener, the overflow being conducted to settling ponds that are cleaned out once or twice a year. The thickened pulp is passed through a Hardinge mill and from there to storage or to agitating tanks, from which it is fed by sprays to the outside surface of steam-heated drying cylinders. The dry ocher is then pulverized and bagged or barreled.

Domestic ocher is used largely as a filler in linoleum and to some extent in paint. While its color falls below the standard of some of the French varieties recent tests indicate that the oil absorption of Georgia ocher is somewhat lower. It is used to a limited extent as a filler in some grades of rubber and paper. A good ocher should have a uniform strong color, pass a 200-mesh screen, be low in lime, and have low oil absorption. This last property is especially important for linoleum and paint. Nearly all American ochers contain some free grit or grains of quartz, but this may be greatly reduced by proper treatment.

SILICA

Silica that may be used as a filler occurs as diatomaceous earth, quartz, and tripoli. Diatomaceous earth has been mined in Florida. Tripoli, which has resulted from the weathering of chert or siliceous limestones, has commonly been produced in preference to quartz as a silica filler on account of the ease of grinding. Water-floated silica or tripoli has, until recently, been obtained from a deposit near Cleveland, Tenn. Abandoned workings have been noted near Tredegar, in northeastern Alabama, Riverton Junction, in northwestern Alabama, and in the northeastern part of Mississippi. Many deposits in the south are pure white and would yield a good product. The Alabama and Mississippi deposits contain considerable proportions of undecomposed chert; accordingly in preparing this material crushing should not be employed as it would introduce angular gritty particles in the finished product. At one plant, the fine material was washed free from the rock fragments, then water-floated or classified into different sizes, dried by steam, and pulverized. As silica dust is injurious to the health of workmen, wet preparation is always advisable.

Silica as a fine-grained filler is used in paint and wood filler, in metal polishes, nearly all scouring soaps and cleansing powders, and to some extent in phonograph records and rubber. Material between 20 and 80 mesh is used for coating sand-surfaced roll roofing.

The pure white grades of silica are most desirable as fillers and command the best price. For most purposes a soft-feeling product with low apparent specific gravity is preferred. Silica should be classified into three or more products according to fineness of grain, as different industries demand different degrees of fineness. Material passing a 350-mesh screen will generally meet the specifications of the most exacting users.

SLATE FLOUR

Ground slate, or slate flour, is a comparatively new filler material. As a rule, it is ground from the scrap resulting from quarrying and working slate used for other purposes. Recently, however, the increased use of slate-surfaced roofing and shingles has created a demand for crushed slate, or so-called granules, that has been met by operating slate quarries especially for this purpose. Deposits of slate that are worthless as roofing slates or other products requiring sound material are suitable for granules if the color satisfies the roofers' demands. Red and green slates are used most widely, though dark gray is used to some extent. Slate granules are now produced in Georgia and Tennessee. The slate fragments are crushed in stages with intermediate screening. Most granules fall between 20 and 30 mesh in size. The proportion of fines varies from 20 to 40 per cent. of gross production and there is some prospect of using this product in a more finely pulverized form as a filler material.

Good slate flour is soft and usually free from grit or grains of quartz, but rather heavy. At present, it is used in asphalt paving, to some extent in the asphalt coating of ready roofing, some grades of linoleum, a little in fertilizers, in plastic roofing and flooring materials, and similar lines. Red slate flour and ground red shale are used as paint filler, and in the backing of linoleum. Slate gives promise of being of value in some grades of rubber and window-shade cloth[3] and is said to be an excellent inert filler for paint intended for roughing or rubbing-down coats before the final finishing coats.

NEED FOR WIDER INVESTIGATION

The technology of mineral fillers is for the most part an undeveloped field. Few of the filler producers have carried on any research work with a view to improving the grade of their products or adapting them to wider uses. The larger consumers have recently begun to realize the importance of careful laboratory tests of fillers, for such tests indicate why certain fillers are best suited to their needs. It is safe to say that no two consumers, even in the same lines of manufacture, have the same specifications for their fillers. Study, experimentation, and publicity will result in a better knowledge of the physical properties of fillers, and will point the way to their more scientific preparation and use. Such studies will also bring about a better understanding between the producer and consumer with an appreciation by each of the problems of the other, and will thus result in general good to the industry.

[3] Oliver Bowles: Bureau of Mines *Reports of Investigations*, Serial No. 2283.

DISCUSSION

RAYMOND B. LADOO, Washington, D. C.—A rough estimate shows that the domestic consumption, in 1920, of the various fillers mentioned reached nearly 1,000,000 tons and were valued at probably over $11,000,-000. But the fillers mentioned are by no means all the mineral fillers used and the total consumption is probably much larger than the figures just given. The importance of the mineral fillers, however, cannot be adequately judged merely by the tonnage produced or the value of the production, for mineral fillers are essential to the production of important products. At one time, mineral fillers were largely considered dishonest or unnecessary additions to various products to reduce the cost of manufacture or increase weight; this use, today, is negligible.

The diversity of mineral fillers used in different industries is shown by the following statistics obtained from one of the largest manufacturers of automobile tires; no figures were given of the use of zinc oxide.

	PER CENT.		PER CENT
Barytes	33.0	Lead carbonate	7.7
Tripoli	15.4	Hydrated lime	6.6
Whiting	13.2	Aluminum flake (clay)	3.0
Litharge	11.0	Asbestine (fibrous talc)	1.9
Magnesium carbonate (magnesite)	8.2		

Other tire companies also use ground gypsum, dolomite, mica, slate, ocher, umber, and other iron oxides.

Some of the principal industries using fillers are: The manufacture of asphalt, dry batteries, cement, cordage, textiles, iron castings, gypsum plaster, leather, linoleum and oil cloth, lubricants, paint, paper, rubber, shoe polish, soap, tooth powder, wood, putty, ceramics, foundry facings, filters, pressed and molded goods, roofing, fertilizers, artificial stone, dynamite, pipe coverings, polishes, insulating compounds, phonograph records, and matches.

Improvements in the methods of mining, milling, and utilization are largely dependent on setting up standard specifications and tests. Such specifications are difficult to formulate because of the complex and little understood action of filler materials in various products. Such problems as the definition of standard white color, the measurement of deviations from a standard white, the measurement of the relative proportion of extremely fine material too fine for screening, the effect of grain size and grain shape on the value of a filler material, etc. are difficult, if not impossible, to solve with our present knowledge and testing equipment. The study of filler materials will involve a great deal of research but such research will more than repay its cost in the production of better materials, in the lowering of producing costs, and in the substitution of low-cost for high-cost products.

When I saw the long list of fillers that the tire manufacturers used I wondered just why each one was used and what its particular properties were, but the manufacturers are unable or unwilling to give the information.

GEORGE C. STONE, New York.—The effect of different fillers is fairly well known to the rubber people but they will not say much about them. We have done some experimenting and have made tires, in many cases making tires half of one material and half of another to get the identical wear so as to learn the effects of the different compounds under identical conditions, and got some interesting results.

RAYMOND B. LADOO.—One company said that about 3 per cent. of its fillers, outside of zinc oxide, was aluminum flake; what is the source of aluminum flake?

H. RIES, Ithaca, N. Y.—It is the weathered outcrop of flint clay in central Missouri and has an exceedingly fine texture; I have wondered if it is this fineness that makes it valuable.

GEORGE C. STONE.—I think the fineness has a great deal to do with it; a coarse material cannot be used in rubber.

RAYMOND B. LADOO.—Rubber manufacturers have different ideas concerning the fineness of the clay; some say 95 per cent. through 200 mesh, some 98 per cent. through 200 mesh, and some 98 per cent. through 300 mesh.

GEORGE C. STONE.—In a 200-mesh screen, the size of meshes will vary 200 per cent.; the fineness also depends on how the screen test is made. When material has been divided into two parts and one lot screened on a simple gravity screen and one on an impact screen there has been a difference of 150 per cent. in the proportion of material under 0.0198 in. in diameter. There should be some proper method of screening agreed upon, both as to the size of screens and the methods of screening.

RAYMOND B. LADOO.—Would it be possible to get the American Society for Testing Materials to work on that?

GEORGE C. STONE.—I think so. A committee has that class of testing in charge; 90 per cent. of the screen tests are made in such a way as to be valueless. What is wanted is something comparable to the screening being done in practice. If screening is done one way in the mill and another way in testing, the results will tell nothing.

OLIVER C. BOWLES, Washington, D. C.—They should do away with screen sizes in numbers per inch and record the size of opening.

GEORGE C. STONE.—The Bureau of Standards has gotten up a list of screens according to the size of opening. Screens regularly made were

selected so that any of the standard-sized screens can be purchased.   If this list were generally adopted it would be a good thing.   The lists cover the full range of sizes anyone would use.

H. Ries.—In addition to standardizing the screen tests, should we not draw up specifications for the materials used for different purposes?

George C. Stone.—That would be a good thing.   The rubber people, for example, know what they want; their whole object has been to turn out more stuff.   They have made great progress in the last few years and are learning fast.   I do not think there would be much trouble getting the information if we went about it in the right way.

Hugh S. Spence, Ottawa, Ont.—Have the researches that have been made been centralized research?

George C. Stone.—No; the researches were made by several companies.

Hugh S. Spence.—It seems to me a tremendous subject to tackle; one that would involve years of research work.

George C. Stone.—Probably more work has been done on rubber in the last two or three years than on anything else.

Raymond B. Ladoo.—Of course, what little work has been done has been done from the utilization end; in some cases, if the producers knew what the consumers wanted they could produce a much better product.

George C. Stone.—That is especially true in the case of materials like the umbers and ochers; the imported material is inherently no better but it is so much better prepared that it will bring two or three times the price of the domestic.

Hugh S. Spence.—It seems that one rather serious impediment to the standardizing of these materials is that they are relatively very low priced and are usually available in scattered sections of the country so that the consumer will take the material from his nearest source.   Now, though those materials may be similar they will vary within fairly wide limits.

George S. Stone.—There should be some definite way of testing, it might be necessary to establish a color scale.

Hugh S. Spence.—It might be prohibitive for a man to get them if a hard and fast standardization were set up.

GEORGE C. STONE.—My experience in the A. S. T. M. has been that the committee is always made up of consumers and producers in about equal numbers, though the consumer is in the majority. A producer is asked to draw up specifications and then the consumers usually make those specifications more liberal so as not unduly to restrict the market. In every case, I think, the producers proposed stricter specifications than were finally adopted.

T. POOLE MAYNARD, Atlanta, Ga. (written discussion).—This paper demonstrates that we have few, if any, mining engineers qualified to carry on the development of our non-metals, because our colleges and universities have largely confined their courses to the metals and our supply of non-metals has, until recently, largely come from Europe. As the trend of mining has been largely confined to the metals, few mining engineers have had enough training in the non-metals to interpret their commercial possibilities or their proper treatment for application in the industries.

The author says: "The Southern States have abundant deposits of non-metallic minerals, suitable for fillers, which supply the manufacturing industries in adjacent states. Many of these deposits are equal or superior in quality to those mined in the North, but cannot compete with them because of the freight rates."

The mineral fillers described are white clays, barite, graphite, mica, talc, whiting, ocher, silica and slate flour. Of these, Georgia produces more white clays, or fillers, than all of the other states combined; practically all of these clays are shipped to the Northeast and Middle West. The largest competitors of the Georgia producers are the clays imported from England.

More barite is produced in Georgia than in all the other states combined, and practically all of it is shipped to the Middle West and Northeast; the principal companies making lithophone products in New Jersey and Delaware own mines in Georgia. The only competitive product is the German barite.

Alabama has all of the flake graphite in this country, with the exception of some mines in New York and Pennsylvania; all of this product is used in the Northeast and Middle West. The only real competitor of Alabama graphite is that from Madagascar.

Mica is produced almost exclusively in the South; North Carolina produces approximately three-fourths of our output and the entire market for this product is in the manufacturing industries of the Northeast and Middle West. Ocher from Georgia is used in the North and East and about 50 per cent. of the production is exported.

Talc, silica, slate and whiting are produced in considerable quantity in the South, but almost the entire output goes to the Northeast and

Middle West. As a matter of fact, freight rates have very little to do with the market for these materials. No matter what the rates were during the recent war, or what the risk, many of our industries using clay continued to purchase English clay, because we were not producing grades of equal quality. Clays of equal quality can be produced in this country by the application of technical control and better methods of refining, but it has been the policy of most producers to aim at larger production, with larger profits, where little technical control was necessary; however, we are learning the importance of production of better quality products.

We import many pigments from Spain, simply because they are prepared under better technical control and are more uniform in quality. The difference in the rate of foreign exchange has more to do with the large importations of graphite from Madagascar and bauxite from France than have the freight rates.

The author says: "There is a need for a wider investigation in the application of non-metallic fillers." A better coöperation must be had from the manufacturers who use mineral fillers, standardizing their requirements and making it possible for the producer to prepare materials that will be of the best type for his use. At the same time, our mines and plants preparing mineral fillers must be placed under better technical control. It is to be hoped that the·investigations now being carried on by the Bureau of Mines at Tuscaloosa will give us more definite information on the requirements of mineral-filler industries.

W. M. WEIGEL (author's reply to discussion).—Very little can be done toward standardization or fixing of specifications of fillers by screen sizing. The finest screen now available, 350 mesh, has an opening 42 microns square and the high-grade fillers, especially the clays, are fine enough so that 90 to 99 per cent. will pass this screen, so that a screen test gives no information regarding the proportion of various sizes below this. Various methods of microscopic measurement of the particles too fine to be screen-analyzed are employed, but all of them leave much to be desired and the operation is laborious.

Consumers who specify that the filler shall be 98 per cent., or some other quantity as the case may be, through a certain mesh screen simply have this requirement to eliminate an undue proportion of large grains, lumps, and foreign substances. For example, assuming a product that was 98 per cent. through 200 mesh (74 microns), depending on the character of the mineral, the average size might vary from 4 to 25 microns, but if it were such that it was all retained on a 350-mesh screen (42 microns), thus varying in size from 74 to 42 microns, it would be unfit for most purposes.

Elutriation of samples by water or air gives a fair idea of the proportion of sizes beyond the range of screens, but is unsatisfactory on the

very fine sizes due to flocculation of the particles, and a microscopic examination must be made to determine the purity of the fractions obtained.

Screen analyses on sizes finer than 150 mesh should be made wet on products that are not affected by water. I use a fine gentle spray on the screen until washings come through clear. It is practically impossible to dry-screen materials that pack easily, as clays, on a fine screen.

Colloid Chemistry and Metallurgy*

By Wilder D. Bancroft,† Ithaca, N. Y.

(New York Meeting, February, 1922)

It is eight years since I have been connected actively with metallography, but in this time I have been learning something about colloid chemistry, which may be considered as the chemistry of bubbles, drops, grains, filaments, and films, all things in which at least one dimension is very small or in which the surface is large relatively to the mass. Films, of course, suggest to the metallographist Beilby's amorphous film theory, about which I am going to speak for a while, though probably not in the way you expect. I come to bury Beilby, not to praise him.

So far as I can judge from looking over the literature hastily, there are two schools in the United States, those who accept Beilby's theory willingly and those who accept it unwillingly. Hoyt[1] says that:

At present there seem to be two schools in metallography, one which has adopted the amorphous-phase theory for the explanation of the effect of cold work, mechanical deformation, etc., and one which is not ready at present to adopt this theory. It seems to me that this theory can at least be used as a convenient means of explaining various facts that are demonstrable in the laboratory but the presence of this amorphous material, through our lack of experimental means, we are unable to prove. That we have failed in this direction should not necessarily hinder us from using a theory as workable as this.

Mathewson[2] says:

I have come reluctantly to believe in the amorphous theory, or some theory admitting a modification of the ordinary crystalline phase so as to permit differential properties with temperature, etc.

If Mathewson represents one extreme, with Hoyt perhaps occupying a middle ground, H. M. Howe[3] may be cited as one of the enthusiastic supporters of Beilby's amorphous theory. It is, says Howe:

[1] *Trans.* (1919) **60**, 572.

[2] *Trans.* (1919) **60**, 562.

[3] "Metallography of Steel and Cast Iron," 373, 386, New York, 1916. McGraw-Hill Book Co.

The first to offer a reasonable explanation of the mechanism of plastic deformation in metals, and in particular to explain ductility, the retention of continuity during the deformation of these crystalline masses, and the extraordinary fact that their tensile strength is usually greater than their elastic limit . . . . This brilliant theory is extremely useful in explaining a great mass of extraordinary and hitherto unexplained phenomena. The doubts which remain today, so soon after its enunciation, as to its competence to explain the intricate phenomena discussed in the last three sections are indeed such as might be expected. They may lead us to class it rather among the precious working theories than among those firmly established.

Beilby's theory[4] is that polishing or working a metal produces a film of amorphous material on the surface or at the slip planes, amorphous being used in the strict sense as referring to a supercooled liquid. Rosenhain goes further than this and believes that chill-cast metals contain amorphous films.[5]

The view has been held for some time by a number of metallurgists and has been particularly advocated by one of the authors (Rosenhain) that a thin layer of amorphous metal in a state corresponding to that of a greatly undercooled liquid exists between the adjacent crystals in any crystalline aggregate, and especially in metals.

Now metals have such a high tendency to crystallize that nobody has ever succeeded in getting any of the orthodox ones in the form of a glass. In fact, Rosenhain and Archbutt[6] admit quite frankly that the evidence for the existence of an amorphous film is entirely circumstantial. When considering some of the main objections to their views, they say:

The first of these difficulties relates to the fundamental question whether an amorphous intercrystalline cement can or does exist. The evidence for and against this has been fully discussed elsewhere; here, it need only be said that until other methods of investigation become available, any direct proof of the existence of the cement can hardly be hoped for.

In other words, the hypothetical amorphous metal is so instable that there is no hope at present of preparing it in mass and yet it is assumed that the amorphous film persists nearly up to the melting point of gold,[7] for instance. This seems so hopelessly improbable that some people have discarded the hypothesis except for cold-worked metal. Hatfield and Thirkell[8] say:

Investigators have, from time to time, made much of the fact that metals have fractured between the crystals. In view of the wide discussion which has taken place upon Doctor Rosenhain's theory as to the existence of an intercrystalline film of amorphous material, the authors feel that the whole question should be reconsidered.

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[4] Rosenhain: "Introduction to the Study of Physical Metallurgy," 246. New York, 1915. Van Nostrand.

[5] Rosenhain and Archbutt: *Proc.* Roy. Soc. (1919) **96.** 58.

[6] *Op. cit.,* 66.

[7] Rosenhain: *Op. cit.,* 260.

[8] *Jnl.* Inst Metals (1919) **22,** 78.

Personally, they are unable to accept this theory of the existence of an amorphous film between the crystals in cast or truly annealed metal. They wish, however, to make their position perfectly clear with regard to the amorphous theory generally. Sir George Beilby, in his classical researches upon the films produced on the surfaces of metal by polishing, established, in their opinion, the existence of amorphous metal. They are prepared to accept the extension of the theory to the effect that in cold-worked material the amorphous phase is brought into existence by the breaking down of the crystals. Whether some of the crystal debris can be truly proved to be the amorphous phase is perhaps at present a little doubtful, but it is obvious to the authors, in the light of several investigations conducted at the Brown-Firth Research Laboratories in recent years, that the physical properties of cold-worked material, as originally indicated by Sir George Beilby and other workers, are entirely different from those of the crystalline metals from which they are produced. The authors therefore accept, for guidance in their investigatory work, the amorphous theory as applied to cold-worked materials. When, however, they come to the existence and properties of an amorphous phase left between the crystals after crystallization, as suggested by Doctor Rosenhain and others, they must confess that they are extremely skeptical.

REASONS FOR REJECTION OF AMORPHOUS-FILM THEORY

I am prepared to go one step further and to reject the amorphous-film theory completely. One of the original strong points about the theory was that it accounted for the increased tensile strength of hard drawn metals and for that it is superfluous. Bennett[9] has shown that the tensile strength of electrolytic copper can be raised to 68,000 lb. per sq. in. by increasing the current density and using suitable stirring. Here it is simply a question of decreasing the size of the deposited crystals and there is no justification for postulating the formation of any amorphous metal. These results do not leave any important part for the amorphous metal to play. If we have got to account for the increased tensile strength as a necessary consequence of the decrease in crystal size, it at once suggests that other characteristics can perhaps be accounted for without postulating amorphous material. It does not seem to have occurred to anybody that if we assign the same arbitrary properties to small crystals that have been assigned the alleged amorphous phase, we account for everything just as well as is done on the other theory.

So far as I can learn, no metallographist has made a serious study of the properties of very small crystals and yet our experience with colloids impresses on us strongly the rather extraordinary apparent changes in properties that occur when a phase is subdivided very much. It seems that the advocates of the amorphous theory have taken whatever properties they needed and have assigned them arbitrarily to the hypothetical amorphous phase. In a letter to H. M. Howe on March 26, 1915, Beilby[10] says:

[9] *Jnl.* Phys. Chem. (1912) 16, 294.
[10] Howe: *Oy. cit,.* 374.

The theory of flow and resolidification in a vitreous-amorphous state for which I am responsible . . . postulates that layers of molecules many molecules in thickness have the mobility of the liquid state conferred on them for a brief period; that it is the transient existence of this mobile phase which makes the slip and movement along the lamellæ possible and easy, and that it is the sudden resolidification of these layers into a non-crystalline or vitreous condition which arrests deformation under a given deforming stress. The original surfaces of any slip are now cemented together by the more rigid material and new surfaces of slip are only developed by higher stresses. The plasticity of the crystalline state is thus gradually used up and the aggregate as a whole becomes more and more rigid. When this has reached a certain stage further increase of stress leads, not to plastic flow, but to disruption.

Beilby does not commit himself as to what gives the disintegrated crystals the mobility of a liquid but Howe says that:

Whether because the small quantity of heat liberated by the friction is so closely localized that it actually melts these particles quite as the close localization of the heat generated by the friction of an earthquake slip may melt the rock locally and thus lead to a lava flow, or for some other reason, the metal torn away by the slip is very mobile momentarily. Finally, that whether because the extremely thin layer thus made mobile cools too rapidly to be able to resume the crystalline state, or because it is of less than crystal unit thickness and hence cannot recrystallize, or for whatever other reason, the metal passes from the mobile into a solid non-crystalline and hence amorphous state, which is stronger than the normal, crystalline state. This greater strength is natural, because the amorphous state lacks the cleavages and slip planes which weaken crystalline substances.

Rosenhain[11] avoids the use of the word liquid; but what he says means practically that:

Just as the rubbing action of polishing produces on metal surfaces a thin layer of altered amorphous material, so we may well expect that the sliding over one another of adjacent slip surfaces will produce a local disturbance of molecular arrangement. If the slip is slight, then it probably happens that the derangement is also slight and temporary and that the disturbed molecules are still able to rearrange themselves pretty much in their original system. If, on the other hand, the slip has been more pronounced, the resulting local disturbance will also be more far-reaching, a greater number of molecules will be disturbed, and they will no longer be able to rearrange themselves in the old crystalline system. A more or less thin layer of amorphous metal will thus be formed on each surface of slip. At first, for a short time, these layers will probably possess a certain degree of mobility, like the surface film which adjusts itself under surface tension. During this period these layers would act as a sort of lubricant, facilitating further slip on the same gliding planes. After a time, however, when the disturbed molecules have had time to "set" in the amorphous condition, we should have on each plane where slip has taken place a layer of hard, non-plastic, amorphous metal.

In view of the high heat conductivity of copper, it seems improbable that any portion of it should be raised to melting during slow working or drawing. I doubt whether anybody believes that tungsten is melted

[11] *Op. cit.*, 246.

continually during drawing and yet that is what Beilby and Howe—and to a lesser extent Rosenhain—either believe or imply. Assuming that it does melt, the next assumption is that it cools so suddenly as to be converted into a glass. It is true that supercooling a melt very rapidly is the normal way to get a glass; but one would not consider the presence of crystals as beneficial and certainly nobody, untrammelled by an unsatisfactory hypothesis, would try to get amorphous metal by pouring the molten metal into a tube of chilled metal. These peculiar properties have been postulated because Beilby did not see how to get along without them.

Beilby [12] performed one striking experiment to show the formation of a surface film as the result of polishing. A slightly pitted copper plate

received its final polishing on fine linen stretched over a hard flat surface and moistened with one of the ordinary commercial brass polishing liquids. On the copper surface prepared in this way, the pits, as seen under high magnification, appear as blue spots on the pale, rose-colored ground of the solid metal. The thickness of the films covering the pits is probably of the order of 10 to 20 micromillimeters.

The blue films produced in this way are Beilby's amorphous, enamellike, transparent surface layer. Nobody will question that they differ in properties from the mass of the metal and the usual reasoning seems to be that what differs from a crystalline substance is necessarily amorphous. This does not follow at all. We have colloidal copper solutions that are either red or blue by transmitted light; but nobody nowadays considers the copper as amorphous in these preparations. By the x-ray interference method, it has been shown that colloidal gold and silver are crystalline[13] and it is quite possible that a similar proof has been given for copper, though I have not seen any reference to it.

It has been suggested that the particles formed in polishing or in the movement of the slip planes might be so small that they could not be crystalline and consequently must be amorphous, from which it soon follows that they are too small to be crystalline and that they are amorphous. This conclusion seems improbable because Oberbeck[14] has found that films of copper deposited electrolytically on platinum to an estimated thickness of $1\mu\mu$ gave the same electromotive force as the massive metal. Unless there is something wrong about these experiments, we do not need to worry much about a film not being crystalline because it is too thin; if there is something wrong with these experiments, that should have been shown before. The probability is that the upholders of the amorphous-film theory are not familiar with these data.

[12] *Proc.* Roy. Soc. (1914) **89** A, 593.
[13] Scherer: *Jnl.* Chem. Soc. (1919) **116** II, 274.
[14] *Wied. Ann.* (1887) **31**, 337.

A rather clear-cut case of the practice of claiming things regardless of whether they prove anything or not is given by Howe[15] and is based on an article by Beilby.[16]

That plastic deformation increases the heat of solution is what should follow from amorphization. When crystalline metal is dissolved, as much heat is absorbed in breaking up its crystalline structure as was evolved in the act of crystallization. When amorphous metal is dissolved, the total heat evolution is greater because this absorption is absent. The amorphization of part of the metal by plastic deformation therefore should lead to a proportional increase in the actual heat evolution on later dissolving the metal.

This is absolutely true but worthless as proof because small crystals have a higher vapor pressure, a higher electromotive force[17] and a higher heat of corrosion. It is impossible, therefore, to tell whether the observed differences are due to fine crystals, which we know we have, or to amorphous metal, which is said to be present.

One of the fundamental postulates of the amorphous-film theory is that the amorphous metal is harder than the corresponding crystalline metal.[18]

Having freely availed ourselves of the conception of "amorphous" metal, it may be well to state precisely what is understood, in this connection, by the term amorphous. In accordance with Beilby's views, and their recent development by the author and his collaborators, the amorphous condition is one in which the crystalline arrangement of the molecules is completely broken up, so that the molecules remain in a state of irregular arrangement similar to that which is supposed to exist in the liquid state. In fact, the amorphous phase is regarded as being, from the point of view of the phase doctrine, identical with the liquid phase. Now extremely undercooled liquids are well known in such substances as glass, vitreous silica, etc., and they are hard, brittle bodies devoid of plasticity and of crystalline structure. Amorphous metal is, therefore, to be regarded as being identical in nature with the liquid metal if that could be cooled down to the ordinary temperature without undergoing crystallization.

Elsewhere Rosenhain[19] says that we have

On each plane where slip has taken place a layer of hard, non-plastic, amorphous metal. . . . As amorphous layers are formed, the capacity of the crystals to yield by slip is diminished and, ultimately, when a certain not very large proportion of the metal has been converted into the amorphous condition, the metal has been rendered incapable of further plastic deformation—it has been rendered hard and brittle.

The implied reasoning is obvious, though not sound. Glass is amorphous and hard; therefore amorphous metal is harder than the corresponding crystalline metal. It is an absolute *non sequitur;* but it seems

[15] *Op. cit.*, 377.
[16] *Jnl.* Inst. Metals (1911) **6**, 22.
[17] Bennett and Brown: *Jnl.* Phys. Chem. (1913) **17**, 177.
[18] Rosenhain: *Op. cit.*, 249.
[19] *Op. cit.*, 246.

to have impressed people all the same. I have tried to find whether there were any facts that would justify the conclusion quite irrespective of logic; but apparently there are none. In my search for information I wrote to Doctor Day of the Geophysical Laboratory. He had nothing on file, but courteously made a few scratching tests with fused silica and quartz, with albite crystals and glass, and with anorthite crystals and glass. He reported that in each case either form will scratch the other; but that, it seemed to him, the crystals scratch the glass rather more readily than the glass scratches the crystal.

One must admire the way in which the amorphous-film theory has been built up, apparently without a single fact back of it. The trouble has been that the people who objected to the theory have always stood on the defensive and have accepted the postulates of the upholders of the theory.

The one good point that the supporters of the theory had is that we get fracture across the crystals in general at low temperatures and inter-crystalline fracture at high temperatures. This was a necessary conse-quence of the existence of a film of amorphous metal because this would soften with rising temperature. In order to account for intercrystalline fracture on the other basis, we must assume that fine-grained metal is the stronger at low temperatures and coarse-grained metal the stronger at higher temperatures. It is far from obvious that this should be and it is especially gratifying to find that Rosenhain, Archbutt, and Thompson[20] have published experiments on certain aluminum alloys which at least indicate that this generalization is true.

While we are on this subject there is one more matter on which I should like to comment. Jeffries[21] says that tungsten is brittle when equiaxed and fibrous when ductile, while the common ductile metals are just the reverse. This is undoubtedly true but it cannot be the whole truth and, to my mind, it misses the important thing. What we want are generalizations and not mere facts, though these last are necessary at some stage in the game. What we really want to know, and what I at least did not learn from Doctor Jeffries' article, is why equiaxed crystals give the maximum ductility in one case for a given crystal size and the minimum ductility in the other.

The metallographists seem, to me, to be very uncritical in another matter and I was very pleased to see Jeffries and Archer[22] take the ground that we do not necessarily have cementite, Fe_3C, dissolved in austenite (gamma iron) or in martensite. In certain cases it is quite a simple matter to tell what are the constituents of a given solid solution. With ammonium sulfate and potassium sulfate there is no reason to assume any

[20] Inst. Mech. Engrs. Eleventh Report to Alloys-Research Committee, (1921) 29, 31.
[21] *Trans.* (1919) **60**, 588.
[22] *Chem. & Met. Eng.* (1921) **24**, 1064

other constituents. With copper sulfate and ferrous sulfate, it seems probable that in one series the constituents are $CuSO_4.5H_2O$ and an unstable $FeSO_4.5H_2O$, and in the other series $FeSO_4.7H_2O$ and an unstable $CuSO_4.7H_2O$. These two cases are quite simple; but what are we going to do with brass and its half-dozen series of solid solutions? Shall we assume six modifications of copper and one of zinc, or vice versa? Shall we assume two modifications of one metal and three of the other? Shall we assume the existence of arbitrary and hypothetical compounds, such as Cu_2Zn, $CuZn$, and $CuZn_2$ for instance?

The orthodox metallographist picks his compounds[23] presumably by intuition. If the hypothetical compound comes in the middle of one of the series, the solid solution is made up of that compound crystallizing with an excess of either component. If the assumed compound lies outside the range, the solid solution consists of one of the components and the unstable compound. After reading a great deal of material of this general type, it is a comfort to come across a man like Doctor Jeffries who has the courage to say that he does not know. I can cite another case that may be of interest to Doctor Jeffries. There is a definite compound Cu_2Sn and also a series of solid solutions extending on either side of the composition Cu_3Sn. Here we know definitely that the compound Cu_3Sn is not a constituent of that series of solid solutions because it differs in properties from the solid solution having the same composition. At present, we have no way of telling what are the constituents of any except the simplest series of solid solutions and it seems to me to be wiser to say so. It is quite possible that an x-ray study of these crystals may give us definite information.

As hardness is unquestionably resistance to permanent deformation, we cannot fail to be impressed by the next step that hardening and strengthening of metals are due principally to interference with slip. I am not sufficiently posted on the details of the martensite controversy to be certain that Jeffries and Archer[24] are absolutely right; but certainly their explanation seems to the layman far and away the best that has yet been put forward. The application to duralumin[25] is very happy and you will please note that this is unquestionably colloid chemistry.

Colloid Chemistry

There is one point which is not colloid chemistry; but which is so closely connected with what I have been saying that I am going to bring it in. That is the question of the iron-carbon diagram. There is a wide-

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[23] Law: "Alloys and Their Industrial Applications," 175. London, 1914, Griffin & Co.

[24] Chem. & Met. Eng. (1921) 24, 1057.

[25] Merica, Waltenberg and Scott: Trans. (1920) 64, 41.

spread belief that at ordinary temperatures ferrite and graphite are the
only phases theoretically stable and that the interesting portion of the
ordinary diagram represents[26] what is, probably, merely a very persistent
metastable system.   So far as I can learn, this belief rests on two facts:
that austenite and graphite are the stable phases along the liquidus and
that, on annealing, a chilled carbon-rich austenite sets free graphite.   The
reasoning is that a spontaneous change takes place in the direction of
equilibrium, that the spontaneous change is to graphite, and that conse-
quently graphite is the stable phase at all temperatures.   This is a
perfectly natural way to reason, but it is bad logic and bad chemistry.
The first phase to appear is a step toward equilibrium but it is not neces-
sarily the final equilibrium.   It was perhaps a natural conclusion as long
as people thought that the carbon in austenite was necessarily present as
dissolved cementite.   Personally, I do not believe that cementite is an
unstable phase, but that is of no special importance.   Whether my
conclusion is right or wrong, the fact remains that the metallographists
are not as critical as they should be and base their conclusions on quite
inadequate evidence.

We can now turn to other less controversial points.   It has been
shown by Fink[27] that if we shake a moderately coarse powder with a much
finer powder, the latter tends to coat the former instead of going into the
voids between the coarser grains.   If coarse thoria, which is a white
non-conducting powder, is mixed with finely powdered tungsten, which is
black and conducts electricity, the tungsten coats the thoria and we get a
black powder that conducts electricity.   If the thoria is fine and the
tungsten is coarse, the same mixture by weight will be a white powder that
does not conduct electricity, because the thoria is now on the outside,
coating the tungsten.

Nobody has worked out the conditions for electrical conductivity in
two-phase alloys; but it is easy to see that there are three possible cases:
When metal A is the continuous phase and metal B the discontinuous or
dispersed phase; when metal B is the continuous phase and metal A the
discontinuous or dispersed phase; and the interlacing system in which
both A and B are continuous.   If A is a metal or phase of high conduc-
tivity and B is a metal or phase of low conductivity, the alloy will have a
much higher conductivity when B is the discontinuous phase than when
A is and yet this point is not considered in the discussion of the
conductivity of alloys.

What looks like a very important application of colloid chemistry is
the breaking of metals under fatigue tests.   There is considerable
evidence to show that non-metallic inclusions, or sonims, are an impor-

[26] Rosenhain: *Op. cit.*, 160.
[27] *Jnl.* Phys. Chem. (1917) **21**, 32.

tant factor in determining the life of a bar under fatigue tests; if this is true and if it proves possible to eliminate sonims economically, this means a complete revolution in engineering practice. The possibilities are wonderful, first to aviation and then to the world at large. I do not guarantee the calculation; but I have been told that if all metal were as good as the best that is now made, it would be possible to cut the weight of a locomotive to that of a Ford car without decreasing its power, though the loss of traction would be large.

RELATION OF COLLOID CHEMISTRY TO CORROSION

Another problem which is essentially colloid chemistry is corrosion. Practically all metals corrode in contact with aqueous solutions and air. While there are all gradations of corrosion and while it is possible that we cannot draw a sharp dividing line, we can distinguish the extreme cases without difficulty. In the one case, the corrosion continues until the metal is destroyed, and in the other the attack comes practically to a standstill. Iron and nickel are two striking illustrations of this. Although iron and nickel are close together in the periodic table, we electroplate iron with nickel in order to preserve it.

If a metallic surface is corroded by the medium, the attack must continue as long as the unchanged surface is exposed to the unchanged medium. A cessation of the attack under otherwise unchanged conditions can occur only when the unchanged medium does not come in contact with the unchanged metallic surface; in other words, when the metallic surface becomes coated with a film that prevents the corroding medium from coming in contact with the metallic surface. In a way this is a truism. Everybody knows it except the people who study corrosion. We paint or varnish metals to protect them. We know that aluminum and nickel owe their relative stability to the formation of an oxide film. Everybody knows that a copper roof turns green fairly rapidly and that the coating of basic carbonate retards the action of the weather on the underlying copper. Parkerized iron owes its resistance to corrosion to the presence of a film of oil and phosphate. Coating iron suitably with ferro-ferric oxide decreases the corrosion, while an unsuitable coating with the same compound accelerates the corrosion. Lead and lead peroxide in sulfuric acid form a storage battery; but the positive plate consists of lead and lead peroxide without being a short-circuited cell, because the lead peroxide coats the lead and prevents the sulfuric acid from coming in contact with it. The whole problem of retarded corrosion is, therefore, essentially a question of film formation. This has been recognized explicitly by Buck, Richardson, and others; but it is not generally accepted as yet and little has been done along the line of determining the theoretical conditions necessary for the formation of a protecting film. Great stress is still laid on electromotive-force measure-

ments, though these are practically valueless. The measurement of the electromotive force shows which of two metals has the greater tendency to start corroding, a thing that is only of academic interest; it shows nothing as to the important question whether the metal will keep on corroding. Electromotive-force measurements would tell us nothing about the marked difference between iron and nickel unless the measurements were made so badly that they were not measurements of the true electromotive force. We find a striking illustration of the futility of electromotive-force measurements in the case of the copper-tin alloys. The electromotive forces of all the alloys lie between those of tin and of copper and one would therefore expect the corrodibility to decrease as one went through the series from tin to copper. As a matter of fact, Curry[28] found that the electrochemical corrosion is relatively slight in most solutions for the bronzes containing 40-75 per cent. copper and is much higher for alloys richer in copper or in tin. The reason for this apparent passivity was found to be the formation of a surface film of stannic oxide; but we do not know why this particular type of film is formed in these particular alloys.

Many people do not believe that there is an oxide film on the surface of passive iron; very few people believe that there is an oxide film on the surface of chromium or of ferrochrome, and we are still more vague in regard to ferrosilicon. An independent test for the existence of such film would be very desirable. I think that a test can be developed which will be applicable in most cases, though perhaps not in all. A lead-plate anode in sulfuric acid becomes coated with lead sulfate, which is then oxidized to lead peroxide; this latter forms a film that protects the lead beneath, pretty well, from further electrolytic corrosion. If the current is reversed, the lead peroxide is reduced to lead sulfate or to spongy lead as the case may be. By successive reversals of the current (in other words, by successive oxidation and reduction) it is possible to corrode the plate to any extent desired. This is the Planté method of forming storage-battery plates; it is sure but it is slow. If an alternating current is passed through lead electrodes in sulfuric acid and the current density made high enough, the lead corrodes completely to lead sulfate. This might be applied in other cases where there is a film; but with so high a current density, one does not know the temperature at the electrode surface. It would be worth while to try superposing an alternating current on a direct current with lead electrodes in sulfuric acid. While this might not work, on account of the insolubility of lead sulfate, we know that it causes the electrolytic corrosion of platinum.[29] It seems probable that an alternating current superposed on a direct current would

[28] *Jnl. Phys. Chem.* (1906) **10**, 474.

[29] Margules: *Wied. Ann.* (1898) **65**, 629; **66**, 540; Ruer: *Zeit. Phys. Chem.* (1903) **44**, 81; *Zeit. Elektrochemie.* (1905), **11**, 661; Haber: *Zeit. Anorg. Chem.* (1906) **51**, 365.

cause passive iron to corrode in concentrated nitric acid, especially as we know that an alternating current[30] alone will cause iron to corrode in nitric acid (Sp. g. 1.40). No experiments of this sort seems to have been made with chromium; but we know that chromium becomes passive when made the anode in hydriodic acid, but dissolves when no current flows, because of the activation due to the hydrogen set free at the surface of the metal.[31] In the case of ferrosilicon, the method might fail, because a film of silica, if formed, would not necessarily be reduced.

After the existence of surface films has been shown in the doubtful cases, it is desirable that suitable methods for the microscopic study of these films should be developed. It is quite likely that an entirely new technique may be necessary. The next step will be to study the properties of these films more carefully than has been done. If a protecting film of stannic oxide can be obtained by corroding certain bronzes, is it not possible to prepare such a film by a method that can be applied to other metals? This is the more probable because of what has been done with parkerized iron. Even if this could be done, it would not be absolutely satisfactory because the film would act like a varnish and would not protect if cracked. What is wanted is a self-healing film. It might be possible to add to pure tin something that would cause it to corrode to stannic oxide with the properties of the stannic oxide obtained by electrolytic corrosion of the copper-tin bronzes containing 40–75 per cent. copper. The reverse is possible because the addition of potassium chlorate to sulfuric acid causes the formation of a porous lead peroxide instead of a protecting film. This makes the rapid formation of peroxide plates a possibility, but the process was not a success commercially because it was not possible to remove the chlorate completely and the plates continued to corrode. Whenever a metal surface is subject to erosion, a self-healing film is absolutely necessary.

We must determine just what sort of a film is formed when copper-bearing steels are exposed to atmospheric corrosion; we should then show that this same sort of film is not formed or does not protect when these steels are immersed in liquids.

A rapid method for studying corrosion must be developed; probably such a test will be an electrochemical one, but there is no convincing proof of this. There is an approximate parallelism between chemical and electrochemical corrosion under certain conditions; but we do not know the limitations nor the best way to carry out the electrochemical tests. Let us consider the chemical corrosion of an alloy in a concentrated sodium-chloride solution exposed to the air. Will the chemical corrosion be the same as the electrochemical? At first sight the question seems

[30] Ruer: Zeit. Elektrochemie (1903) 9, 239.
[31] Bennett and Burnham: Jnl. Phys. Chem. (1917) 21, 147.

absurd because the chemical corrosion is obviously due to oxygen while electrolysis of a sodium-chloride solution sets free chlorine; and everybody knows that the action of chlorine on alloys is, or may be, quite different from that of oxygen. On the other hand Curry[22] found that the chemical corrosion of the bronzes in sodium-chloride solutions was quite different from that in sodium-persulfate solutions and was unexpectedly like the electrochemical corrosion in chloride solutions, while the persulfate solutions behaved like electrolyzed sulfate solutions. The difficulty disappears if we do not consider the oxygen of the air as acting directly upon the alloy. If we consider that we have a cell with a metal anode and an oxygen cathode, the results of the chemical corrosion will be identical with those of the electrochemical corrosion except in so far as variation of current density affects the results. Experiments should be made with carefully selected alloys to determine chemical corrosion in different salt solutions of different concentrations, and these experiments should be paralleled by electrochemical experiments with the same salt solutions at different current densities so as to determine at what limiting current densities, if any, chemical and electrochemical corrosion are identical or nearly so.

Of course the results of a rapid electrochemical test will not necessarily be applicable directly to a case of atmospheric corrosion. The behavior of the copper-bearing steels shows that. If one part of the problem is worked out in good shape, it is practically certain that it will be a simple matter to modify either the experiments or the conclusions so that atmospheric corrosion can be satisfactorily discussed. The difficulty at present is not that we cannot explain everything; we are absolutely certain of nothing definite. We ought to get real results in the future because the whole problem of corrosion is to be taken up by a committee of the National Research Council. This committee will work in coöperation with committees of the scientific societies interested in this problem.

In electrolytic refining, the size of the crystals of the deposited metal are modified by adding to the solution something, usually a colloid, which is adsorbed strongly by the electro-deposited metal and thereby keeps down the size of the crystals. This has made the commercial electrolytic refining of lead feasible. A disturbing colloid phenomenon in electrochemistry is the peptization of molten metal by the bath, producing the so-called metal fogs. When making sodium by the electrolysis of fused caustic soda, it is necessary to keep within narrow temperature limits. The lower limit is that temperature at which the bath freezes. If the temperature is allowed to rise more than 20° above this point, so much metal is peptized and burned that the current efficiency drops practically to zero. With aluminum it is stated that the practical tem-

perature range is a little wider, say 75°. I had always supposed that the peptized metal was burned at the anode; but I was told recently that this is not the case, and that the metal is oxidized before it has moved far from the cathode surface.

RELATION OF COLLOID CHEMISTRY TO ORE FLOTATION

The last matter I am going to mention is ore flotation, which is an important process technically and one of great interest to the colloid chemist. In the process, as worked under the patents of the Minerals Separation Co., a relatively small amount of oil is added to the pulp and air is beaten into it. If there were only oil and water present, we should get bubbles of air with an oil film between the air and the water; these would be so fragile that they would break at once on reaching the surface. As the sulfide ore is wetted more readily by the oil than the siliceous gangue, it passes into the oil film making the latter very viscous and producing a stable froth that rises to the surface, carrying much of the valuable minerals with it, while the siliceous gangue settles to the bottom of the tank. If the amount of air bubbles is very large relatively to the ore, as in the Callow process, the bubbles are not so heavily coated with sulfide and consequently are much more fragile. The one process aims to give a stable froth and the other a fragile froth. To this extent at least the two processes are fundamentally different. The separation of the sulfide particles from the gangue is said to take place at the top of the tank in the Callow process and in the mass of the liquid in the Minerals Separation Co.'s process; but a discussion of this would carry us too far afield.

In this sketch I have tried to make clear that bubbles, drops, grains, filaments, and films are of distinct interest and importance to the members of the Institute of Metals.

(*Discussion of this paper is printed on page* 1152.)

Mechanism of Metallic Oxidation at High Temperatures

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(Rochester Meeting, June, 1922)

THE corrosion of metals is one problem on which time and thought have been expended for many years.   In the effort to avert the destructive action of a hostile environment, attention has been directed almost entirely to the conditions under which most metallic structures are called upon to perform, and the extensive and complicated phenomena attending the corrosive action of atmospheric agencies, acting at the ordinary atmospheric temperatures, have been studied, but there is unanimity of opinion regarding many fundamental facts.

One neglected phase is the corrosive action of various gases upon metals exposed to temperatures considerably higher than the atmospheric range, and more especially the attack of atmospheric oxygen upon exposed metallic surfaces.   That the problem is difficult is connoted by the practical dominance of industrial electric heating by alloys of one type (nickel-chromium).   Proprietary interest may account, in part, for the paucity of information available upon even the most general factors determining the behavior of metals and alloys when exposed to such exacting conditions.

The present paper gives some conclusions reached as the result of experimental work, extending over several years, on the mechanism and contributory factors of the high-temperature oxidation of a number of simple unalloyed metals; it is free from experimental detail and proof.

High-temperature metallic oxidation may be considered as a phase of metallic corrosion.   The scheme outlined below, in which corrosive agent and environment are the principal criteria, is one way of analyzing the several groups of corrosion phenomena.

Erosion.—Disintegrating or destructive action by physical causes.

Corrosion.—Disintegrating or destructive action by chemical causes.

Low-temperature Corrosion.—Solid participating phases essentially immobile: (*a*) Gaseous agent; (*b*) liquid agent: action electrolytic and action non-electrolytic.

High-temperature Corrosion.—Solid participating phases possess diffusive mobility: (*a*) Liquid agent; (*b*) gaseous agent: reducing gases, oxidizing gases.   (1) Water; (2) miscellaneous elemental and combined oxygen.

Except for the division between high- and low-temperature action, the classification is obvious. High and low are only relative terms, with no boundary between them. Similarly, there is no reason for the statement that diffusion is inoperative at a certain low temperature; but there is an essential difference because when the temperature is raised sufficiently the forces of diffusion become active with such rapid amplification as to become dominant. The diffusive mobility of the solid constituents of alloys when highly heated is a property recognized by every metallurgist, and some of the fundamental operations of physical metallurgy, such as the heat treatment and cementation of ferrous alloys, rest upon it as a foundation. The solution and diffusion of gases in solids are appreciated by all who have handled hot hydrogen.

Oxidation is a chemical reaction, and a reversible one; that is, in the presence of oxygen, at a sufficiently high temperature, a metal may be oxidized or its oxide decomposed by suitably adjusting the pressure (concentration) of the oxygen atmosphere. At pressures greater than the equilibrium dissociation pressure of the lowest oxide, oxidation occurs; at lower pressures, the oxide decomposes and the metal is unchanged in contact with free oxygen. This dissociation pressure is not a constant; it increases with rising temperature somewhat after the manner of vapor pressures. For example, oxidation of silver and palladium occurs, at atmospheric pressures, at low temperatures and reduction at higher ones.

The first great division in the behavior of metals may be made on the basis of the order of magnitude of the dissociation pressure of the lowest oxide, serving to separate the noble from the base metals. The former fall within the range of atmospheric pressures and above; the latter are extremely minute (e.g., at 1000° C., the dissociation pressure of cuprous oxide is estimated[1] to be of the order of ten-millionths of an atmosphere). The few metals that by this distinction may be classed as non-oxidizing are gold, platinum, rhodium, silver, palladium, and some chemically similar.

The base metals comprise the remainder of the common metallic elements and may be divided according to the physical texture of the oxide formed; the determining factor is one of relative metallic and oxide density. As the surface of a metallic wire, for example, is being oxidized a cylindrical shell of metal is replaced by a cylindrical shell of oxide, which may or may not take up more space than the former. The characteristics of the formation of subsequent layers of oxide depend on which of these conditions is fulfilled. If the layer of oxide formed takes up more space than the volume of metal that was oxidized to produce it, the oxide layer forms a tight-fitting, solid sheet; if the reverse is the case, there is

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[1] W. Stahl, *Metallurgie* (1907) 4, 682.

more space available within the original metallic dimensions than the oxide density requires, and a porous, bricky structure is the result.   In the former case, the enveloping oxide layer may impede further oxidation; in the latter, because of its porosity, it does not.   The critical density ratio is given by the expression $\dfrac{Wd}{wD}$, in which W is the molecular weight of oxide formed; w, formula weight of metal; D, density of oxide formed; d, density of metal.   The cellular oxide structure results when this ratio is less than unity; the solid crystalline oxide when it exceeds unity.   Table 1 gives values for this critical ratio computed from data available in current physical tables.

TABLE 1

METAL	CRITICAL DENSITY RATIO	METAL	CRITICAL DENSITY RATIO
Aluminum	1.28	Nickel	1.68
Barium	0.78	Potassium	0.51
Cadmium	1.32	Silicon	2.04
Caesium	0.42	Sodium	0.32
Calcium	0.78	Strontium	0.69
Chromium	3.92	Thorium	1.36
Cobalt	2.10	Tin	1.33
Copper	1.70	Tungsten	3.30
Iron	2.06	Zinc	1.59
Lead	1.31	Zirconium	1.55
Lithium	0.60		
Magnesium	0.84		
Manganese	2.07		

Those metals that fall into the cellular oxide structure group comprise most of the so-called light metals; the characteristics of oxidation of the group have been confirmed experimentally with magnesium and calcium. Wires of these two metals heated in dry oxygen below their ignition temperatures (about 525° C.) form oxide coatings, the thickness of which increases directly proportional to the time of oxidation; that is, the formation of an oxide film does not interfere with the speed at which subsequent films are formed.   After complete oxidation of a calcium wire, the resulting lime wire had the same micrometer diameter as the initial metallic diameter, and from the known densities of calcium and calcium oxide it appears that 45 per cent. of the volume of this lime wire was void.   A steady stream of oxygen will pass through this porous layer to the oxidizing metallic surface without detectable impedance; expressed differently, the oxygen pressure gradient through an oxide coating of this type is zero.

The remaining group, which is composed of most of the common metals, form compact enamel-like jackets of oxide, being free to expand outwards as they form, and all have a progressive decrease in rate of

oxidation at constant temperature, proportional to the thickness of the layer of oxide formed. This leads to a parabolic relation between time and quantity of oxidation, the square of the weight of oxide formed per unit of surface being proportional to the time of oxidation. Quantitative verification of this fact has been obtained with copper, iron, nickel, zinc, lead, cadmium, and aluminum. At any time, the rate of oxidation is inversely proportional to the thickness of the layer of oxide then on the surface. The fact that this rate various quantitatively with the amount of oxide overlying the metal surface points to an important conclusion, that the speed of the reaction is not determined by any property of the metal at all, but is determined by some combination of physical properties of its oxide: To explain the known differences in the behavior of the metals, as regards high-temperature oxidation, metallic properties may be forgotten, but a knowledge of the characteristics of their oxides is imperative. Unfortunately, the determinative properties are of a type in which quantitative information is lacking almost entirely.

The way in which the oxide layer limits the rate of oxidation can only mean that the transmission of oxygen through this layer to the reaction surface, which is the juncture of metal and oxide, is effected against a resistance that is proportional to its thickness. For many reasons it appears that this transmission must be accomplished by the actual solution of oxygen in the metallic oxide, and its diffusion while in the dissolved condition from the outside to the inside surface. The motive force of diffusion is difference in concentration, and speed of diffusion is determined by: (a) the concentration gradient of the diffusing substance and (b) the diffusional resistivity of the obstructing medium. In the case of metallic oxidation, the concentration of free oxygen at the reaction surface is maintained at or near a value corresponding to the oxide dissociation pressure; at the outer surface it is that of the maximum solubility of oxygen in the oxide, and the difference between the two is the operating force causing a steady stream of free oxygen to pass through.

The critical oxygen pressure that we interpret as bearing a close relation to maximum oxygen solubility may be determined by rate observations in oxygen atmospheres of varying tenuity, and the diffusional resistivity deduced from it and the diffusion rate. Table 2 gives a comparison of the critical constants for three metals for a temperature of 900° C.

TABLE 2

	ELECTROLYTIC COPPER	ELECTROLYTIC IRON	ELECTROLYTIC NICKEL
Rate of oxidation $\frac{\text{gm.}^2}{\text{cm.}^2\text{ hr.}}$	0.000127	0.00065	0.00000076
Diffusional resistance of oxide (arbitrary units)	2,840	310	50,000
Critical oxygen pressure, mm. Hg. (maximum solubility)	0.36	0.20	0.04

Nickel at this temperature is 165 times as resistant to oxidation as copper, because the oxygen solubility of nickel oxide, as compared with cuprous oxide, is one-tenth as great and its resistance to diffusion seventeen times greater.

The way in which the rates of oxidation change with temperature, according to our previous analysis, is equivalent to the temperature change in diffusivity of oxygen through oxide, inasmuch as the corresponding change in oxygen solubility appears to be slight. The quadratic rate constant (isothermal) for all of the metals we have examined is related to the temperature of oxidation according to the type equation

$$k = AT^n,$$

a parabolic equation in which

k = rate constant;
T = absolute temperature;
A and n = characteristic constants.

The shape of the curve is determined by the value of n, which is different for the several metals, but always fairly large and usually between 10 and 20. The rate of oxidation thus changes rapidly with temperature; but the generality of the equation makes a simple comparison between two metals difficult, except, which is quite unlikely, when the exponents are equal. Thus, the ratio between the isothermal oxidation rates of two metals varies with temperature and may even reverse in relative magnitude.

As to the fidelity with which metallic oxidation follows this simple law, we can only say that in the case of copper, experimentally observed points in the temperature range 400–1000° C. lie on this curve with great closeness, although at temperatures of 600° C. and below there are deviations in the direction of faster rates, which approach the theoretical rates as a lower limit. The cause for this deviation is apparently due to fissuring of the protective oxide coating by the expansive action of the formation of new underlying oxide layers, and the failure of the original oxide surface to accommodate itself to the expansion on account of insufficient plasticity.

Aluminum and cadmium behave in an anomalous manner in that oxidation proceeds according to the parabolic law, as is required by their density ratios, until an oxide coating that is hardly more than a film is formed, whereupon oxidation ceases. At a temperature of 600° C., this film forms upon aluminum in about 60 hr., and further exposure up to 900 hr. produces no increase. This critical oxide film weighs about 0.00010 gm. per sq. cm., corresponding to a thickness of 0.00002 centimeter.

DISCUSSION

PAUL D. MERICA, New York, N. Y. (written discussion).—This paper is, as far as I know, the first attempt to give what might be called theoretical form to our knowledge of high-temperature oxidation and should prove most serviceable in providing a skeleton theory of the phenomenon, by which such future tests and measurements may be expressed in definite and explicit terms.

The principal argument of the paper—that the rate of oxidation of metal at high temperatures is determined principally by the properties of its oxides—has for some time been generally admitted by all interested in the subject. In fact, the marked effect of the films or layers of corrosion produced in accelerating or retarding corrosion in general, whether at high or low temperatures or by gases or solutions, is thoroughly recognized. Thus the resistance to corrosion of aluminum, the passivity of iron, and the resistance to oxidation of nickel and nickel alloys are quite generally assumed to be due to film interference.

The arguments and data presented by the authors seem quite reasonable in view of our experience with high-temperature oxidation. We have frequently noted in oxidation tests that the oxide coatings of iron, copper, and nickel alloys formed at high temperatures are under a stress of compression, as they will often fly off quite violently when scratched or jarred; this agrees with the authors' idea that it is to the density of the oxide caused by formation with increase of volume that its resistance is due.

The exponential relation postulated by the authors between rate of oxidation and temperature also agrees with our experience, as well as the fact that this exponent is quite widely variable. Thus, wrought iron, within 15 days, shows a penetration of oxidation, at 1000° C. in air, of about 0.30 in. and at 800° C. only about 0.01 in.; pure nickel shows a penetration of about 0.001 in. at 800° C. and about 0.01 in. at 1000° C. in the same time.

I assume that the values described as critical oxygen pressures represent pressures of oxygen above which no increase in oxidation rate for constant oxide-layer thickness and temperature takes place. In this connection I should like to add that the transfer of oxygen through oxide layers may also be considered as due to formation and diffusion of solid solutions of a higher and a lower oxide, as described, for example, by Doctor Sosman in the case of the iron oxides. In that case the values of the oxygen pressures might be regarded as dissociation pressures of the oxide solid solutions. Have the authors formed any opinion on the structure and constitution of these oxide layers?

· The viewpoint of the authors may perhaps be most strikingly illustrated by the fact that· although an oxygen pressure of only 10^{-12}

atmosphere will oxidize nickel at 1000° C., one of 10^{-6} atmosphere is necessary to oxidize copper at the same temperature; actually copper oxidizes at a rate about 200 times that of the rate of nickel at this temperature.

Carbon dioxide and even carbon monoxide gases may be oxidizing in their action, although not ordinarily considered so. Thus a furnace gas containing 15 per cent. carbon dioxide and 1 per cent. carbon monoxide, though usually considered reducing in character, will oxidize a metal, such as iron or even nickel, having an oxygen dissociation pressure as great as 15^{-13} atmosphere at 1000° C. The criterion as to whether a metal will or will not be oxidized by any mixture of carbon monoxide and carbon dioxide is that the equilibrium pressure of oxygen—metal—oxide be less or greater respectively than the dissociation oxygen pressure of the reaction $2CO_2 \rightarrow 2CO \downarrow O_2$ at that temperature.

N. B. PILLING.—It is difficult to find a suitable term for this critical oxygen pressure. If an oxidized piece of copper is held in a space of constant volume in temperature with a limited amount of oxygen gas, and the oxidation, measured by the change in pressure, is taken up by the copper, oxidized pressure falls off, according to the parabolic relation, down to a certain oxygen pressure, which is usually about 0.9 mm. The value of the oxygen pressure is not much affected by the temperature, although it is different for the different metals.

In metals oxidized at a given temperature in oxygen at different absolute pressures, going down from atmospheric pressures, the rate of oxidation is sadly influenced by the change in pressure. We get down to the same critical pressure and from there on it drops off very rapidly.

Crystal Structure of Solid Solutions

By Edgar C. Bain,* M. Sc., Cleveland, Ohio

Of the important phenomenon of the hardening of steel, Professor Sauveur[1] says:

> It would seem as if the methods used to date for the elucidation of this complex problem have yielded all they are capable of yielding and that further straining of these methods will only serve to confuse the issue, the point having been reached when this juggling, no matter how skilfully done, with allotropy, solid solutions, and strains is causing weariness without advancing the solution of the problem. The tendency of late has been to abandon the safer road of experimental facts and to enter the maze of excessive speculations, in which there is great danger of some becoming hopelessly lost. The conclusion seems warranted that new avenues of approach must be found if we are to obtain a correct answer to this apparent enigma.

With some degree of fitness we may say this of other problems. It seems likely that one of the new avenues of approach to many metallographic problems is the study of crystal structure, or more accurately, the atomic arrangement, by means of the x-ray spectrometer.

The crystal structure of some materials has been carefully worked out by the mineralogist, but his methods are, in general, unsuited to commercial metals and a method that yields even better information is at hand as the result of work on the diffraction of x-rays by Laue and the Braggs[2] and later by Hull,[3] St. John, and others. It should be understood that the contour of free crystals, which is the subject of measurements by the crystallographer, is the result of the atomic structure and, in general, bears a simple and obvious relation to it. The use of the x-ray spectrometer[4] has been described frequently, so the procedure will not be given here.

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[1] "Metallography and Heat Treatment of Iron and Steel," 314. Chap. XVIII.

[2] *Proc.* Roy. Soc. London (1913 and 1914); *Met. & Chem. Eng.* (July 1, 1916) **15**, 35.

[3] *Phys. Rev.* (December, 1917) **10**.

[4] Jeffries and Archer: Atoms and Metals. *Chem. & Met. Eng.* (March 23, 1921) **24**, 507; Crystalline Structure of Metals. *Chem. & Met. Eng.* (May 4, 1921) **24**, 771; E. C. Bain: Studies of Crystal Structure with x-rure. *Chem. & Met. Eng.* (Oct. 5, 1921) **25**, 657.

All metals are normally crystalline. Crystallinity requires regularity in atomic spacing; hence, parallel planes of regular spacing can be passed in many ways through a crystal grain containing all the atom centers. The distance between such planes are the data recorded photographically by the x-ray spectrometer. These planes act as a three-dimensional
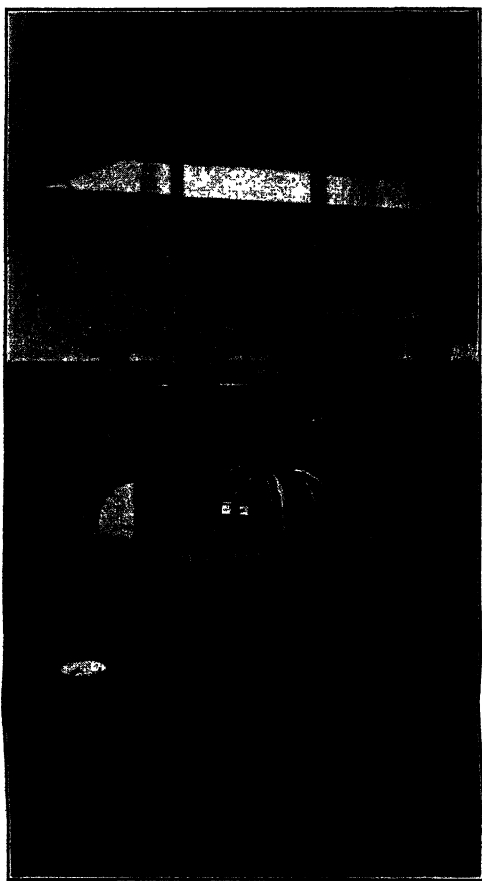
FIG. 1.—A SPECTROGRAPHIC APPARATUS.

grating for diffraction; the nature of the assortment of such interplanar distances found in a crystal determines its type. The ultimate measure of these plane spacings is merely the distance along a strip of photographic film where darkened lines appear exactly as they appear on a photograph of the spectrum of visible light, for instance. Of course, instead of having a constant grating and analyzing the wave length, we have a constant wave length and measure the grating. When we derive the angle of

diffraction from these dark lines and deduce the atomic plane spacing by the classical formula $n\gamma = 2d \sin \theta$ (in which n is the order, γ the wave length, d the atomic spacing, and θ the angle of diffraction) we have only to construct in space a lattice that has spacings to conform to those recorded. A photograph of a complete spectrographic equipment is shown in Fig. 1.

The logical first step in applying such study to metals is the determination of the atomic arrangement or crystal structure for all the useful pure metals. This work has already been done largely by Hull[5] and Davey. In his investigations, the author has added one or two elements to the list and has had occasion to work out independently many of the metals announced by Hull. In every case, the greater the refinement of experimentation the closer was the agreement with Hull's figures.

Table 1[6] shows the metals as they fall in the various types of crystallinity, together with their atomic spacings and ratios. It is fortunate that the metals crystallize in so few simple types of great symmetry; the workability of the majority of metals probably follows as a result of this circumstance. To date, metals have been found only in the face-centered cubic, body-centered cubic, face-centered tetragonal, body-centered tetragonal, close-packed hexagonal, and diamond arrangement in both isometric and rhombohedral types. Some metals have been found to exist in two lattices, corresponding to allotropic modifications with stability in different temperature ranges; for example, iron and cobalt.

As has been stated, both the type and the dimensions of the space lattice for the atoms in metals are computed from the pattern of lines on the spectrogram. Hence, this photographic pattern is essentially characteristic for each metal. A number of such photographic films are here shown. So far as is known, the pattern for any metal differs at least in dimensions from the pattern of any other material. Conceivably, two metals might be identical in lattice size but the author has not found any cases of apparent identity that, with greater accuracy of measurement, did not show some difference. Thus, the cube edge of tungsten differs from the cube edge of molybdenum by only about 0.25 per cent. So the spectrogram takes on the significance of a highly characteristic, even though insensitive, qualitative analytical method. The expressions copper lattice or zinc lattice, for example, refer to perfectly definite spacings—atom center to atom center—so that we may with no sense of vagueness refer to a copper lattice as an entity even though some points in the lattice are occupied by other elements. Equipped with a qualita-

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[5] *Trans.* A. I. E. E. (1919) **38**, Pt. 2, 1445–1466. *Phys. Rev.* (May, 1921) **17**, 549.

[6] The data set forth here are gleaned from the references cited in footnote 5 and from private communications from Doctor Davey.
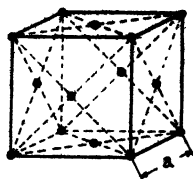
1. FACE-CENTERED CUBIC

	Cm. $\times 10^{-8}$		Cm. $\times 10^{-8}$
Copper	3.60	Cobalt	3.544
Silver	4.06	Nickel	3.540
Gold	4.08	Rhodium	3.820
Calcium	5.56	Palladium	3.950
Aluminum	4.05	Iridium	3.805
Lead	4.92	Platinum	3.930
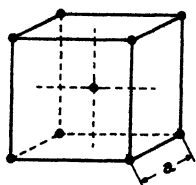
2. BODY-CENTERED CUBIC

	Cm. $\times 10^{-8}$		Cm. $\times 10^{-8}$
Lithium	3.50	Chromium	2.899
Sodium	4.30	Tungsten	3.150
Titanium	3.144	Molybdenum	3.143
Tantalum	3.272	Iron	2.86
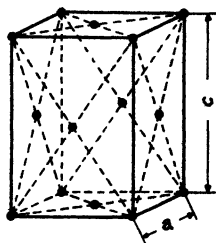
3. FACE-CENTERED TETRAGONAL

	Cm. $\times 10^{-8}$	$\frac{C}{A}$
Indium	4.58	1.06
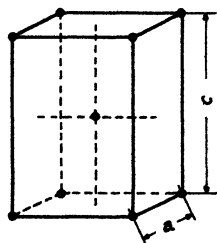
4. BODY-CENTERED TETRAGONAL

	Cm. $\times 10^{-8}$	$\frac{C}{A}$
Tin	4.00	0.94

(First approximation by the author)
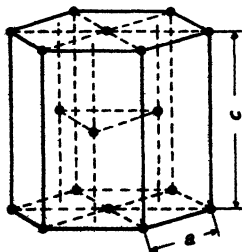
5. HEXAGONAL CLOSE-PACKED

	Cm. $\times 10^{-8}$	$\frac{C}{A}$
Magnesium	3.22	1.624
Zinc	2.670	1.86
Cadmium	2.980	1.89
Cobalt	2.514	1.63
Ruthenium	2.686	1.59

TABLE 1.—CRYSTAL STRUCTURE OF SOME METALS AND THEIR ATOMIC SPACING.

tive method for recognizing the characteristic atomic arrangement of atoms in the various metals, a method that is able to penetrate many thousands of atoms deep into a specimen and destroy none of the specimen, we might hope to glean knowledge of some heretofore undiscovered facts.

Physically, the simple eutectic in metals is regarded as a mixture of the two more or less pure elements. As might be expected, from eutectic specimens the film obtained shows simultaneously two patterns, the one of metal A the other of metal B. The relative intensities of the two superimposed spectra are dependent (among other things) on the composition of the particular eutectic in question. The pattern grows more intense for A or for B as the eutectic is mixed with proeutectic A or B by a change in composition. There is, then, no inherent difference between a eutectic and a mechanical mixture of the powders of two metals, except that in the eutectic the grain size of one constituent is larger than the grain size of the other, the continuous phase having the larger grain.

In eutectics studied by the author, there were cases of limited solubility in the solid. But the two patterns revealed on the film were substantially identical with the two pure-metal patterns. Thus, the tin-lead eutectic showed both the pattern of lead and the pattern of tin, while in fact there were no crystal-grains of pure lead present but a solid solution of a small amount of tin in lead. Silicon-aluminum eutectic showed exactly the same evidence, though aluminum dissolves over 1 per cent. of silicon. This led to the prediction that a solid solution has a crystal structure nearly identical with the structure of the solvent.

COPPER-ZINC SERIES

The question immediately was raised, "What happens to the atomic arrangement in a crystal grain of copper when zinc dissolves in it in the solid?" Here the x-ray spectrometer gives the answer at once. In Fig. 2 are shown the first few lines of the pure copper pattern and alongside,

Alpha Brass

Copper

FIG. 2.—SPECTROGRAMS OF COPPER AND ALPHA BRASS, SHOWING SIMILARITY.

for comparison, the same pattern range for alpha brass of over 30 per cent. zinc content. They are identical so far as our measurement shows— face-centered cubic with cube edge 3.60×10^{-8} cm. Here 30 per cent. of zinc has been dissolved in solid copper without changing the copper atomic arrangement or spacings to an extent that could be observed on

the pattern. Zinc atoms replace copper atoms in the space lattice without materially altering the copper lattice. If we start with a unit volume of copper and add to it zinc in solid solution, maintaining the whole homogeneous, we shall find that the increase in volume is equal to the atomic volume of copper multiplied by the number of zinc atoms added. When metal n dissolves in metal m, the density of the solid solution will be represented by

$$D_n \text{ in } m = d_m \times \frac{\dfrac{\text{per cent. } M}{\text{at. wt. } M} + \dfrac{\text{per cent. } N}{\text{at. wt. } M}}{\dfrac{\text{per cent. } M}{\text{at. wt. } M} + \dfrac{\text{per cent. } N}{\text{at. wt. } N}}$$

In this D is the density of the mixture and d the density of the solvent metal. By this assumption we have fixed the density of solid solutions having the same lattice dimensions as the solvent metal. It is unfortunate that the author has not found data on density of solid solutions of sufficient accuracy or in sufficient abundance to prove this hypothesis completely. The approximate identity of alpha solid solution with copper has been shown for zinc, aluminum, and tin additions. Another example of limited solution in the solid is carbon dissolving in gamma iron to form austenite, whose lattice is that of gamma iron. From these tests and from those described subsequently under the isomorphous series, we are able to give a real distinction to the words solvent and solute. It seems logical to call that metal, of a pair in solid solution, whose lattice persists, the solvent. In the case of liquid solutions, such as alcohol and water for instance, there is little reason for giving either water or alcohol the distinction of being the solvent for their molecules are supposed to be constantly in erratic motion and never in a regular and orderly arrangement. What is usually done is to speak of the material in predominance as the solvent; it is thus that Gulliver[7] uses the term in connection with solid solutions.

Just as there is a limit of solubility of a solute in a liquid, further additions of which refuse to become homogeneous with the saturated solution, in the case of solid solutions there is a limit to the amount of widely different foreign atoms that can remain in a parent lattice. The copper lattice can permit only about 35 per cent. of its points to be occupied by zinc atoms, or 6.8 per cent. of its points by tin. Further additions of solvent gather together, along with the largest number of solvent atoms possible, and establish a new lattice which is compatible with the leaner solvent composition; thus, in the copper-zinc and copper-tin series, a body-centered cubic lattice is formed for the still more diluted copper. It is impossible to say whether or not this is strictly the beta range of the constitution diagram for there is disagreement in the regions for those

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lattices that exist at room temperature. For still higher concentrations of zinc (around 75 per cent.) another stable lattice is rhombohedral. For pure zinc and its solution of copper for the very narrow range, the hexagonal zinc lattice is the only one observed, making four structures in all for the series.

It may be interesting to speculate on the possibility of a compound of copper-zinc being the parent lattice for the body-centered range of the series; it fits nicely into our conception of space lattices. The body-centered lattice is, in reality, two interpenetrating simple cubic lattices with the same cube edge as the body-centered type. If we picture each simple cubic structure as being entirely of one metal, we have a very symmetrical form for a 1:1 compound. Knowing the ease with which copper is replaced by zinc, we can easily believe that this compound lattice could exist with some preponderance of zinc over the equal atomic ratio; this is in accord with the observed fact. We then have a case of solution of an element in a compound. The x-ray spectrographic evidence which the author has secured is a corroboration of the results of Andrews,[3] who deals with the crystallinity of the entire copper-zinc series. It appears that the structures of the copper-zinc series is paralleled by the copper-tin alloys, although the spectroscopy is much more difficult, caused probably by the great disparity in atomic weights.

Solid-solution Types

The question arises as to what dispensation we shall make of the so-called continuous solid-solution types exemplified by copper-manganese, copper-nickel, nickel-iron, nickel-chromium, nickel-cobalt, iron-manganese, copper-gold, silver-gold, tungsten-molybdenum, and other pairs. With the definition given, relating to the retention of the solvent lattice, how can there be isomorphism in an entire range that has two types of crystallinity depending on the two different solvents—two parent lattices? Obviously, in such a case a solid solution of A in B could not be identical with a solution of B in A, even of the same composition. The x-rays reveal the surprising fact that often they are not strictly isomorphous. It appears that the condition for a so-called isomorphous series is merely that metal A can dissolve in its lattice an amount of B that overlaps the composition of B in the solution of A and B lattice. Over a certain range both structures exist simultaneously.

These conclusions are based on the examination of the three series, iron-nickel, nickel-chromium, and copper-manganese. The two last-mentioned series were studied throughout their entire range from alloys chosen at 10 per cent. composition intervals. The first was studied only for a limited range; it had been the subject of investigation of very

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[3] X-ray Analysis of Three Series of Alloys. *Phys. Rev.* (1921) **18**, 245.

accurate work by Andrews[9] and accordingly only a few tests were made.

The chromium-nickel series was chosen for the example of continuous solution without minimum in melting points. All proportions melt
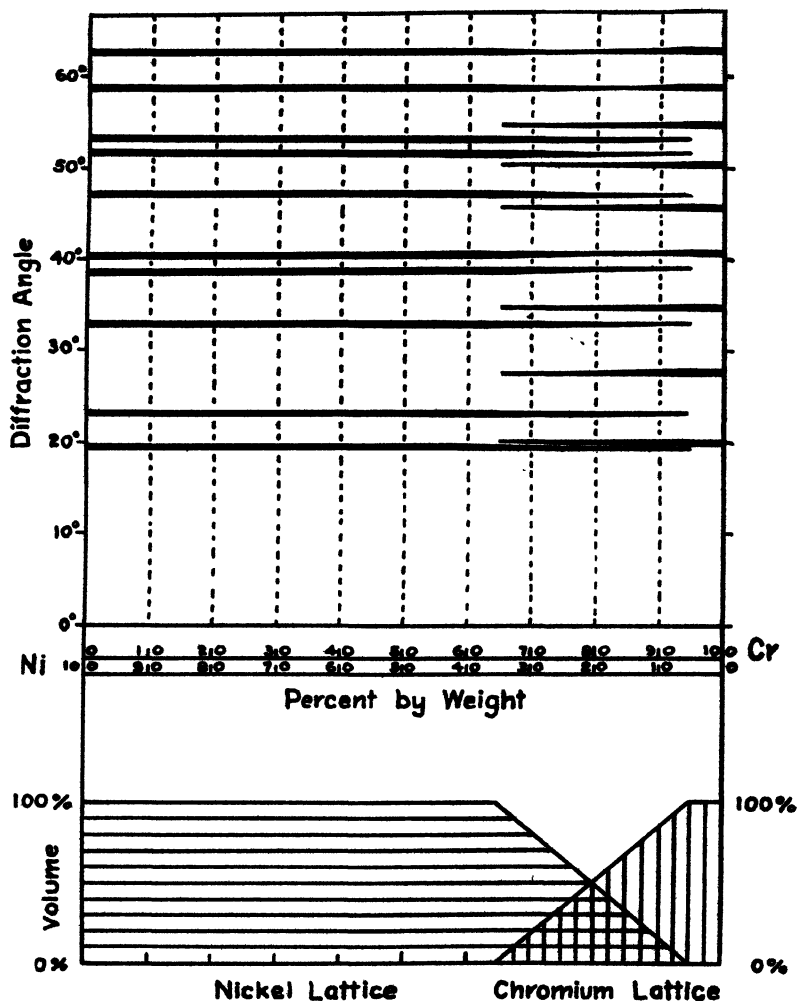
FIG. 3.—CONSTITUTION-PATTERN DIAGRAM; SPACE BETWEEN EACH 10 PER CENT. CHANGE IN COMPOSITION REPRESENTS PATTERN FOR THAT RANGE; HORIZONTAL BANDS ARE SPECTRAL LINES OF CONSTITUENTS EXACTLY AS PHOTOGRAPHED.

between the fusion temperatures of the pure metals. The lattice of chromium is of a different type from nickel, as may be seen from the table. Their patterns are so different as to enable one to follow the

changes in structure very easily; Fig. 3 shows the patterns as they appeared on the successive films. The width of the bands are indicative of the intensity of the two patterns; the probable volume of the two lattices is also charted in the same figure. These relative volumes of the two structures are, of course, only approximate. This figure introduces what we shall call the constitution-pattern diagram. Fig. 4 shows photographs of the nickel and chromium spectra along with their Ni 20-Cr 80 alloy.

Nickel

Nickel-chromium
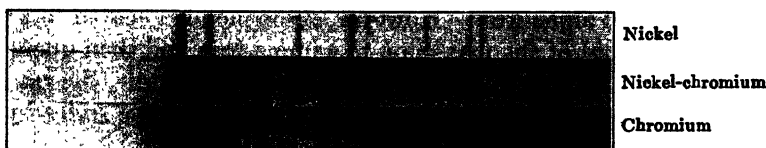
Chromium

FIG. 4.—SPECTROGRAMS OF NICKEL AND CHROMIUM AND THE NI 20-CR 80 ALLOY.

For the type of continuous solid-solution series presenting a minimum in fusion temperature, copper-manganese is the example used. A vast difference in crystalline habit exists here and, as in the case of nickel-chromium, it is easy to follow the change in lattice by observing the pattern. Fig. 5 shows that the range of overlapping patterns is from 60-40 per cent. to 90-10 per cent. Mn-Cu. One would expect this range to include the minimum in freezing temperature, but such is not the case.
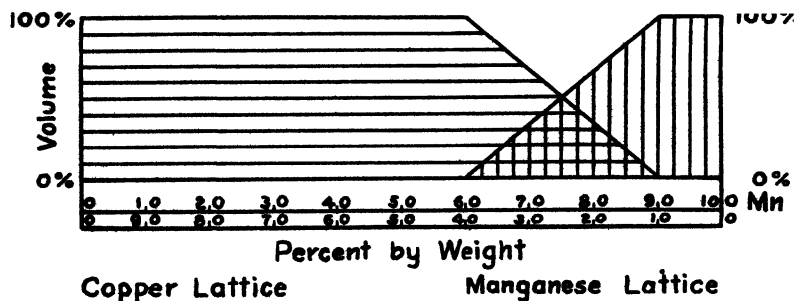
FIG. 5.—CONSTITUTION-LATTICE VOLUME DIAGRAM FOR COPPER-MANGANESE SERIES.

The iron-nickel alloys present a continuous series of solid solutions with a minimum melting point near the pure nickel range. This series has the distinction of being allotropic. The normal room-temperature condition for nickel is face-centered cubic and for iron (alpha iron or ferrite) body-centered cubic like chromium. The alternate form for iron (gamma iron) is face-centered cubic of almost identical dimensions

with nickel. The patterns show that nickel dissolves in ordinary alpha iron to the extent of 25 to 30 per cent. Beyond this point we have the lattice of nickel alone. Gamma iron and nickel are as alike as copper and nickel and no doubt form the same kind of a series, as explained later.

We may assume that the iron, having a stable range in a lattice so like nickel, would be very soluble, having only to exhibit the tendencies of its allotrope to be completely soluble. In amounts of nickel around 25 per cent., the carefully annealed metal shows only the body-centered lattice of alpha iron. However, if the metal is heated to a temperature above that of the transition to gamma iron and then quenched, the whole is rendered face-centered cubic. In other words, the range of overlapping patterns is shifted toward the iron end of the series by the quenching. Of course, if much carbon is present the metal is much more inclined to remain austenitic or face-centered. The alloys that are purely of the nickel lattice are only slightly magnetic, while those of the overlapping range vary up to almost that of iron. Apparently the body-centered crystal is necessary to magnetism in iron.

The effect of thermal treatment was also evident in the case of the nickel-chromium series. In the well-annealed specimens, the chromium lattice was evident as far as 40 per cent. nickel–60 per cent. chromium; while in the same specimens quenched from 1000° C., it was just visible only at 30 per cent. nickel–70 per cent. chromium. This was unexpected, inasmuch as chromium has the higher melting point and would have relatively a more rigid lattice at the higher temperature.

As to the state of aggregation of the two forms of crystallinity existing simultaneously in some ranges of the continuous solution series there is little information, but there is some reason to believe that within the same grain both forms may be found. In the nickel-chromium series, rather large grains were developed by prolonged heating near the melting point. These films revealed, in the main, the characteristic dashed pattern of coarse grains. But in the overlapping pattern range, the stronger pattern alone was of the coarse-grained type; that is to say, above 80 per cent. chromium the nickel was fine grained and below this chromium content the nickel was coarse grained. It would seem, therefore, that a sort of martensitic structure was evolved. Within the grains of the dominant pattern small, perhaps submicroscopic, crystal units of the other lattice were present. In order to escape microscopic detection at high magnification after etching, the weaker lattice must be in extremely small units and likely aided in remaining undifferentiated by a chemical constitution identical with the dominant lattice, so that etching would proceed more evenly. The author hopes to grow excessively large grains of the alloys in the double-structure range and then study the pattern obtained from a very small number of such grains. In connection with another prob-

lem,[10] specimens ranging from one to twelve grains only were examined. The effect would be much more aggravated by this condition.

As we have seen, many of the continuous solid-solution series are really aggregates of two kinds of crystallinity. But this is not necessarily the case. We know that thermal expansion and mechanical stress can alter the size of an atomic space lattice without destroying it or altering its type. The limits of this sort of purely elastic, temporary distortions are very small, perhaps not over 1 per cent. in most cases, roughly in the order of magnitude of the accuracy of our measurements with the spectrometer. It seemed possible to the author that if we could find a solution of two

Nicke

Copper

FIG. 6.—SPECTROGRAMS OF COPPER AND NICKEL.

metals of very similar atomic volume and identical in type, we might discover a complete series whose atomic arrangement remained homogeneous but whose distances gradually increased from the dimensions of lattice A to the dimensions of lattice B. Actually, as later discovered, the lattice size may differ considerably and still give this condition.

Copper and nickel proved to be one example; the patterns, Fig. 6, differ by about $3/32$ in. (2.38 mm.) in the large angle lines. This is a deviation quite sufficient for identification; indeed with the patterns superimposed, the lines beyond the first three appear separate and distinct, as shown in Fig. 7, which was made from mixed powders. Bearing

Nickel and Copper Powders

Same after Sintering

FIG. 7.—SPECTROGRAPHIC EFFECT OF ALLOYING ON MIXED POWDERS OF COPPER AND NICKEL. UPPER SPECTROGRAM IS UNALLOYED; THE LOWER, SAME PRESSED MIXTURE AFTER PROLONGED HEATING TO PRODUCE SOLID SOLUTION.

these facts in mind, the patterns of the nickel-copper series present a unique appearance. In no case was there the slightest appearance of both patterns superimposed. The lattice for the solutions remains perfectly face-centered cubic in character throughout the entire range, the size only changing gradually from the dimensions of one to the dimensions of the other. It may be seen in Fig. 7 that causing solution in the solid to take place, by heating mixed powders, has changed a typical superimposed pattern of nickel and copper into a pattern of a

[10] Jeffries and Bain: Mixed Orientation Developed in Crystals of Ductile Metals by Plastic Deformation. *Chem. & Met. Eng.* (Oct. 26, 1921) 25.

single face-centered lattice of intermediate size. The film is lacking in contrast; the lines are difficult to see, but it offers ample proof of the reality of our true homogeneous lattice. Fig. 8 is the constitution-pattern diagram for the copper-nickel series. The alteration in lattice size, as measured from pattern change step by step, is almost too slight to be seen—yet the change is not quite linear. It is probably linear on an atomic per cent. scale.
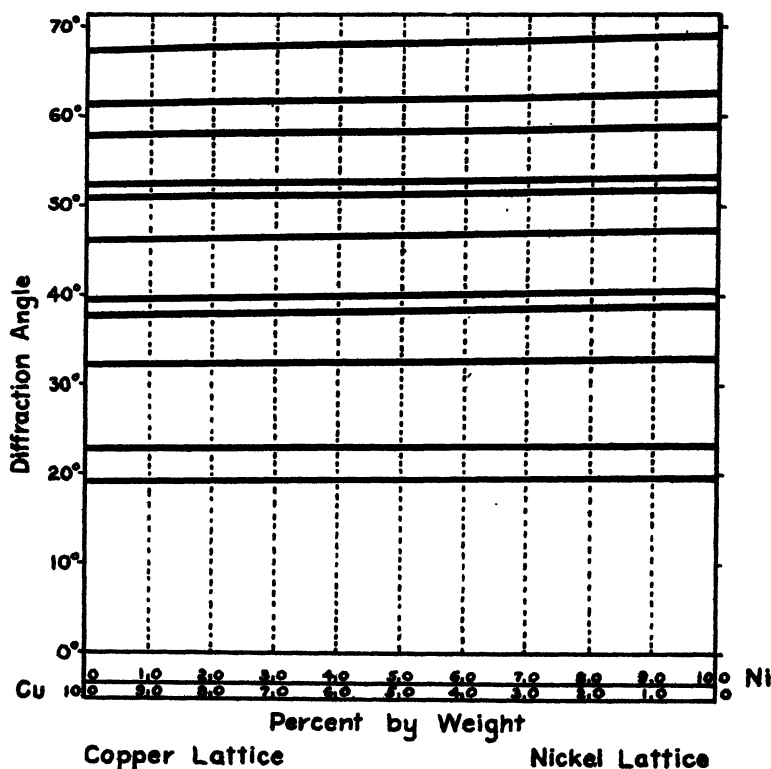
FIG. 8.—CONSTITUTION-PATTERN DIAGRAM FOR COPPER-NICKEL SERIES.

Why the stretching or shrinking of the parent lattice in the continuous solutions of the widely differing crystal types studied to date should have been less than our inherent inaccuracies (except perhaps in the manganese-copper series) is not known. It is very likely that tungsten and molybdenum have the single lattice structure throughout their entire range, as they are more nearly identical in structure than even nickel and copper.

If we consider the stresses set up within the single crystal grain by distortions exceeding that of ordinary elastic deformation, the extraordinary hardness of some of the solid solutions of soft metals entirely lacking a separate system of keying substances becomes easier to under-

stand. The slipping of adjacent layers of atoms must certainly be opposed by such forces.

Another series studied was that of gold and copper. Both gold and copper are face-centered cubic but they are quite different in dimensions. The cube edge of copper is 3.60×10^{-8} cm. and the same dimension in gold is 4.08×10^{-8} cm.; *i.e.*, larger by 13.3 per cent. Even half this figure is far in excess of any elastic deformation recorded. It seemed unlikely that this series would be like the copper-nickel series but it showed precisely the same behavior. The addition of 25 per cent. of gold expanded the copper lattice only slightly, while the addition of 25 per cent. of copper contracted the gold lattice only a little. Compositions between these amounts changed rapidly in lattice size with composition, always retaining the perfect face-centered cubic type. Fig. 9 shows, side by side, the spectrograms of pure copper and pure gold. Fig. 10 is the constitution-pattern diagram for the series.

Gold

Copper

FIG. 9.—SPECTROGRAMS OF COPPER AND GOLD.

A rather surprising deduction was made from some particularly well-annealed specimens of the copper-nickel and copper-gold series. It resulted from the explanation of some faint lines observed on the films corresponding to the 75–25 atomic per cent. alloys. These lines were entirely without the possibilities of a face-centered lattice. They proved to correspond to spacings just double some of the normal spacings of this lattice. The explanation is fairly simple. If a number of solute atoms were to distribute themselves with the utmost homogenity possible, they would arrange themselves regularly throughout the crystal grain in a manner presenting the greatest possible symmetry. The simplest way to arrange a ratio of 3:1 (75 per cent. *A* and 25 per cent. *B*) of unlike atoms is to choose alternate planes of atoms and in these alternate planes replace half of the atoms of *B* with *A* atoms. If we choose the planes of greatest symmetry in the structure for this alternate layer replacement we shall have a discontinuity of mass every other 1 1 1 plane (Miller indices). This produces a periodicity twice that of some of the normal distances. Bearing in mind that it is mass-periodicity that gives us the pattern we should expect these new lines. As the atomic weights of copper and nickel are in so close agreement, this discontinuity is extraordinarily slight and the surprising feature is that the lines are sufficiently contrasted to be visible. They were quite strong in one gold-copper pattern. This is not entirely a new phenomenon, for the extreme case of this sort of

action is observed in the pattern of silver chloride, in which the chlorine atoms are so entirely different in diffracting power from silver that only the silver spacings are revealed and the fundamental structure is inferred from the spacing of half the atoms. But so far as the author knows, it is the first occasion in which the actual atomic location of the separate metals has been shown in solid solutions. In this connection a reference should be made to the hypothetical structure of cementite composition in the austenitic condition by McAdam.[11] If such material could be isolated it could be absolutely identified with x-rays.
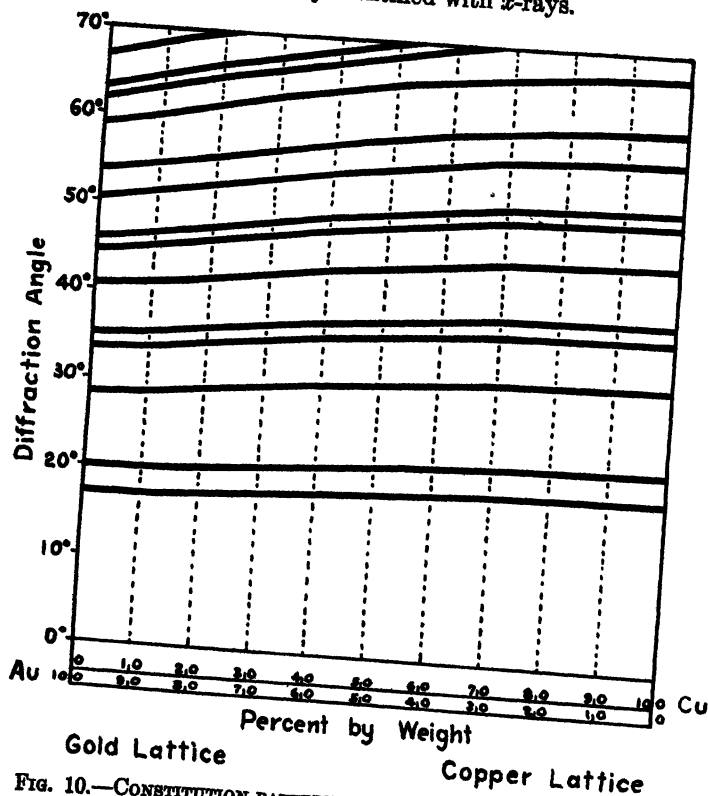
FIG. 10.—CONSTITUTION-PATTERN DIAGRAM FOR COPPER-GOLD SERIES.

SUMMARY

1. A solid solution forms by replacement of solvent atoms by solute atoms. If the pure solute is of a different crystal type from the solvent, the parent lattice changes very little in dimensions. In such cases, a limit of solubility is reached at which point a new lattice is formed which may be an intermediate or the lattice of the solute.

[11] D. J. MacAdam, Jr.: The Constitution of Martensite and Troostite. *Chem. & Met. Eng.* (Sept. 28, 1921) 25.

2. Intermediate lattices, based on weak compounds probably, are present in the complex series.

3. When two metals, alike in crystallographic type, form continuous solid solutions there may be a gradual change in lattice size from one pure metal to the other.

4. Other supposed continuous isomorphous series are actually composed, for some range, of two separate crystalline phases. Both forms are likely in the same grain.

5. Thermal treatment may shift the range of overlapping lattices toward one or the other pure metals. The effect is most noticeable if one of the pair of metals is allotropic.

6. For some constitutions of perfectly annealed solutions, the location of the solute atoms is probably known.

7. No case was observed where solid solution occurs by new atoms filling in the interatomic spaces (if such spaces exist), and it is not predicted that such cases will be found.

DISCUSSION

ANCEL ST. JOHN, Long Island City, N. Y.—Although the author refers to an alloy including manganese, he does not give the crystal form or grating space of manganese. I have searched the literature for some time but have not found that particular reference.

The statement, in conclusion 4, that both forms of two crystalline phases are likely from the same grain, may seem open to challenge, but Dr. A. W. Hull, of the General Electric Laboratory at Schenectady, has found two distinct crystalline phases existing in a single crystal of carborundum.

I would like to suggest that we cease talking about x-ray spectrograms when discussing these crystal analysis pictures, because that is just what they are not. A spectrogram, as was shown by Bassett and Davis,[12] is a diagram in which a grating, prism, or some other means of analyzing the light into its various wave lengths is used to produce a photographic record of the spectrum. Here we are using a single wave length, as

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[12] Spectrum Analysis in an Industrial Laboratory. See page 662.

nearly as we can approximate it, and are reflecting that from a number of gratings existent in the crystalline material. What we get is a "crystallogram," and as this is a new departure in crystallographic study, I propose that we refer to these pictures as crystallograms.

EDGAR C. BAIN.—I am in hearty accord with the suggestion that we adopt the name "crystallogram" for these film records. It seems to me to be a much better word than spectrogram.

I have not constructed the space-lattice for manganese, although the dominant lines of its pattern are well known. We can identify its structure when discovered but have not made a model for its atomic arrangement. For reasons not entirely clear manganese films are difficult to work with.

As for the evidence of the second crystal form within the grain of the first, since writing this paper, I have secured more conclusive proof of its existence. The x-ray crystallograph contains, to quite an extent, a measure of the grain size, and we frequently find in parallel cases a remarkably fine-grained pattern superimposed upon the coarser grained pattern of the continuous phase. As the microscope shows no regions of fine grains, they must exist within the large ones. This finding of internal discontinuities, offering interference to slip, is in excellent agreement with the hardness of such alloys.

· PAUL D. MERICA, Bayonne, N. J. (written discussion).—The discoveries of Laue and the Braggs have provided metallography with a new tool which, in the hands of Bain and others in this country, is yielding results that besides vastly increasing our information concerning metals and alloys may ultimately alter our knowledge concerning them. By that I mean that we have been accustomed to regard metals in terms of chemical composition, microstructure, and mechanical tests. If the x-ray method fulfills its promise, we may shortly begin to regard them in terms of atoms and atomic forces. I hope that presently more investigators may be in a position to adopt this method of study.

If the author will turn the equation on page 630 upside down he will have derived from atomic considerations an empirical law of a certain degree of validity; namely, that the specific volume of solid solutions (as well as mechanical aggregates) is a linear function of composition by weight. This law is generally stated in textbooks on the subject and some experimental evidence is given illustrating it. His own proof of it is especially good; it is rare that we can give proofs of such precision. It will be noted that his work shows that this law cannot hold in the case of solid solutions, such as copper-nickel, in which the lattice constant changes continuously with composition. In describing the formation of new lattices, it is evident that, in general, a compound is formed with a different type of lattice from that of the components and therefore

will generally have a density not calculable from its percentage composition.

The author states that the nickel-chromium series was chosen as an example of continuous solid solution without minimum melting point. I am forced to conclude that he is in possession of some unpublished information on this series as the only reference with which I am familiar on this subject is that of Voss, who shows a minimum melting point of 1290° C. at 40 per cent. nickel. This composition is not far from that at which simultaneous overlapping patterns are obtained. Although I have not examined this series of alloys very closely I have observed indications of a second phase in some nickel-chromium alloys, which I have always found difficult to explain. The photographs of Voss also showed traces of this same second phase.

In the case of the nickel-iron equilibrium, it is well known that there is a region of limited solubility of beta nickel in alpha iron between, roughly, 10 and 30 per cent. by weight of nickel. The natural meteors nicely illustrate this segregation. It is most interesting that this range also agrees with that of the overlapping patterns of alpha iron and beta nickel. It is generally difficult to obtain evidences of two phases in these alloys by the ordinary metallographic methods but it has been done.

It would be valuable to seek in the sealloys for the evidences of binary-phase structure and to correlate it with the interpretations of the x-ray patterns. Similarly a study of the hardness and ductility, as well as density of these solutions, in connection with the x-ray results, should yield most valuable information and can hardly fail to throw light upon the much discussed subject of hardness and plastic flow.

Would it be possible with the present x-ray method to detect atom-piling distortion in grains, such as might conceivably be produced in the immediate vicinity of colloidal particles of cementite in steel or of $CuAl_2$ particles in duralumin?

Our speculations on the subject of ductility are not simplified by observation of the following series of facts: Substitution of zinc atoms for copper with unchanged lattice and lattice constant gives high brass with undiminished (at least) ductility; this we might expect. Substitution of chromium atoms for nickel, up to 50 per cent., gives nickel-chromium alloys with diminished ductility; but the lattice is also unchanged. Substitution of copper atoms for nickel atoms, up to 50 per cent., gives copper-nickel alloys with substantially undiminished ductility; the lattice type is unchanged but the constant is changed and in a sense the lattice distorted or strained.

It is evident that this new method will have much the same history as the older microscopic one; it brought the solution of many age-old and .vexatious problems with ease, but it also brought a new set of problems more difficult than the old ones. But this is the course of progress.

Occurrence of Blue Constituent in High-strength Manganese Bronze

BY E. H. DIX,[*] JR., M. E., M. M. E., DAYTON, OHIO

DURING an investigation of high-strength manganese bronze by the Engineering Division of the Air Service, at McCook Field, Dayton, Ohio, particles of a "blue constituent" were noted in the microstructure of most of the bronzes examined.

The constitution of the simple copper-zinc base alloy is still a matter of dispute in spite of the fact that the copper-zinc series is one of the oldest industrially important groups of alloys, and has received the early consideration of scientific research. The diagram, according to the most recent work,[1] is shown in Fig. 1. The disputed questions involve the termination of the beta field and the line at 470° C. The earliest experimenter, Roberts-Austen, detected an arrest in some of his cooling curves at approximately this temperature and ascribed them to the formation of a eutectic; it has since been proved that there are no eutectics in this series. Shepherd[2] was unable to detect any thermal arrests at this temperature; his diagram was essentially as shown in Fig. 1, except that the line at 470° C. was omitted and the beta phase carried down to normal temperatures. Later, Carpenter and Edwards,[3] by a very painstaking research, definitely established the authenticity of the line but attributed it to the breakdown of the beta constituent into a eutectoid of alpha and gamma. Microscopic evidence of the formation of the eutectoid is doubtful and it seems more likely to be due to an allotropic change in the beta constituent as shown by Fig. 1. At least for practical purposes, it is more convenient to think of the structure of a 60–40 brass as a mixture of alpha and beta prime crystals rather than alpha plus an unresolvable eutectoid of "apparent beta" as it has often been called.

[*] Chief, Metals Branch, Engineering Division, Air Service.

[1] O. F. Hudson: *Jnl.* Inst. of Metals (1914) **12**, 89.

[2] E. S. Shepherd: *Jnl. Phys. Chem.* (1914) **8**, 421.

[3] H. C. H. Carpenter and C. A. Edwards: *Jnl.* Inst. of Metals (1911) **5**, 127.

The effect of the addition of a third element to the useful alloys of the copper-zinc series has been studied by a number of experimenters. Hudson and Jones[4] have given the useful diagram for the tin brasses shown in Fig. 2. A more complete, though much more complicated, ternary diagram had previously been established by Hoyt.[5] In describing the structure of the brasses examined, Hudson and Jones say, "The beta prime appears bright yellow, the alpha is reddish yellow, and the delta (or gamma) clear light blue." Desch, in a discussion of their paper,
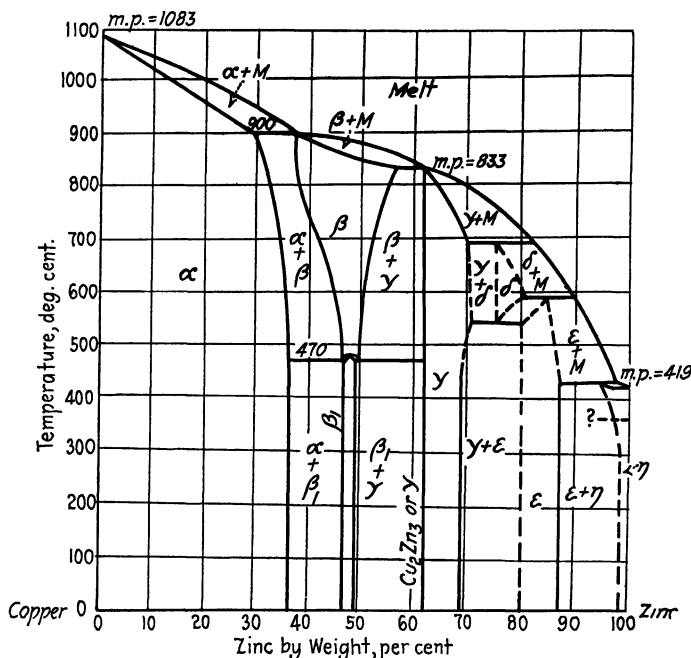
Fig. 1.—Equilibrium diagram of copper and zinc. (*After Hudson.*)

suggested that "as no difference had been found between the gamma constituent of the brasses and the delta of the bronzes, it seemed probable that these two substances were isomorphous, this being the view taken in Professor Hoyt's paper, and a similar isomorphism having been observed between the gamma constituent of the zinc-copper and aluminum-copper series, respectively, by Carpenter and Edwards. As it was a familiar fact that brasses were rendered brittle by even traces of the gamma crystals, whilst bronzes could tolerate fairly large proportions of delta, it would be interesting to know at which point brittleness made its appearance in the ternary series. Probably delta was harmless when

[4] O. F. Hudson and R. M. Jones: *Jnl.* Inst. of Metals (1915) **14,** {

[5] S. L. Hoyt: *Jnl.* Inst. of Metals (1913) ˙**10,** 235.

accompanying alpha, whilst gamma was usually harmful when accompanying beta prime; yet some of the alpha-beta prime alloys containing the 'blue constituent,' could be made use of."

The effects of some other elements, such as aluminum, iron, and manganese, have not been so thoroughly worked out. We are, however, indebted to Leon Guillet[6] for an ingenious scheme for converting these ternary alloys to equivalent copper-zinc binary alloys. He has found that these elements, when added in small amounts, go into solid solution and have the same effect on the microstructure as the addition of more zinc. Thus he has conceived of a fictitious zinc percentage calculated from the "coefficient of equivalence" for each element. These coefficients are: aluminum, 6; tin, 2; iron, 0.9; manganese, 0.5. The total value of zinc thus obtained is recalculated with the known copper content to give 100 per cent. Thus a 60–40 brass in which 2 per cent. of the zinc has been replaced by as much aluminum will have a fictitious composition of 54.5 per cent. copper and 45.5 per cent. zinc, and the relative amounts of alpha and beta constituents in the annealed brass will conform closely to what would be expected for this composition, about 10 per cent. alpha. When the solubility limit was exceeded, a special constituent appeared. Thus, in the annealed specimens, the special constituent appeared with
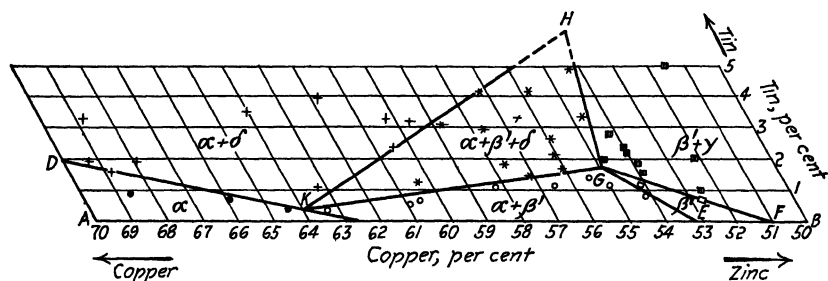
FIG. 2.—TERNARY DIAGRAM FOR COPPER, ZINC, AND TIN. (*After Hudson and Jones.*)

more than 7.1 per cent. aluminum in a brass containing 69 per cent. copper; with more than 8 per cent. manganese in a 60-per cent. copper alloy; and with more than 0.7 per cent. tin in a 63-per cent. copper alloy. He denotes the third phase by gamma, which he suggests may be constituted by a definite compound, a solid solution, a eutectic or a double or triple eutectoid. He further states, in discussing the tin brasses in a later paper,[7] that "the tin brasses contain a special con stituent which bears a strange resemblance to the delta of the bronzes (a solution closely corresponding to the compound Cu_4Sn). This constituent is accompanied

[6] *Rev. de Métallurgie* (1905) **2**, 97; (1906) **3**, 243; (1920) **17**, 561. Abs. *Chem. Met. Eng.* (Jan. 26, 1921) **24**, 177; (Apr. 6, 1921) **24**, 609.

[7] *Rev. de Métallurgie* (July, 1921) **18**, 445; Abs. *Chem. & Met. Eng.* (Nov. 30, 1921) **25**, 1009.

by hardness and brittleness; it injures the mechanical properties of the alloy, especially when present in sufficient amount. Annealing for 2 hr. at 800° C. does not cause this special constituent to disappear if the tin content is over 1 per cent. Quenching from either 600° or 800° C. causes the special constituent to disappear, but the 800° C. quench gives a low elongation, whereas the 600° C. quench improves both the tensile strength and elongation." He states that the special constituent forms a white rim around the alpha.

Millington,[8] who has also studied the effects of these elements on a 60–40 base alloy in which some of the copper was replaced by the third element, reached the following conclusions: (a) 1.11 per cent. tin give no other than the ordinary alpha-beta structure, but with 2.13 per cent. tin the three phases, alpha, beta, and delta were observed; (b) 2.8 per cent. manganese failed to produce any evidence of the third constituent; (c) 1.06 per cent. aluminum gave a structure of acicular cystallites in large beta crystals, giving every evidence of brittle material; 1.92 per cent. aluminum revealed the blue constituent in a beta field; (d) 1.2 per cent. iron does not produce any blue constituent, but this constituent was noted with the presence of 1.88 per cent. iron. These results were obtained on sand castings without any annealing. In regard to this special constituent he says, "Owing to the similarity in appearance of the two constituents gamma and delta when viewed through the microscope, it is difficult to distinguish between them with certainty in a ternary alloy, since they both present a bluish color after etching and for this reason will be referred to as the blue constituent." He further states that, "The addition of any of the above metals is accompanied by an increase in strength and a decrease in elongation even before the appearance of the blue constituent, but the decrease in elongation becomes very great as the blue constituent begins to appear."

Campbell[9] has confirmed the alpha, beta, delta field of the ternary diagram, Fig. 2, as given by Hudson and Jones for the annealed tin brasses, with the exception of minor differences in the location of the points of the triangle, and has further indicated these points for the alloys as cast. His location of these points is:

		TIN, PER CENT.	ZINC, PER CENT.	COPPER, PER CENT.
1	As cast	4.5	37.5	58.0
2		2.0	41.5	56.5
3		0.2	31.4	68.4
1	Annealed	4.0	39.0	57.0
2		1.4	43.6	55.0
3		0.25	35.5	64.25

[8] E. Millington: *Jnl. Soc. Chem. Ind.* (1918) **37**, 149T.
[9] William Campbell: A. S. T. M. (1920, Part II), 105.

He also gives the following limits for the phases of the copper-zinc binary alloys:

	PER CENT. ZINC AS CAST	ANNEALED
Alpha	0–32.0	0–36.5
Alpha-beta	32.0–47.0	36.5–45.5
Beta	47.0–49.0	45.5–49.0

He further states that "the delta of the copper-tin series, or Cu_4Sn, and the gamma of the copper-zinc series, or Cu_2Zn_3, form a single series of solid solutions; hence, in the ternary alloys, rich in both tin and zinc, it is impossible to distinguish delta from gamma."

Gulliver[10] states that in certain complex brasses the iron is combined with, or dissolved in, some of the zinc, the resulting constituent appearing as small blue globules distributed through the alloy. He further says that iron has comparatively little effect on brass, slightly increasing its strength and hardness.

Manganese bronze consists of a 60–40 copper-zinc base alloy, to which is added small quantities of each of the elements, tin, aluminum, iron, and manganese. Usually the percentage of any one of the elements added is so small that, if added alone, it would be taken into solution and the only effect on the structure would be to give more beta prime and less alpha than would be expected for the true zinc content. This effect can be satisfactorily estimated by the use of Guillet's "fictitious zinc" calculation. However, when even these small percentages of the four elements are added in the same alloy, the joint effect often produces small regular particles of a third constituent of a characteristic blue color. That these particles are much harder than the alpha-beta prime matrix is shown by an examination of the polished surface unetched. They stand out in relief. Contrary to the findings of the experiments previously reviewed, in which the appearance of a special constituent was accompanied by a severe reduction in the elongation, a moderate amount of this constituent is found in ductile manganese bronze (having an elongation of at least 30 per cent.) and a considerable amount is usually present in particularly high-strength manganese bronze (over 80,000 lb. per sq. in. tensile strength).

The results of the examination of a number of melts of manganese bronze made in the McCook Field foundry are compiled in Table 1. Specimens 257–3 and 1108–2 were made from ingot purchased outside, the latter melt having the zinc content reduced by many remeltings; specimens 431–2 and 435–3B were from ingot purchased from another concern. The remaining specimens were taken from experimental melts mixed in the foundry. In those containing less than 0.35 per cent. iron, each element was added separately, except the manganese which was

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[10] G. H. Gulliver: "Metallic Alloys," 279. Charles Griffen & Co., London.

TABLE 1.—*Results of Tests*

	Chemical Analysis							Physical Properties			Metallographic Data			
Specimen No.	Per Cent. Copper	Per Cent. Zinc	Per Cent. Fictitious Zinc	Per Cent. Aluminum	Per Cent. Tin	Per Cent. Manganese	Per Cent. Iron	Tensile Strength, Lb. per Sq. In.	Per Cent. Elongation in 2 In.	Brinell, 500 Kg.	Character of Alpha	Per Cent. of Alpha	Amount of Blue Constituent	Grain Size
175-3	57.10	40.30	45.5	0.78	1.13	0.56	0.13	77,960	18.5	108	Small	30	Scarce, fine	Medium
207-1	57.51	40.85	43.6	0.21	1.06	0.12	0.25	72,990	9.5		Fine	15	Scarce, fine	Small
257-3	57.10	40.35	45.2	0.75	0.73	0.10	0.97	70,890	18.5	96	Medium	35	Some	Fine
431-2	57.86	39.42	44.9	0.92	0.46	0.03	1.31	70,210	30.0	105	Medium	40	Some	Fine
435-3B	58.28	39.46	44.0	0.75	0.51	0.05	0.90	70,260	36.5		Medium	40	Some	Fine
460-5	58.10	40.76	43.1	0.26	0.58	0.03	0.27	59,690	45.5		Coarse	50	None	Large
491	57.83	41.11	43.6	0.47	0.27	Trace	0.30*	67,620	33.3	100	Medium	50	None	Medium
527-3	56.90	40.00	44.8	0.50	1.00	0.55	1.05*	67,010	22.0	156	Medium	60	Much	Fine
540-1	56.62	40.54	46.0	0.99	0.58	0.47	1.14	83,770	22.5	156	Fine	20	Much	Fine
568	56.67	40.15	45.1	0.63	0.43	0.41	1.31	83,380	19.5	120	Fine	15	Much	Fine
597-1	56.67	39.64	45.4	0.74	0.53	0.50	1.41	83,670	21.5	130	Fine	20	Much	Fine
603-1	57.41	39.85	45.6	1.16	0.33	0.39	1.09	70,590	13.0	143	Small	30	Much	Fine
608-1	57.18	38.97	44.8	0.65	0.55	0.47	1.30	83,530	23.2	156	Fine	20	Much	Small
622-4B	58.59	38.59	43.8	0.81	0.43	0.14	1.06	62,540	30.0	93	Medium	45	Much	Medium
972-1	57.02	39.80	46.2	1.10	0.48	0.10	1.50	77,920	14.0	117	Fine	20	Much	Fine
976	Remelt of 972							79,050	14.0	119	Fine	25	Much	Fine
979	Remelt of 972							77,430	15.0	109	Fine	25	Much	Fine
1028-3	Remelt of 972							77,410	14.7	108	Small	35	Much	Fine
1056-2	56.4	37.50	51.8	3.3	0.59	0.13	2.0	67,170	3.5	101	Fine	10	Much	Small
1057-A	Remelt of scrap							70,480	23.5	90	Medium	50	Much	Small
1108-2	59.87	37.48	41.5	0.35	0.75	0.08	1.47	64,040	33.5	80	Coarse	60	Some	Fine
1130	56.54	40.69	45.9	0.87	0.40	0.11	1.39	69,320	2.7	114	Fine	10	Much	Small
1234-2	59.10	38.26	43.4	0.85	0.47	0.10	1.22	65,340	13.0	110	Medium	45	Much	Small
1249-2	Remelt of 1234							64,800	17.7	100	Medium	45	Much	Small
1260	56.94	39.68	45.6	0.93	0.57	0.49	1.39*	74,010	8.5	130	Fine	15	Much	Small
1270-1	61.70	35.00	41.6	0.97	0.50	0.62	1.27*	61,580	34.0	93	Coarse	70	Much	Small
1272-1	59.05	37.50	43.6	0.97	0.50	0.62	1.27*	66,960	26.0	100	Medium	60	Much	Small
1285-2	59.85	38.75	45.4	0.97	0.50	0.62	1.27*	64,200	14.5	105	Medium	50	Much	Small
1289-2	57.25	39.37	45.4	0.97	0.50	0.62	1.27*	65,700	10.5	109	Small	40	Much	Small
1291-1	57.79	39.06	44.1	0.90	0.50	0.48	1.18	67,800	10.2		Medium	35	Much	Fine
1301-1	57.62	39.00	45.1	1.00	0.47	0.61	1.25*	66,750	9.5		Medium	35	Much	Small
1365-1	54.22	42.60	48.2	0.92	0.0	0.59	1.20*	62,750	4.0	114		0	None	Fine
1368-1	55.00	45.00	45.00	0.0	0.0	0.0	0.0*	47,870	8.5	130	Fine	15	Abundant	Small
1378-1	57.14	40.00	45.3	0.97	0.0	0.62	1.27*	60,260	11.5	96	Small	35	Much	Large
1382-2	50.00	50.00	50.0	0.0	0.0	0.0	0.0*			97		0	None	Large

* No analyses made, composition as mixed.

added in the form of a 30-per cent. manganese copper; no iron was intentionally added except a small amount in the manganese copper. For the other melts, a hardener[11] consisting of copper, manganese, iron, and aluminum was first made up and melted with the copper, the zinc being preheated and added in small pieces. The melting was done in plumbago crucibles in an oil-fired furnace.

The test bars were cast in green sand, the metal being led into the end of the bar by means of a long skim gate. The test specimen was ⅝ in. (15.87 mm.) in diameter over the gage length of 2 in. (50.8 mm.) with shanks ¾ in. (19.05 mm.) in diameter. This was machined to 0.505-in. (12.83 mm.) gage diameter with ends threaded for self-aligning adapters. It was fed by a wedge riser extending within ½ in. (12.7 mm.) of each end. This riser was joined to the test bar by a ¼-in. (6.35 mm.) web, above which it tapered to a width of 2⅜₆ in. (55.56 mm.). The height of the cope used was 3 in. (76.2 mm.). This method was found to give uniform and dependable results that compared favorably with those obtainable in actual castings. As an example, compare the results from melt 972 with its remelts. All of the melts were poured between 1700° and 1775° F. (927° and 968° C.) with a maximum furnace temperature of 1800° F. (982° C.). The temperature was measured by a chromel-alumel couple encased in a chromon-tip protecting tube used in conjunction with a high-resistance millivoltmeter.

For obtaining the metallographic data, a specimen was cut from the end of each test bar and polished in the ordinary way, except that the final polishing was done by means of an automatic device[12] which greatly reduced the time required for hand polishing. It was then examined at 500 magnification for the amount of the "blue consituent," after which it was etched, by swabbing with a fresh solution of approximately one part H_2O_2 in nine parts NH_4OH. For photographic purposes, it was necessary to etch rather deeply in order to distinguish between the light alpha and the blue constituent. With this etching, the beta prime is a clear bright yellow, the alpha a mottled yellow or a reddish yellow, and the blue constituent is unattacked except by very prolonged etching, when it begins to dissolve around the edges. The specimens were examined at 100 magnification, in order to study the character of the alpha and to estimate the percentage present; the comparative grain size was also estimated at this magnification. For the specimens containing the blue constituent, 500 was found to be a satisfactory magnification for comparison. For a careful detailed examination of the blue constituent a magnification of 1000 was obtained by means of a Bausch & Lomb 1.9-mm. oil-immersion fluorite objective.

[11] See E. H. Dix, Jr., Methods of Casting Manganese-bronze Test Bars as a Check on Melts of Small Castings, A. S. T. M., 1921.

[12] Designed and patented by E. V. Schaal, McCook Field, Dayton, Ohio.

Figs. 3 to 11 show three bronzes of approximately the same chemical composition at the three magnifications. According to Table 1, specimen 540-1 is a good example of high-strength manganese bronze; specimen 1260 is weaker by 10,000 lb. per sq. in. and its elongation is too low to meet even the easiest specification; while specimen 1130 is still weaker and extremely brittle. A comparison of the structures at 100x shows that the amount of alpha constituent decreases from approximately 20 per cent. in 540-1 to 15 per cent. in 1260, and to 10 per cent. in 1130. ' The character of the alpha constituent is the same, being recorded as "fine" in the table; the grain size of the original beta constituent is also about the same in all three. The structures at 500x reveal considerable blue constituent (white regular particles in photographs) in all three specimens. The photographs of specimens 540-1 and 1260 at this magnification were made with a blue filter; therefore the blue constituent shows a clearer white than in the photograph of specimen 1130 but there should be no difficulty in recognizing this constituent in the latter photograph. In a visual examination, this constituent is a clear blue that is instantly recognized. A careful visual examination over the whole area showed that specimens 540-1 and 1130 had about equal amounts of the blue constituent but a slightly greater amount was present in specimen 1260. The photographs at 1000x reveal the blue constituent very clearly. It will be noted that these particles are closely associated with the alpha constituent and usually are embedded in it. Certainly the structure does not give any evidence that the brittleness of specimens 1260 and 1130 is due in any way to the presence of the blue constituent. Is the difference in the percentage of the alpha constituent present in the three sufficient to account for the difference in elongation? In mixing these three melts, the same chemical composition was aimed at and the actual chemical analyses obtained are very close; the fictitious-zinc calculations give 46.0 per cent. for specimen 540-1, 45.9 per cent. for specimen 1130, and 45.6 per cent. for specimen 1260. Melts 1130 and 1260 were made from a different lot of electrolytic copper and from a slightly less pure spelter than melt 540. The analyses of these materials are:

		MELT 540, PER CENT.		MELTS 1130 AND 1260, PER CENT.
Zinc	Pb	nil	Pb	0.22
	Fe	0.05	Fe	0.04
	Cd	nil	Cd	0.50
	Zn	99.95 (Dif.)	Zn	99.24 (Dif.)
Copper	Cu	99.80	Cu	99.79

That the amount of alpha constituent present cannot exert so great an effect on elongation is shown by specimen 1301-1, which contains
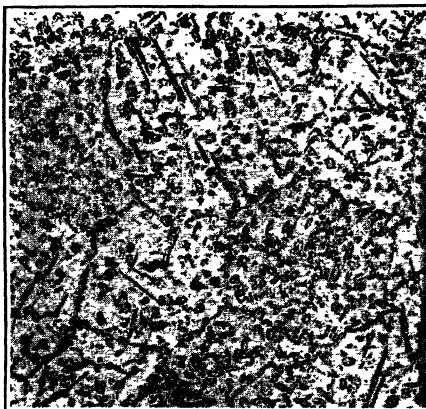
FIG. 3.—SPECIMEN 540–1.   × 100.   FIG. 4.—SPECIMEN 540–1.   × 500.
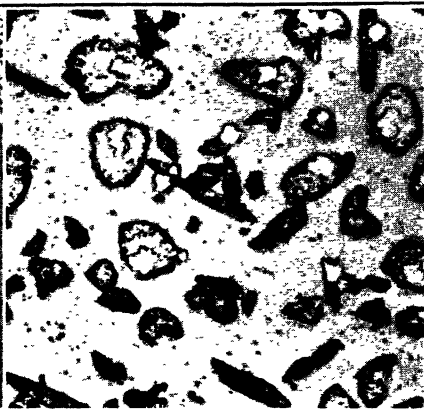
FIG. 5.—SPECIMEN 540–1.   × 1000.   FIG. 6.—SPECIMEN 1260.   × 100.
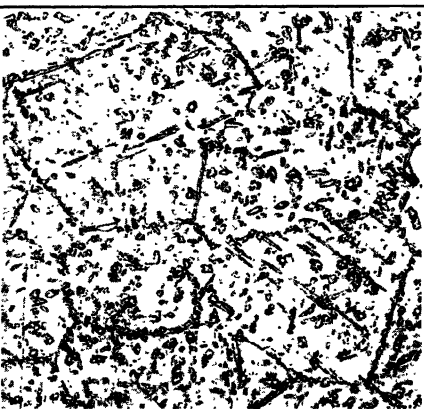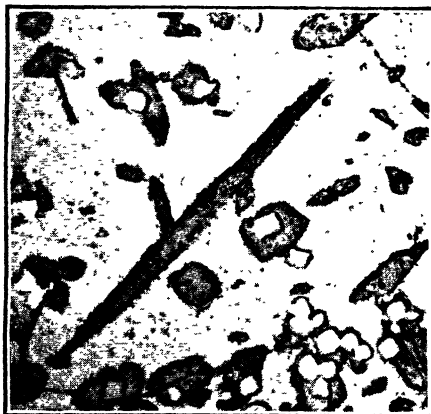
FIG. 7.—SPECIMEN 1260.   × 500.   FIG. 8.—SPECIMEN 1260.   × 1000.
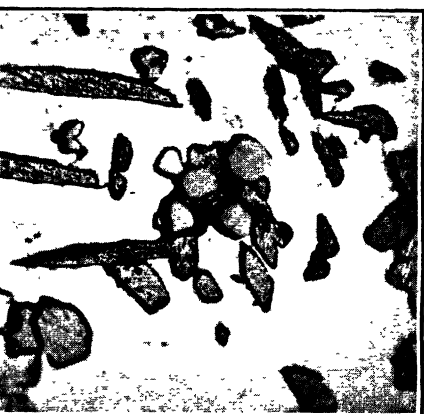
All specimens etched with $NH_4OH + H_2O_2$.   Specimen 540–1, tensile strength, 83,770 lb. per sq. in.; elongation, 22.5 per cent.; Brinell, 156.   Specimen 1260, tensile strength 74,010 lb. per sq. in.; elongation, 8.5 per cent.; Brinell, 130.

slightly less zinc than the three previously considered and consequently shows approximately 35 per cent. alpha. However, the elongation of this melt was only 9.5 per cent. and as this was made from the same copper and zinc as the melts 1130 and 1260 it seems reasonable to conclude that low elongation is due, at least in part, to the different grade of raw material used. Numerous references in the practical literature substantiate this view and it is regretted that a more complete analysis was not made of the raw materials used in these experiments.

The effect of the true zinc content on the microstructure is seen by comparing Figs. 7, 12, 13, 14, and 15. The percentage of hardening elements in these five specimens is approximately the same. Melt 1260 was first made with 40.0 per cent. zinc and melt 1270 with 35 per cent. zinc. Equal weights of these two melts were then melted to give melt 1272 with 37.5 per cent. zinc. This last melt was melted with an equal weight of 1260 to give melt 1285 with 38.75 per cent. zinc. Sufficient zinc was then added to give 42.6 per cent. in melt 1365. The percentage of blue constituent was not appreciably altered by the change in the zinc content, but the character and amount of the alpha constituent was greatly affected. There was no alpha constituent present in melt 1365. The effect on the physical properties was an increase in elongation, which was directly proportional to the decrease in the zinc content; the tensile strength and hardness were only slightly affected.

Fig. 16 shows the structure of a very ductile bronze in which the percentage of total hardening constituents is low and no blue constituent is formed. The elongation of this specimen was 45.5 per cent. but the tensile strength was less than 60,000 lb. per sq. in. The alpha constituent is present in large amount and is coarse. Specimen 491 is the only other bronze that did not contain some blue constituent. In specimens 175–3 and 207–3, the blue constituent was scarce and very fine. Specimen 175–3 represents good physical properties, although both tensile strength and elongation are less than those of melt 540. None of the other melts possesses particularly desirable properties.

Remembering the work done on tin brasses, in which the occurrence of a blue constituent had been described by a number of experimenters, it was decided to make up a melt of manganese bronze containing no tin in order to be sure that the blue constituent was formed by the presence of other elements besides tin. Thus, melt 1378 was made up of the same percentages of aluminum, manganese, iron, and zinc as melt 540 but the tin was omitted. The cast structure was found to contain a larger amount of the alpha constituent than melt 540, but about the same amount of blue constituent. Another melt was made up containing 8 per cent. tin, 8 per cent. zinc, $2\frac{1}{4}$ per cent. manganese, and $81\frac{3}{4}$ per cent. copper, and in this a network of the delta constituent was obtained. This delta constituent, unetched, was a paler blue than the blue constituent in
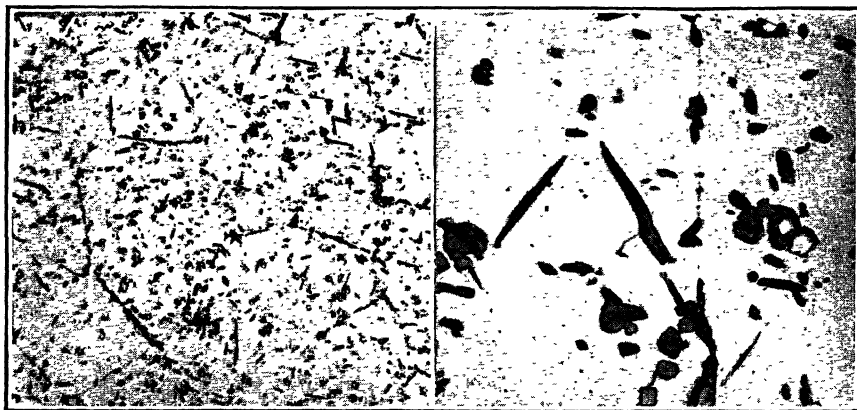
FIG. 9.—SPECIMEN 1130. × 100.

FIG. 10.—SPECIMEN 1130. × 500.
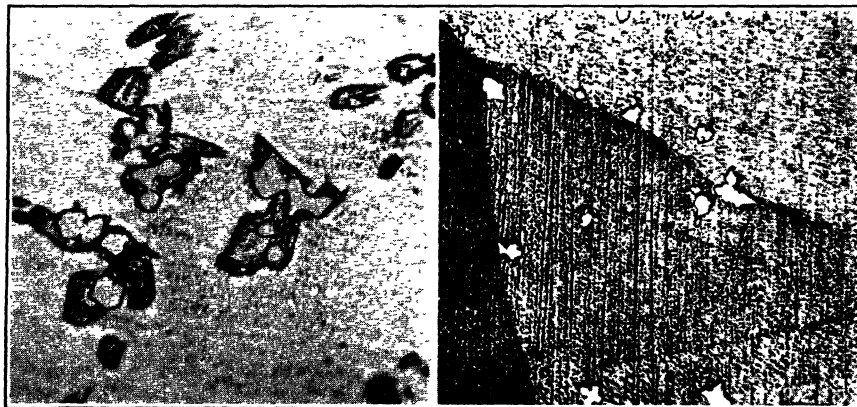
FIG. 11.—SPECIMEN 1130. × 1000.

FIG. 12.—SPECIMEN 1365–1. ZINC 42 PER CENT. × 500.

FIG. 13.—SPECIMEN 1285–2. ZINC 38.75 PER CENT. × 500.

FIG. 14.—SPECIMEN 1272–1. ZINC 37.50 PER CENT. × 500.

All specimens etched with $NH_4OH + H_2O_2$. Specimen 1130, tensile strength, 69,320 lb. per sq. in.; elongation, 2.7 per cent.; Brinell, 114.

manganese bronze and was always present as a network structure and never formed the regular particles found in manganese-bronze. However, its reaction with the etching reagent was similar to that of the blue constituent of manganese-bronze.

To test out Gulliver's hypothesis regarding the iron-zinc compound or solution, an attempt was made to dissolve 25 per cent. iron in molten zinc. The iron was added as thin sheet, but after holding the zinc at as high a temperature as possible without too great a loss from volatilization for 2 hr., it was found that only one-fifth of the iron had been dissolved. The resulting alloy was hard and brittle. When polished and examined microscopically, it did not show any particles that resembled the blue constituent in manganese bronze. However, when 40 per cent. of this alloy was added to 60 per cent. molten copper, the microstructure of the resulting brass showed an abundance of the blue constituent.

An alloy of 20 per cent. iron and 80 per cent. copper was made by melting the two metals together in a small electric induction furnace. The metals were thoroughly mixed in the molten state and the resulting alloy was very fluid. When poured into iron ingot molds however, large globules, probably of the iron-rich solid solution, liquated to the top of the ingot. A microsection revealed numerous particles resembling the blue constituent in manganese bronze; these were evidently composed of the same iron-rich solid solution that had segregated in the top of the ingot. This alloy was remelted and sufficient zinc added to give a resulting mixture of 57.5 per cent. copper, 39.8 per cent. zinc, 2.7 per cent. iron. A polished section of this alloy revealed a few particles as large as pinheads and numerous specks just visible to the naked eye. A microscopic examination showed in addition an abundance of particles of approximately the same size as the blue constituent in manganese bronze; both large and small particles had the same characteristic blue color. The writer has also observed similar microscopic particles of the characteristic shape and color in aluminum bronze of the following composition: 86 per cent. copper, 10 per cent. aluminum, and 4 per cent. iron. Thus it appears that iron is capable of forming a constituent with copper, without the presence of zinc, which resembles the blue constituent in manganese bronze.

However, hardeners for manganese bronze containing approximately 50 per cent. copper, 25 per cent. iron, 10 per cent. tin, 10 per cent. aluminum, and 5 per cent. manganese do not contain any blue constituent. Further, it has been found that when this hardener is added to copper in the process of making manganese bronze, no blue constituent is found in a sample taken just before the zinc is added. This alloy, which has approximately the following composition—95.7 per cent. copper, 2.15 per cent. iron, 0.86 per cent. aluminum, 0.86 per cent. tin, and 0.43 per cent. manganese—consists of a single solid solution. When the zinc is added

to give a composition of 55.5 per cent. copper, 42.0 per cent. zinc, 1.25 per cent. iron, 0.50 per cent. aluminum, 0.50 per cent. tin, and 0.25 per cent. manganese, considerable blue constituent is present.   Thus, the zinc is responsible for the precipitation of the blue constituent from the copper solid solution.

Several melts were then made up containing a little more tin and aluminum than is usual in manganese bronze but without any iron.   Careful examination at high power failed to reveal any blue constituent.   The other melts given in Table 1 in which the blue constituent is either scarce or entirely absent are all low in iron (below 0.30 per cent.).

From this, it seems safe to conclude that in manganese bronze of the usual composition (aluminum and tin under 1 per cent., iron under 1.5 per cent., and manganese under 0.5 per cent.) the occurrence of the blue constituent is due principally to the presence of iron.   Thus, the manner of the occurrence of this constituent is explained.   For, because of its high melting point, it first forms in the liquid as minute globules, which coalesce until the alpha constituent begins to form around them.   It has been noted that the particles of blue constituent are usually embedded in the alpha constituent, although some may remain to be entrapped in the beta.

In order to compare the structure of the manganese bronzes with that of yellow brass of a similar zinc content, two melts were made up.   Melt 1368, having a zinc content of 45 per cent., which is close to the fictitious content of melt 540, and another, melt 1382, having 50 per cent. zinc, which is slightly higher than the fictitious zinc content of melt 1365. Melt 1368, as cast, was found to have a simple alpha-beta prime structure with approximately 15 per cent. alpha.   Melt 1382, as cast, contained no alpha constituent but a considerable quantity of gamma; its structure is shown in Fig. 17.   This specimen was etched in the same manner as the manganese bronzes but a very light application of the etching reagent tarnished the gamma constituent and a slightly deeper etch dissolved it, leaving the black areas shown.   This is particularly interesting inasmuch as the blue constituent of the manganese bronzes was dissolved only after prolonged etching and then only at the boundaries, the surfaces remaining a clear untarnished blue.   An examination of the gamma constituent before etching showed it to be a slightly paler shade of blue than the blue constituent of the manganese bronze.

Two specimens of melt 540 were heated for 7¾ hr. at 1560° F., ± 10°. One specimen was water quenched from this temperature and the other allowed to remain in the furnace with the power cut off.   The pyrometer record showed that the furnace cooled rapidly for the first few hours and then much more gradually until after 8 hr. there was very little slope to the curve.   After 16¼ hr. the temperature had dropped to 480° F. and this specimen was taken out and water quenched.   The structures of
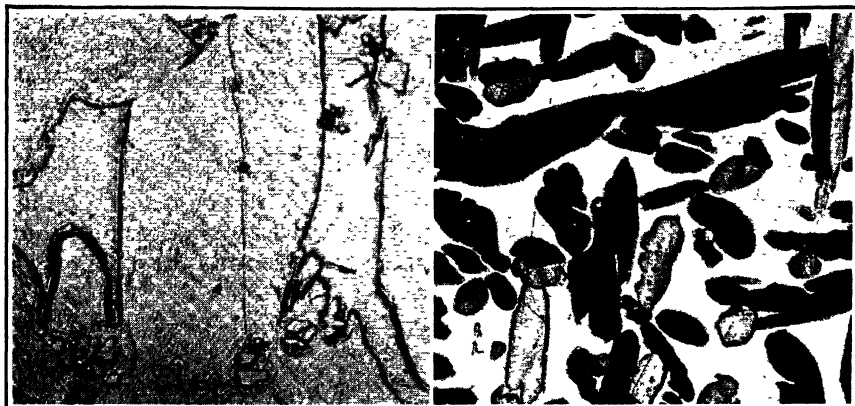
FIG. 15.—SPECIMEN 1270–1. ZINC, 35.0 PER CENT. × 500.

FIG. 16.—SPECIMEN 460–5. ELONGA-TION, 45.5 PER CENT. × 100.
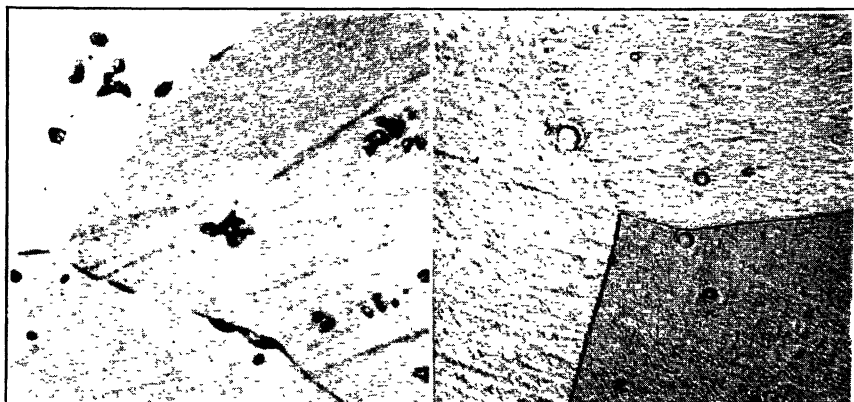
FIG. 17.—SPECIMEN 1382–2. COP-PER, 50 PER CENT.; ZINC, 50 PER CENT. × 500.

FIG. 18.—SPECIMEN 540–1. WATER QUENCHED FROM 1560° F. × 500.
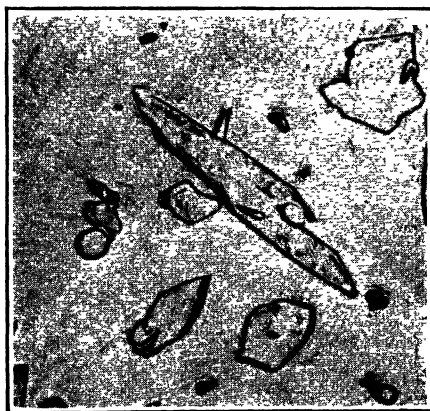
FIG. 19.—SPECIMEN 540–2. SLOWLY COOLED TO 480° F. × 500.
All specimens etched with NH₄OH + H₂O₂.

these two specimens are shown in Figs. 18 and 19. The specimen quenched from 1560° F. showed grains of the beta constituent in which were embedded rounded particles of the blue constituent. The specimen cooled to 480° F. and water quenched showed coalesced particles of the alpha and the blue constituent in the matrix of beta prime. The amount of the blue constituent in these two specimens was slightly less than in the specimen as cast and the particles much more rounded (compare with Fig. 4). No difference in the matrix of Figs. 18 and 19 was discernible, but one is called beta and the other beta prime to conform to the equilibrium diagram of Fig. 1.

Three specimens of melt 1270 were heated to 1560° F. and water quenched after 2¾, 5½ and 7¾ hr., respectively, at that temperature. No difference in the amount of blue constituent present could be noted in these three specimens. A small amount of alpha was present in all three.

Two specimens of melt 1382 (50 per cent. copper and 50 per cent. zinc) were treated at the same time and in the same way as the specimens of melt 540. The specimen quenched at 1560° F. showed no gamma constituent present, whereas that cooled to 480° F. had about the same amount of gamma as Fig. 17. In a like manner, and at the same time, two specimens of melt 1368 (55 per cent. copper and 45 per cent. zinc) were treated. The specimen quenched from 1560° F. showed grains of pure beta and the one cooled to 480° F. showed the normal alpha-beta prime structure.

CONCLUSIONS

1. When the hardening elements (aluminum, tin, iron, and manganese) are added to the 60–40 copper-zinc alloy in sufficient quantities to produce a high-strength manganese bronze, a third constituent appears in the alpha-beta prime complex, which has a characteristic clear blue color.

2. The occurrence of this constituent in reasonable amounts is not necessarily accompanied by brittleness.

3. Annealing for 7¾ hr. at 1560° F., followed by either water quenching or slow cooling, does not cause the disappearance of this constituent.

4. This constituent, when examined unetched, appears to be of a deeper blue than either the delta constituent of the copper-tin series or the gamma constituent of the copper-zinc series. When etched with NH_4OH plus H_2O_2, the blue constituent is unattacked except for a slight eating away at the edges under prolonged etching, whereas the gamma constituent with the same reagent is immediately tarnished and rapidly dissolved. The characteristic shape of the blue constituent is decidedly different from that of the delta of the copper-tin alloys.

5. For practical purposes in the study of the microstructure of manganese bronzes, it seems highly desirable to regard the blue constituent

as distinct from either the delta or gamma, and it is suggested that it may reasonably be denoted as delta prime.

6. In the manufacture of manganese bronze, the particles of delta prime are precipitated from the copper-rich solution by the addition of the zinc. This precipitation is strongly influenced by the presence of iron and, therefore, the particles probably consist of a solid solution of iron and copper with or without some tin, aluminum or manganese.

The author wishes to acknowledge his indebtedness to Lieut. A. J. Lyon for his kind assistance in discussions and experimental work and to Mr. J. L. Hester for assistance in the photographic work.

DISCUSSION

GEORGE F. COMSTOCK,* Niagara Falls, N. Y. (written discussion).— The work of Gulliver[13] and Hudson[14] had led us to believe that this blue constituent was an iron-zinc compound, but the author seems to have proved that copper is necessary for its formation. Would it not be reasonable to assume, at least until further data are obtained, that this constituent is a ternary compound of copper, zinc, and iron, instead of merely an iron-copper compound precipitated from solution by zinc? The author seems to have been influenced toward the latter conclusion because of the discovery of a similar-appearing compound in aluminum-bronze containing iron but no zinc. Our experience with this iron constituent in aluminum bronze, however, has been that it is strongly attacked by the acid ferric-chloride etching solution recommended by Carpenter and Edwards, while the blue constituent in manganese bronze is not attacked by acid ferric chloride. Possibly, if the author should try this method of etching, a difference would be found between these two constituents, and the theory that the one in manganese bronze contained zinc would not be found untenable.

It has always seemed to the writer that one of the important differences between manganese bronze and a plain copper-zinc alloy, such as muntz metal, is the presence of these fine blue crystals in the former and not in the latter. The superior physical properties of manganese bronze are probably only partly caused by the simple hardening effect of the iron, aluminum, etc. in solid solution, and, to a considerable extent, by the refinement of the grain brought about by the formation, in the molten alloy, of these little crystals, which then constitute nuclei for starting the growth of numerous copper-zinc alloy crystals. Without

* Titanium Alloy Mfg. Co.
[13] See foot-note 10.
[14] Copper Alloys of High Tenacity, *Jnl.* Birmingham Metallurgical Soc., May (1919).

the blue crystals as nuclei, fewer alloy crystals are formed and their size is therefore greater, other conditions being the same.

We are not in sympathy with the suggestion to call this blue compound "delta prime," for the other Greek letters are most commonly used for phases of either the copper-tin or copper-zinc systems, and it would introduce an element of confusion into the nomenclature to apply this particular Greek letter to a substance that is primarily an iron compound. It seems more reasonable to retain the name "blue constituent," or perhaps "iron constituent," for this substance does not occur in either the copper-tin or copper-zinc series.

PAUL D. MERICA, New York, N. Y.—Some years ago, when examining various specimens of wrought and cast manganese bronze, it was found that the use of a reagent for iron, consisting of dilute hydrochloric acid and potassium ferricyanide, indicates that the blue constituent in manganese bronze may be of two varieties, one that turns blue on the addition of this reagent and one that does not. There was also in the specimens examined a marked difference in the form of these two. The blue constituent, which I think the author has principally in mind, generally occurs in the form of star-shaped crystals, whereas the one that turns blue on the addition of the iron reagent generally is of an irregular rounded shape and generally larger; the latter obviously contains a high percentage of iron.

WM. B. PRICE, Waterbury, Conn.—I have found this blue constituent in a special naval brass that we manufacture, containing as high as 62 per cent. copper, under certain conditions of heat treatment.

E. J. DECKER.—Does the hard substance affect the tool in machining?

E. H. DIX, JR.—When the managnese bronze was etched with the FeCl$_3$ in hydrochloric acid, the blue constituent turned black almost immediately. The suggestion that the name delta prime be given this constituent was made from a desire for some shorter term.

Some time ago Jesse L. Jones suggested that we try potassium ferricyanide; we did and the whole specimen became blue. We have found both the rounded and the star-shaped particles in the same specimen and the star-shaped in a particularly high-strength manganese bronze containing no tin but large amounts of iron and aluminum. I do not doubt but that, if the tin is high enough and the copper-zinc ratio is right, you will get your delta of the copper-tin series present.

We know that these particles are hard because in polishing they stand out in relief, but we have never had any trouble in machining because they are well distributed.

BRUNO WOYSKI,* Buffalo, N. Y.—While for the time we considered this blue constituent a compound of iron and zinc, the author points out the possibility of zinc drawing this compound out of copper solution. Zinc and iron form about five compounds, so to reproduce this blue constituent in zinc it would be necessary to dissolve much less iron in zinc; about 4 per cent. of iron in zinc appears in globules. In zinc-aluminum alloys, in the presence of a small quantity of iron, a similar constituent appears, which has different forms under different cooling conditions. In manganese bronze not containing aluminum, this blue constituent is of a little different form than in the presence of aluminum.

I agree with Mr. Comstock that it is doubtful if this compound contains copper and am inclined to believe that the compound in manganese bronze contains iron, zinc and aluminum.

WM. B. PRICE.—If it were an iron, zinc, and aluminum compound you would get it in some other alloys. The iron-zinc compound that you mention, of course, occurs with small amounts of iron, very much less than you mention, in spelter.

C. H. BIERBAUM,† Buffalo, N. Y.—I do not believe the blue constituent is a primary compound; it is a transition from one state into another, because it is directly continuous and contiguous with the other crystals. Of course the small globular crystal we have called the alumnoferrite is quite a different thing.

Mr. RODELL.—In what form do you introduce iron into the manganese?

E. H. DIX, JR.—We introduce it in a hardener in copper containing also tin, aluminum and manganese.

Mr. RODELL.—We introduce the manganese first and the copper afterwards, and have gotten very good results.

R. R. SHIPMAN, Knoxville, Tenn.—In these tests, what copper was used? Is there any difference in a Lake Superior copper, Tennessee copper, or Montana copper? What difference is there in American zinc and Belgian zinc? This test may have been made with Lake Superior copper and American zinc and somebody else may try it with Tennessee copper and Belgian zinc. A lot of us are left in the dark by not knowing what copper is used.

E. H. DIX, JR.—That is a question that has been worrying us, are there any differences in the raw materials? The first experiments were made with American zinc, I do not know the brand, and electrolytic copper; later we changed the source of supply and we got different results.

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* Chief Metallurgist, Lumen Bearing Co.
† Vice-President, Lumen Bearing Co.

If we had some way of specifying sand, we could do it. It was supposed to be Albany 00, but every time we got a load it was different.

HENRY S. RAWDON, Washington, D. C. (written discussion).—The author has given us an admirable bit of "reconnaissance" work in this discussion of the nature and influence of the blue constituent in manganese bronze. The writer believes that the author intended this study only as preliminary, for, as is very evident, an elaborate investigation would be required to establish with positive assurance the exact nature of this constituent.

It is to be regretted that the observations and tests were not extensive enough to permit rather definite conclusions being drawn regarding the influence of this constituent upon the properties of the alloy, for after all, it is the properties of materials that all metallurgists, metal microscopists, and testing engineers are ultimately concerned with. The author concludes that, when present in reasonable amounts, this constituent is not necessarily accompanied by brittleness. The data given, however, are too few to enable one to decide just what constitutes a reasonable amount. According to Hoyt[15] "iron is the principal ingredient affecting the properties of the finished material (i.e., manganese bronze), the function of the manganese being to assist the copper to take up the iron in solid solution." The inference is that the iron must be in solid solution in order to assert its maximum influence. The author of the article under discussion concludes as a result of his observations that " the occurrence of the blue constituent is due principally to the presence of iron." It would seem then, that there should be some relation between these two facts, the determination of which should be of considerable practical importance.

One circumstance that renders it impossible to draw definite conclusions from the author's data is the presence of a relatively high percentage of aluminum in nearly all of the alloys. The beneficial effect of aluminum on the mechanical properties of manganese bronze is, of course, well known and its addition is a matter of common usage, particularly for castings of this material.

The writer regards the suggestion that the blue constituent "may reasonably be denoted as delta prime" an unwise one. The real delta constituent of the copper-zinc series is found only in alloys of high zinc content (70–80 per cent.) and does not ordinarily occur in these alloys except at elevated temperatures and so would never be found in manganese bronze. The term delta prime immediately suggests that in some manner, by the additions made to the alloy, this constituent has

[15] S. L. Hoyt, Metallography II, Metals and Common Alloys, McGraw Hill Co. (1921), 130.

been caused to appear out of its proper position in the family of copper-zinc alloys. The blue constituents, which, as the author shows, results from the addition of iron, is properly to be regarded as a constituent of a ternary, or even a more complex, system and may bear no relation whatever to the real delta constituent of the simple binary copper-zinc series.

In general, the experience of the writer in the metallographic examination of iron-bearing brasses, aluminum bronzes containing iron, as well as ordinary manganese bronze, confirms the conclusion reached by the author that the blue constituent is most properly to be regarded as a result of the addition of iron.

Spectrum Analysis in an Industrial Laboratory

By W. H. Bassett* and C. H. Davis,† Waterbury, Conn.

The ease and value of the application of spectrum analysis to industrial chemistry appears to be appreciated in few of the large works laboratories of this country. For 8 years, this analysis has been of great value to the authors in both analytical and metallurgical work.

To the analyst the complete qualitative analysis of an unknown alloy, revealed by a spectrogram, is a sure basis for the planning of the most direct and rapid method of attack. As the determination of each element proceeds, the purity of precipitates may be checked as often as desired. The spectrograph proves invaluable in the recognition of impurities, the separation of which would entail a lengthy and difficult procedure, or when the weight of an unknown is less than is necessary to complete the desired determinations. A few hundredths of a gram will usually suffice for spectrographic analysis.

In the separation of the rarer elements, the spectrograph will show, with precision, the point where each step has been successful. In zinc of certain classes, indium, gallium, and germanium have been isolated in this way; the presence of antimony in lake coppers has been proved; and bismuth has been detected in zinc and in alloys where the presence of a few thousandths of a per cent. would be harmful. An antimony precipitate in a delicate determination may contain a considerable proportion of tin or arsenic, both of which will be revealed. The spectrograph is consequently important to the metallurgist as well as to the analyst.

Impurities in raw material are often a source of annoyance, especially when their detection involves delay and costly analytical work. Whether low conductivity in copper is due to arsenic, nickel, or something else may be quickly found out. Residual traces of boron, magnesium, manganese, silicon, vanadium, and other deoxidizing agents are easily identified where wet analysis may fail to reveal their presence, even after days of effort. Complex alloys of any kind are dissociated by the spectro-

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* Technical Superintendent and Metallurgist, American Brass Co.
† Assistant Metallurgist, American Brass Co.
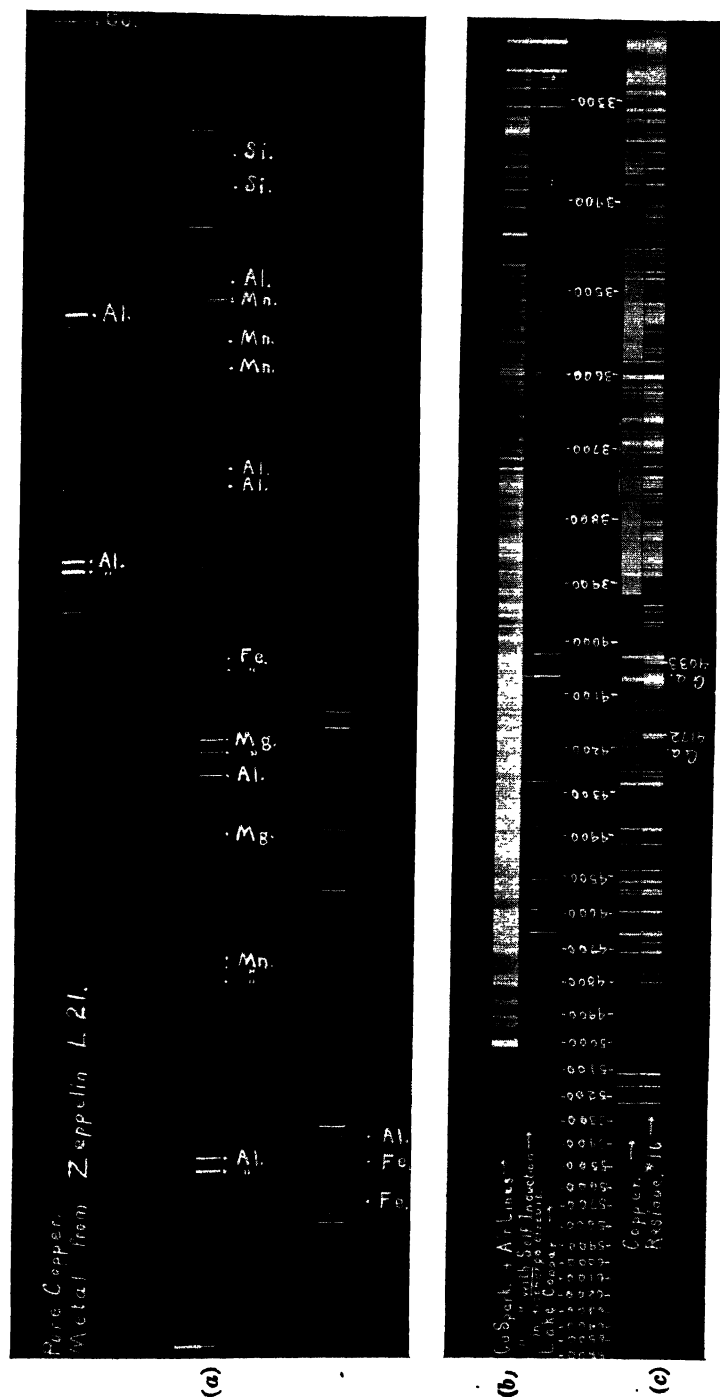
FIG. 1.—(a) SPECTRUM FROM 4600A–2100A, IN THREE PARTS; COPPER VS. METAL FROM FRAMEWORK OF ZEPPELIN L21 SHOT DOWN AT CUFFLEY, ENGLAND. (b) COPPER SPARK WITH AND WITHOUT SELF-INDUCTION IN THE DISCHARGE. (c) COPPER (ARC) WITH PRECIPITATE CONTAINING GALLIUM.

graph into a spectrum, the reading of which gives the elements present, together with an idea of the relative amounts of each.   The secrets of the inventors of alloys and hardened metals are no great problems when, for their solution, one can depend on the application of the microscope and spectrograph.

The Hilger Quartz "D" spectrograph, which has been used, has an optical train of one quartz lens of 70 mm. clear aperture and 170 cm. focal length, and a 30° prism of quartz 98 mm. length of refracting face by 57 mm. high.   The second face is coated with tin-mercury amalgam. The light is focused upon the slit by a cylindrical quartz lens and, entering, is reflected along the camera tube by a right-angled prism of quartz. It is then collimated by the lens, enters, is reflected back by the quartz prism, and retraces its path through the lens to the photograph plate. Adjustments make it possible to photograph the complete spectrum, about 24 in. (61 cm.) in length, by three exposures.   Several exposures may be taken upon the same 4 by 10 in. plate.

Certain changes have been made in the apparatus to increase the delicacy of determinations.   The three windows of the sliding diaphragm covering the slit have been so designed that comparison spectra slightly overlap, thereby enabling the observer to note the exact superposition of a line in one spectrum upon the same line in the spectrum above.   The collimating lens has been stopped down so that the beam of light is confined to a vertical opening $1\frac{1}{8}$ in. (28.6 mm.) across.   Though lengthening the exposure slightly, this device corrects the tendency of the spectrum lines to feather out at the ends, and gives perfect definition.

For the arc spectra, a 110-volt direct-current circuit is used with resistance to give 4–5 amp.   Drawn rods of pure electrolytic copper are generally employed as poles, both as a standard for comparison and for the examination of residues.   Copper has many advantages over carbon, especially in the visible portion of the spectrum.   If reference lines of greater precision are needed, iron, palladium, or other elements may be introduced into the arc.

Spark spectra are obtained by using a $\frac{1}{2}$-kw. transformer (110 volts, 60 cycles) having a secondary voltage of 2000, 3000, 4000, and 5000 on the four variations of power respectively.   Two $\frac{1}{4}$-kw. glass-plate condensers joined in multiple give approximately 0.06 microfarad total capacity.   If sufficient self induction[1] is placed in the discharge circuit, the spectrum of the air lines entirely disappears.   Increasing the self induction causes lines of many metalloids to disappear successively while lines of the metals remain.   About one hundred turns of 0.050-in.(1.27 mm.) insulated copper wire around a 2-in. (51 mm.) wooden core have

[1] Hemsalech: Compt. Rend. (1899) 122, 285.

A. deGramont: Recherches sur les spectres de dissociation.   Ann. de Chim. et de Phys. [8] (Aug., 1909) 17.
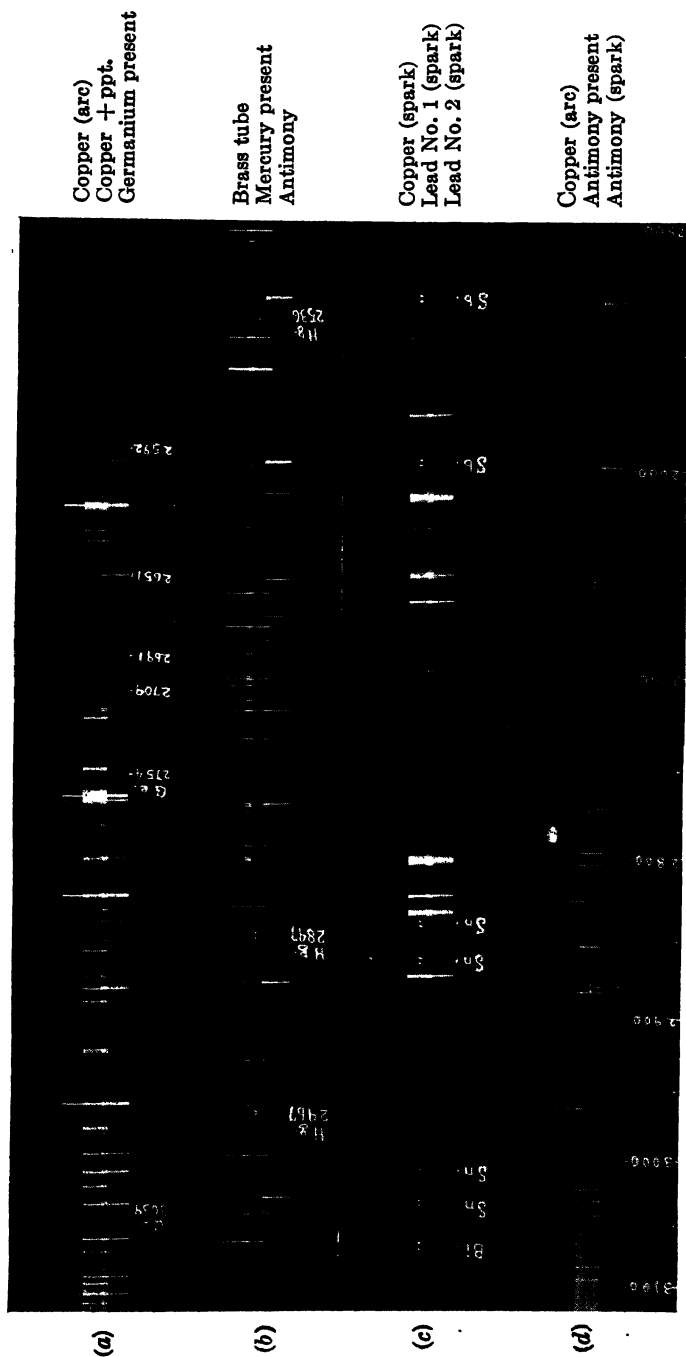
FIG. 2.—(a) GERMANIUM IN COMMERCIAL ZINC; H₂S PRECIPITATES IN COPPER ARC. (b) MERCURY FOUND IN CRACKS OF DEFECTIVE TUBE. (c) IMPURITIES IN COMMERCIAL ZINC; (d) ANTIMONY IN COMMERCIAL LEAD. (d) ANTIMONY IN LAKE COPPER; Sb₂O₄ RESIDUE IN CARBON ARC.

been found sufficient to eliminate the air lines when using the transformer and condensers described.

The length of exposure necessarily varies from a few seconds, in the case of tin, lead, etc.. to over 2 min. when dealing with those elements having a higher volatilizing point. In general, alloys are given a 1-min. exposure for the arc and 2 min. for the spark spectra. The ordinary commercial slow plate having a good thin emulsion has been found most satisfactory for use. Lines in the region 5200A–7000A are rarely essential for the investigation of alloys; in fact, with the quartz spectrograph, the region below 3300A has been of the greatest value.

For reading the spectrum plates an 80-mm. comparator accurate to 0.001 mm. is employed. With the Quartz "D" spectrograph, the dispersion in the extreme ultra-violet region is as great as with a large grating spectrograph. For instance, between wave lengths 2100A–2130A, there are 21 mm. on the plates; or in other words, $1A^2 = \frac{2}{3}$ mm., approximately. This dispersion will allow the calculation of the wave lengths of unknown lines with an error of $\pm 0.01A$. Toward the region of longer wave lengths, determinations are correspondingly less accurate.

An index of arc and spark lines has been compiled using the international standards as a basis. This index consists of all the iron lines given in the Bureau of Standards *Scientific Papers* 251 and 274, as well as all the copper lines, and finally the more important lines of the remaining elements, especially the "raies ultimes"[3] of M. A. deGramont. This index of wave lengths, with their corresponding comparator reading, is typed on the back of cross-section paper having twenty lines to the inch, so that when the sheets are assembled into a book, a curve of the wave lengths can be made on the cross-section surface opposite. By selecting standard settings, both on the spectrograph and on the comparator, and by using certain copper lines as a basis for zero readings, the examination of a spectrum plate becomes a matter of routine. The accuracy is such, however, that the wave lengths of unindexed lines may be quickly determined from the plot without calculation.

Qualitative determinations by means of the spectrograph quickly build up a certain amount of data that are of value in a quantitative way; this is especially true when the percentage of the element is very small. But, although several writers have told of success along definite lines and under constant conditions of exposure, the accumulation of general quantitative data on the spectra of elements would be a complicated undertaking, requiring the careful study of every element in its relation to every alloy or compound in which it might be present. Each operator must work out his individual problems with the instrument and accessories at his command. Experience, however, does bring to

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[2] Ångström units = wave lengths expressed in ten-millionths of a millimeter.

[3] *Compt. Rend.* (Dec., 1920) **171**, 1106.

one an increasingly clearer idea of the percentage of each element present in an unknown, especially when all conditions of exposure are kept as constant as possible.

In the accompanying plates several spectrograms are given showing the dispersion and quality of the lines and instances of the general usefulness of the large quartz spectrograph. Here can be seen well illustrated the value of the spectrographic plate record as opposed to visual spectroscopic readings. The ultra-violet field is the one of greatest interest, for below 3300A are found most of the "raies ultimes," or most persistent of the spectrum lines of an element. Here, also, the carbon arc, when its use is necessary, is free from bands which obscure part of the visible spectrum. In the ultra-violet field, the dispersion is so great that wave lengths in the prismatic spectrum may be determined accurately. Perhaps the greatest advantage of a spectrograph, however, is that of the permanent record which may be made by a laboratory assistant and checked at any future time by any trained observer.

This article is but a summary of a few instances where spectrum analysis has proved its value. It is intended to show ease, accuracy, and rapidity of accomplishment in one field only—that of metallurgy; to give testimony also to the inspiration received from those who laid the foundation of the science, and to those who, like M. A. deGramont, have written freely of their work especially along industrial lines. To Prof. H. S. Uhler, of Yale University, we wish to express our appreciation of the design of the sliding diaphragm giving overlapping comparison spectra, and of his many helpful suggestions.

DISCUSSION

C. S. BRAININ, New York, N. Y.—The author states that some work has been done in quantitative analysis. That is an important new problem that the spectro-chemist is attempting to solve; can he give some of the information he has gained on that point?

C. H. DAVIS.—Inasmuch as most of our work has been done directly with the chemists, we have not gone extensively into work along quantitative lines. Messrs. Hill and Luckey have published papers[4] upon quantitative work, and more are to be published, I understand, by others. Our own work along quantitative lines has been very limited. From experience gained in reading several hundred plates, we estimate the approximate amount of each element, and in that way give the chemist an accurate idea of how to proceed with his analysis.

W. H. BASSETT.—The spectroscope is used in connection with the quantitative analysis of the metals. For instance, in the complete analy-

[4] Spectroscopic Determination of Lead in Copper. *Trans.* (1919) **60**, 342.

sis of refined copper, the precipitate of a very small amount of impurity that has been obtained from a relatively very large amount of copper may be of questionable purity; the spectrograph will quickly and accurately answer the question. For a long time there was a controversy as to whether lake copper contained antimony; we knew that electrolytic copper carried a few thousandths of a per cent. By the spectrograph we proved that the precipitates obtained from lake copper were actually antimony precipitates. Also, the spectroscope can be used as a method of qualitative analysis, with some idea of the amounts present, so that the analyst may go ahead without any question as to what he is to find and with a rough idea as to the proportions present.

GEORGE K. BURGESS,* Washington, D. C.—Much work of this kind, along various lines, has been done at the Bureau of Standards. One case, the examination of tin for boiler plugs for the Steamboat Inspection Service,[5] was quite an onerous job for the chemist; the spectroscope gave satisfactory results. The method of determining the existence of a small quantity of an impurity is based on a method of constructing comparison standards; that is, a series of alloys of known compositions is made, then, when an unknown metal is examined spectroscopically, by a determination of the intensity and number of the lines, it is possible to get a good estimate of the quantity of the added element, if it is not in too great excess.

In the work the Bureau is doing on the platinum metals, the spectroscopic method is used as an auxiliary for the determination of the purity of a metal. Analytically, it is extremely difficult to separate certain of these elements, so two auxiliary physical methods of analysis have been used—the thermoelectric and the spectroscopic method. After having made up certain standard alloys, it is possible to obtain the exact analysis of an alloy, by a comparison of spectra. For a pure metal like platinum, the chemist makes an analytical determination, the thermoelectric test gives him a better idea as to how well he has done in his purification, and then the spectroscopist can give him an analysis of the final impurities and, in the limiting case, certainty of their elimination. It has been possible, for example, to check out the purest gold that has ever been made by the Mint through the use of this method, namely, 99.9999 per cent. gold.

GEORGE K. ELLIOTT,† Cincinnati, Ohio.—Has this method been definitely applied in the determination of very small quantities of alumi-

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num—a few hundredths of a per cent.—in bronzes? The complete absence of aluminum is important in the manufacture of bronze castings that must withstand liquid or gas pressure tests because only a few hundredths of a per cent. of aluminum is enough to make the number of rejections due to leakage extremely high. If the spectroscope is a ready and sure means of determining small percentages of aluminum, it will be most valuable to these manufacturers.

LEONARD WALDO,* New York, N. Y.—We have used the metallurgical spectroscope for a number of years, and have found it an exceedingly simple and quick shop method of detecting zinc, aluminum, and magnesium. All the work referred to so far, as far as I know, could have been equally well done for manufacturing purposes with the flint-glass prism apparatus, or two prisms could be used. The main thing is the observing telescope must be long enough to give easy vision.

The deflagrating apparatus for metals should receive careful attention; an efficient arrangement is a storage cell of three elements with a 10-in. Richie induction coil. The Hilger spectrograph is a luxury except for precision work; in most cases it is not necessary to photograph the spectrum. By using a good micrometer scale moved by a micrometer screw at the end of the scale image forming telescope excellent results, for metallurgical purposes, can be obtained quickly and easily.

KEIVIN BURNS, Pittsburgh, Pa. (written discussion).—The subject of quantitative analysis has recently been advanced by deGramont and others, and I think its possibilities are not yet fully realized. While the number of standard samples necessary to cover the whole field of spectroscopic analysis is beyond number, for a particular case a single sample may suffice, or two samples analogous to "go" and "no go" gages. When dealing with small quantities of an impurity, one pair or set of standard samples may frequently be sufficient for the determination of a large number of elements, as for instance, iron, copper, zinc, lead, etc., in boiler-plug tin.

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* Consulting Engineer.

Testing Artillery Cartridge Cases*

By J. Burns Read,† Golden, Colo., and S. Tour,‡ Brooklyn, N. Y.

(New York Meeting, February, 1921)

It is the purpose of this paper to summarize, as far as possible, the metallurgical information and experience gained by the Ordnance Department, during the war, in the manufacture of artillery cartridge cases, to describe the tests to which artillery cartridge cases were put, and to summarize the knowledge and experience gained in this testing. The authors wish to acknowledge the aid given by manufacturers, army officers, and civilian employees of the Ordnance Department in obtaining these data.

Artillery cartridge cases are cold drawn from circular disks punched from strips of rolled brass. The manufacture involves from 25 to 30 main operations, which are done almost entirely on a punch press. Descriptions of the operations have appeared in various periodicals and, therefore, will not be given here.

After each cupping, or drawing, operation, except the last, the cases are annealed at from 540° to 650° C. (1000° to 1200° F.), to remove all the hardening stresses set up in the metal and to allow complete recrystallization. After each annealing, the cases are pickled in niter cake or sulfuric-acid solution and then washed in water to remove all traces of acid. In some instances, the cases are then immersed in a cyanide-salt solution, and afterwards rinsed in water, to remove the thin layer of copper that forms on the surface of the cases during the acid pickle.

The last cold-drawing operation is followed by the heading operation which, by flattening the head of the case on the outside, causes the metal to flow on the inside to approximately the proper finished dimensions. The mouth, or open end, is then annealed at 430° to 480° C. (800° to 900° F.) by immersion in a salt bath or by gas flames. After being washed, the cases are tapered, when they are ready for machining, inspection, and testing.

CARTRIDGE-CASE DISKS

Good disks of proper composition and physical properties are essential to the production of good cartridge cases. The Ordnance Department

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has specified the following chemical composition: copper, 68 to 71 per cent. and zinc, 29 to 32 per cent. It requests that copper plus zinc, minimum, shall be 99.88 per cent.; lead, maximum, 0.07 per cent.; iron, maximum, 0.05 per cent.; tin, antimony, bismuth, cadmium, or
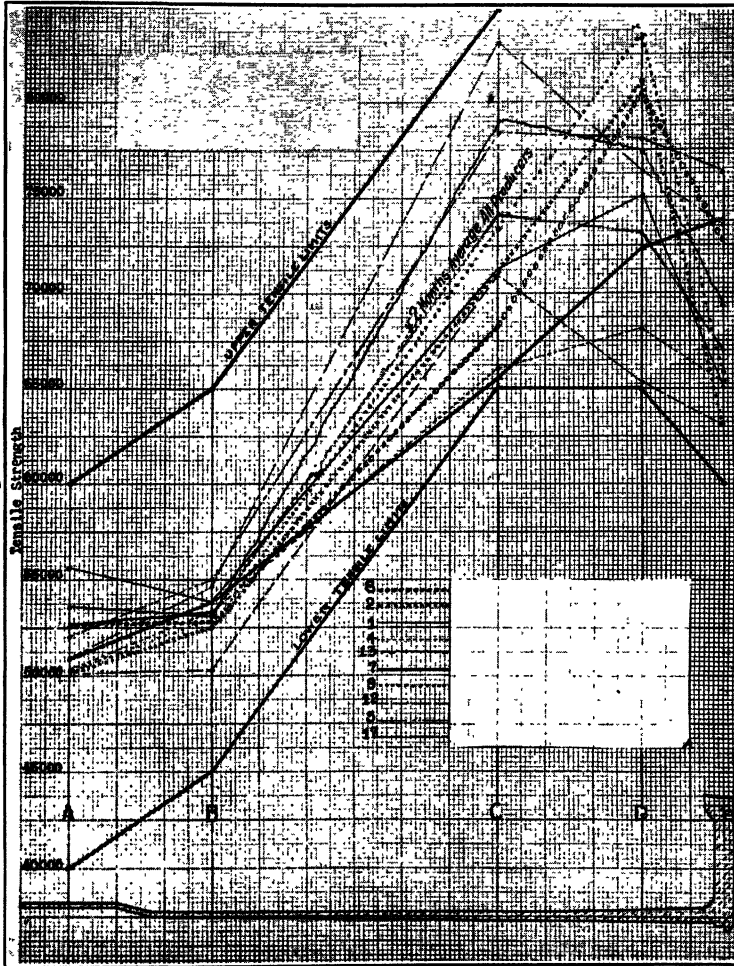
FIG. 1.—COMPARATIVE TENSILE-STRENGTH TESTS OF TYPICAL 75-MM. CARTRIDGE CASES FROM DIFFERENT MANUFACTURERS.

arsenic, none or only a trace. The materials used are prime lake or best electrolytic copper and grade A zinc.

The specifications do not give physical properties but suggest that "an average of ten readings should be between 50 and 65 Brinell and no individual test should be below 49 or above 69."

The limits of 49 to 69 Brinell with the average between 50 and 65

cover nearly all the disks manufactured into cartridge cases during the emergency. Some manufacturers preferred the higher and some the lower Brinell limits, depending largely on their equipment and tools, for a disk may be too soft to be workable by one set of tools though it may be satisfactorily worked by another. A very soft or a very hard

FIG. 2.—STRUCTURE AND TENSILE STRENGTH OF 75-MM. CASE AS MADE BY
MANUFACTURER NO. 8.

disk has a tendency to punch through instead of cupping. The harder the disk, the greater is the stress on the press forming it into a cup, but it is generally believed that the harder disk will form a better case than the softer disk. Whether this is true will depend on the conditions of working and annealing in the process of manufacture. If these conditions are

properly controlled, very soft and very hard disks will form equally good cases.

Fig. 31 shows the results obtained from 42.5 and 71 Brinell disks. This difference was obtained by annealing disks at 500° C. (930° F.) and
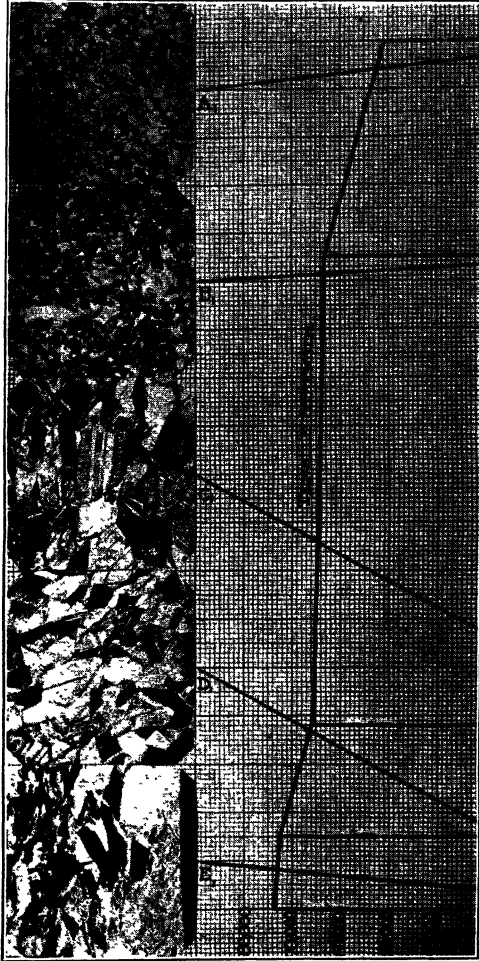
FIG. 3.- -STRUCTURE AND TENSILE STRENGTH OF 75-MM. CASE AS MADE BY MANUFACTURER No. 7.

750° C. (1260° F.) for 70 min. The grain size shown is typical of this anneal. These disks were manufactured into cartridge cases by the same processes and under identical conditions of work and annealing and it will be noted that the results are practically the same in spite of the fact that the disks were beyond the extreme limits of hardness. It is not contended that the cases are ideal, but they represent the prod-

uct manufactured at the time this test was made. If this test were applied to present practice the results would be identical.

The limit of hardness on the upper side of these Brinell limits is dependent on the press capacity and a sufficient anneal to permit cupping;
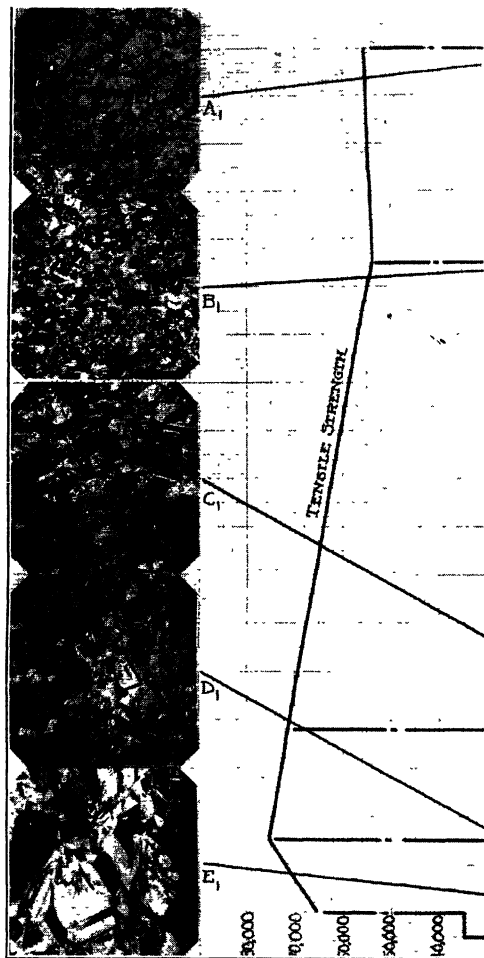
FIG. 4.- -STRUCTURE AND TENSILE STRENGTH OF 75-MM. CASE AS MADE BY MANUFACTURER No. 1.

the limit on the lower side is dependent on the anneal, which will deteriorate or volatilize some of the metal or allow penetration because of tool design. The coarse crystallization caused by too high an anneal will result in a grainy, or orange-peel, surface. Such a condition on the finished case may be due to too high an anneal between draws.

Approximately, the Brinell hardness and grain size that should result

from annealing at various temperatures for ½ hr. after 35.1 per cent. reduction are:[1]

ANNEALING TEMPERATURE, DEGREES C.	BRINELL HARDNESS	GRAIN SIZE, MILLIMETERS	ANNEALING TEMPERATURE, DEGREES C.	BRINELL HARDNESS	GRAIN SIZE, MILLIMETERS
450	75	0.020	650	55	0.062
500	70	0.023	700	50	0.110
550	65	0.033	750	46	0.197
600	61	0.044	800	43	0.300

SPECIFICATIONS FOR AND INSTRUCTIONS ON TESTS

The testing of cartridge cases for acceptance may be divided into five sections: Chemical analysis, determination of tensile strength and percentage of elongation, microscopic and macroscopic examination, mercuric-chloride test, and ballistic or proof test. The testing by chemical analysis will not be discussed here.

During the recent emergency, the metallurgists of the Ordnance Department based their work on artillery cartridge cases on the idea that starting with a disk made of high-grade cartridge brass that had passed a thorough visual inspection, with a well-ordered shop practice covering thorough control of annealing and amount of draft, the finished case is bound to meet all requirements specified by laboratory tests and gun-proof, in so far as the material is concerned.

Up to the time the United States entered the war, little attention had been given to the metal quality of cartridge cases. Specifications mentioned metal conditions and placed no limits other than chemical. Little systematic consideration had been given to the possibilities of control of the final product by the control of manufacturing processes. Because of many changes resulting from quantity manufacture and an increased knowledge of the requirements of cartridge cases, it was necessary to revise these specifications. The revision requires photomicrographs if the microscopic examination continues to reveal an unsatisfactory condition of the crystallized structure in several lots. This would indicate that photomicrographs, when the metal is good, were not to be taken.

The revision also provides for tensile and mercuric-corrosion tests "merely for the guidance and general information for the manufacture, and no lots will be rejected if cases fail to pass these tests." It retains the requirements of the previous specifications as to ballistic tests and storage.

In order to provide for the proper application of these general specifications, from time to time instructions were issued to inspectors directing, in detail, methods of inspection and test.

[1] W. H. Bassett and C. H. Davis: A Comparison of Grain Size and Brinell Hardness of Cartridge Brass. *Trans.* (1919) **60**, 428.

One of the first instructions directed that microscopic examinations include the making of photomicrographs. These instructions required that photomicrographs of six sections be made from each of five cases taken at random from every lot. It was then thought that the photo-
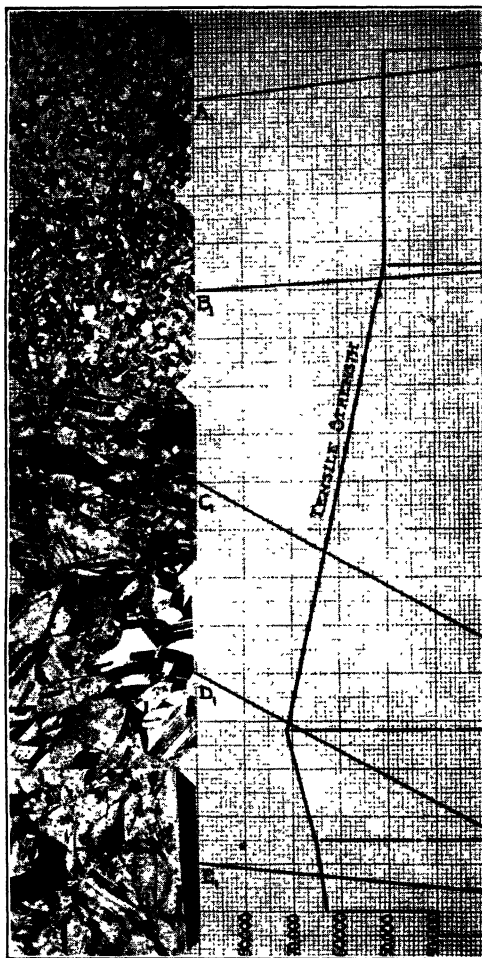
FIG. 5.- -STRUCTURE AND TENSILE STRENGTH OF 75-MM. CASE AS
MADE BY MANUFACTURER No. 12.

micrographs would furnish sufficient information for proper control of the inspection. This did not prove to be the case because of the many variables in worked brass that has not been annealed. Inexperienced personnel, varying processes of manufacture, and varying laboratory methods added to these variables and the photomicrographs did not reveal the quality of the case which they represented.

Later instructions provided optional methods of testing each lot as follows: Mercuric test of one case and microscopic examination and photomicrographs of this case and five others, six in all being required; or, tensile-strength and elongation tests of two cases, mercuric test of
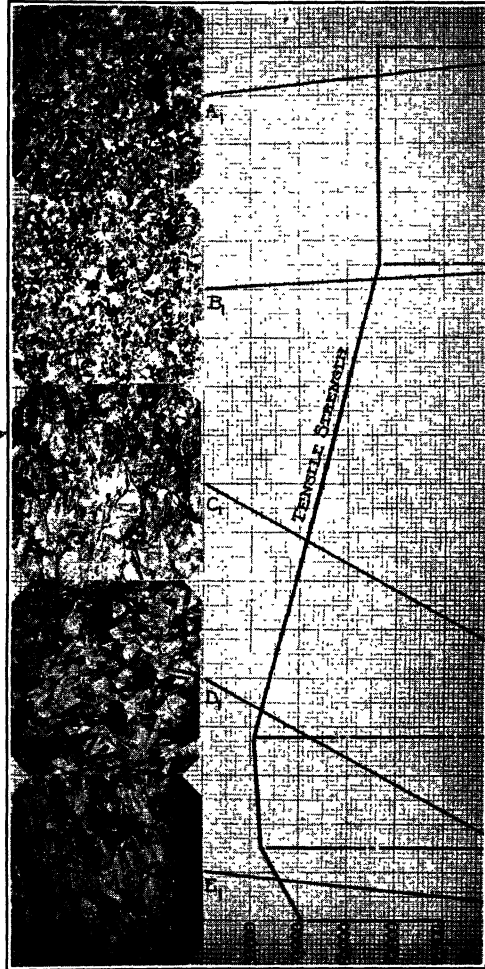
FIG. 6.—STRUCTURE AND TENSILE STRENGTH OF 75-MM. CASE AS MADE BY MANUFACTURER No. 13.

one case, microscopic examinations and photomicrographs of one case, four cases in all being required.

Samples of the product of several contractors were sent to the Bureau of Standards for mercuric-test study and tensile-strength tests; the results showed that these tests were a proper means of inspection.

Under either method of testing, a failure of a case to stand the mer-

curic test and retest constituted a rejection of the lot it represented. At first most, and later all, of the contractors adopted the second method of testing. Final acceptance or rejection, of course, is based on the results of the ballistic test.
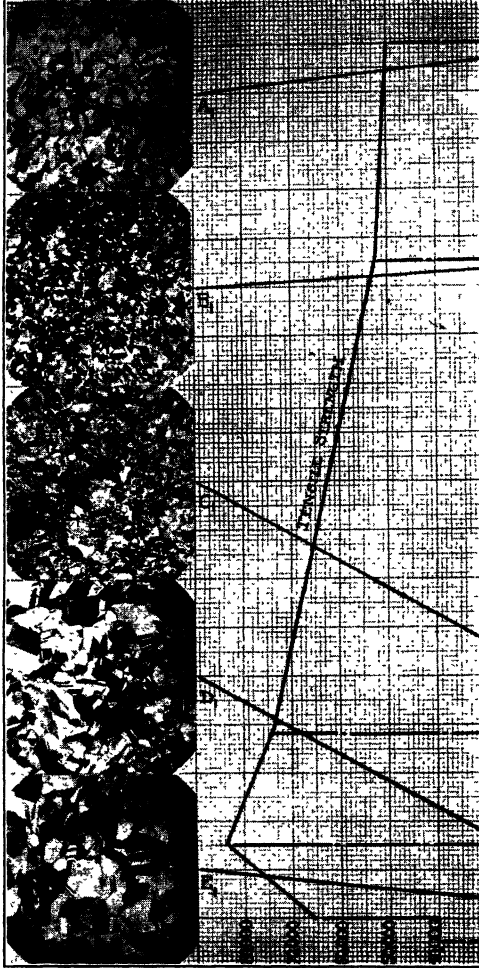
FIG. 7.—STRUCTURE AND TENSILE STRENGTH OF 75-MM. CASE AS MADE BY MANUFACTURER No. 9.

TENSILE-STRENGTH TESTS AND MICROSCOPIC EXAMINATIONS

The physical and microscopic properties of the metal of cartridge cases are largely dependent on the combined control of the various annealings and redraws, particularly the last. As shown elsewhere, an increase in the amount of work done on the metal in any step reduces the temperature at which the anneal of that metal will take place, and

vice versa; therefore, as the work affects the results of the anneal, the anneal affects the results of the work and both affect the physical properties. It is essential that these be considered together.

Specimens for tensile-strength tests were taken from the various
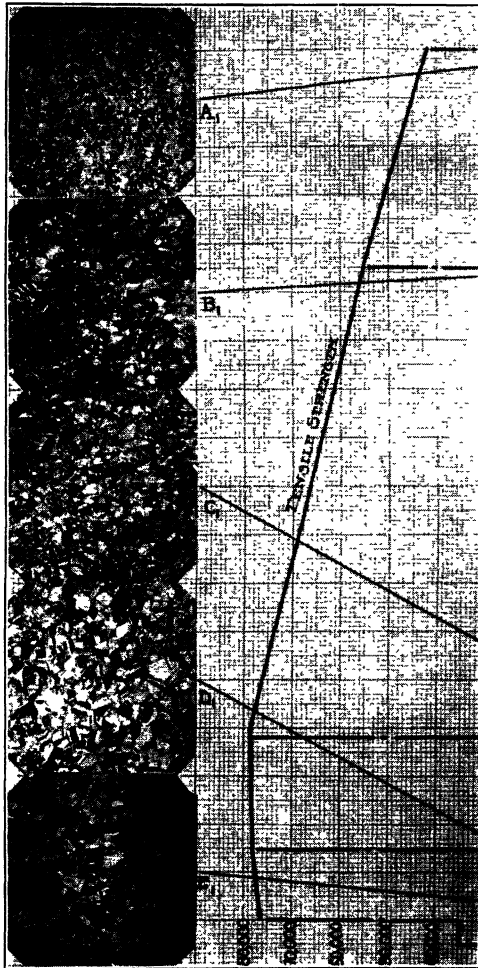
FIG. 8.—STRUCTURE AND TENSILE STRENGTH OF 75-MM. CASE AS MADE BY MANUFACTURER No. 5.

sizes of cases, as indicated in Figs. 12, 17, 23, 25, 26, 27, 28, and 39. Tensile-strength and elongation tests were determined on each piece. These methods of taking tests and the size of the test pieces proved very satisfactory with the exception of the E piece, the location of which is such that the elongation of one side varies from that of the other so that the results are not indicative of either the metal at the center of the head

or at the primer hole. This is due to the method of manufacture, which works the metal around the primer hole more than the remainder of the head.

Recent instructions on the testing of cartridge cases provide for the

FIG. 9.—STRUCTURE AND TENSILE STRENGTH OF 75-MM. CASE AS MADE BY MANUFACTURER NO. 17.

making of an additional test, if desired, as shown in Fig. 37. This test would furnish definite information as to the ability of the primer hole to resist expansion in firing, and would, therefore, be more dependable than the test now in use. The objection to this test is that the microscopic examination of the primer could not be made on the case used for physical tests, unless this examination were made on the physical-test

piece before pulling. It is believed, however, that such a test will give more consistent results, will tell more than the photomicrograph, and, as it can be applied to the small sizes (37-mm. and 1.457-in.), that it should be adopted in place of the now prescribed E tensile-strength
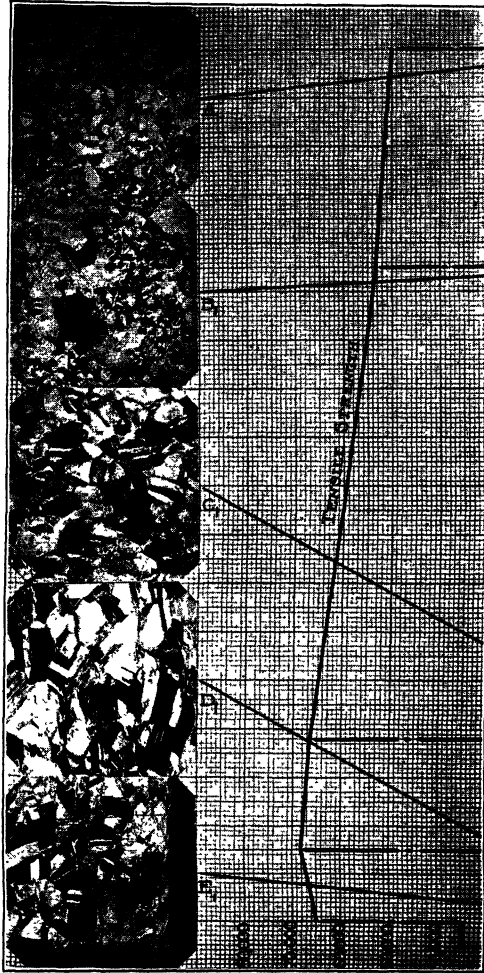
FIG. 10.—STRUCTURE AND TENSILE STRENGTH OF 75-MM. CASE AS MADE BY MANUFACTURER No. 4.

test. In the larger cases, the present head tensile-strength test should also be made. As this primer-hole test has not heretofore been made, no definite statements can be made as to what results should be obtained. They should, however, be slightly in excess of the results obtained from the present E section, as the primer-hole hardness should increase the strength of this section over the other. For all cases 3 in. and smaller,

this strength would probably be 65,000 lb. and for larger cases 60,000 lb. or more.

The tensile-strength limits for cartridge cases stated in the instructions to inspectors were as follows:

Type of Case	Tensile-strength Limits, in Pounds per Square Inch				
	A	B	C	D	E
37-mm. cartridge case...............	45,000 to 55,000		Over 60,000	Over 60,000	
1.457-in. gun and howitzer subcaliber cartridge case....................					
75-mm. cartridge case...............	40,000 to 60,000	45,000 to 65,000	65,000 to 85,000	65,000 to 85,000	Over 65,000
3-in. 15-pdr. cartridge case...........					
4.7-in. gun cartridge case............	40,000 to 60,000	45,000 to 65,000	65,000 to 85,000	65,000 to 85,000	Over 55,000
All other howitzer and trench mortar cases	40,000 to 60,000		Over -60,000		Over 53,000

The requirements as to microstructure were as follows:

The photomicrographs should show a well-defined recrystallization at the A_1 section. The B_1 section should preferably show recrystallization, but if recrystallization has begun, this is satisfactory. The C_1 section should show a good hard metal structure. The D_1 and E_1 sections should show well-worked metal, especially on the primer hole.

The control of the manufacture and the acceptance of cartridge cases by tensile-strength and microscopic tests have been quite satisfactory and the experience indicates the correct limits. It might be possible, however, to develop equipment for determining hardness instead of tensile strength and to make such determinations without destroying the case. The "Baby" Brinell machine developed by the Ordnance Department can possibly be adapted to this purpose. Impressions made by this machine are small and would not affect the properties of the case or its proper functioning. Microscopic examination could be made of this case without cutting into sections. Such a method of testing would also permit a greater number of cases to be tested with less work, and thus make the sample more accurate and complete.

The physical requirements listed above were determined by testing a large number of cases, manufactured by various concerns, that functioned properly in ballistic tests and indicated that they possessed the proper requirements for long life and a sufficient factor of safety.

The Ordnance Department required that elongation tests be made and recorded whenever tensile-strength tests were made, although no limits were set. The elongation test usually adds to the information furnished by the tensile-strength test but it is not believed to be of sufficient importance to warrant its use as an inspection test. Fig. 42 shows the results of elongation tests corresponding to the tensile-strength

tests recorded in Figs. 1 and 18. Nothing is shown here that would not be expected from the tensile-strength tests. Microscopic examination and mercuric tests will furnish additional information.

A number of articles have been published on the physical properties
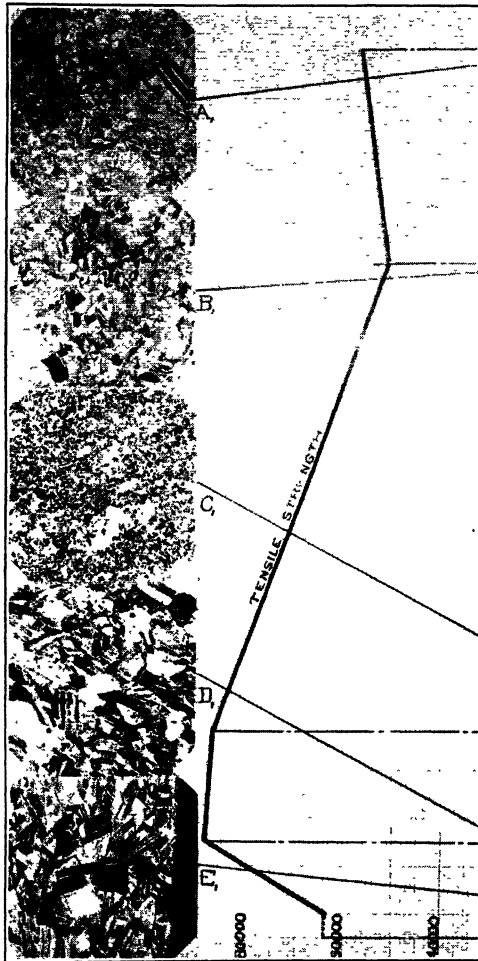
FIG. 11.—STRUCTURE AND TENSILE STRENGTH OF 75-MM. CASE AS MADE BY MANUFACTURER NO. 2.

of alpha brass; two of these[2] are particularly applicable to the manufacture of cartridge cases. The photomicrographs of Bassett and Davis

[2] Bassett and Davis: *Op. cit.*

· C. H. Mathewson and A. Phillips: Recrystallization of Cold-worked Alpha Brass on Annealing. *Trans.* (1916) **54,** 608

show the metal condition resulting from variations in annealing and should be carefully studied.

During each annealing between redraws, the metal will become completely recrystallized and, if annealed as specified (not to exceed
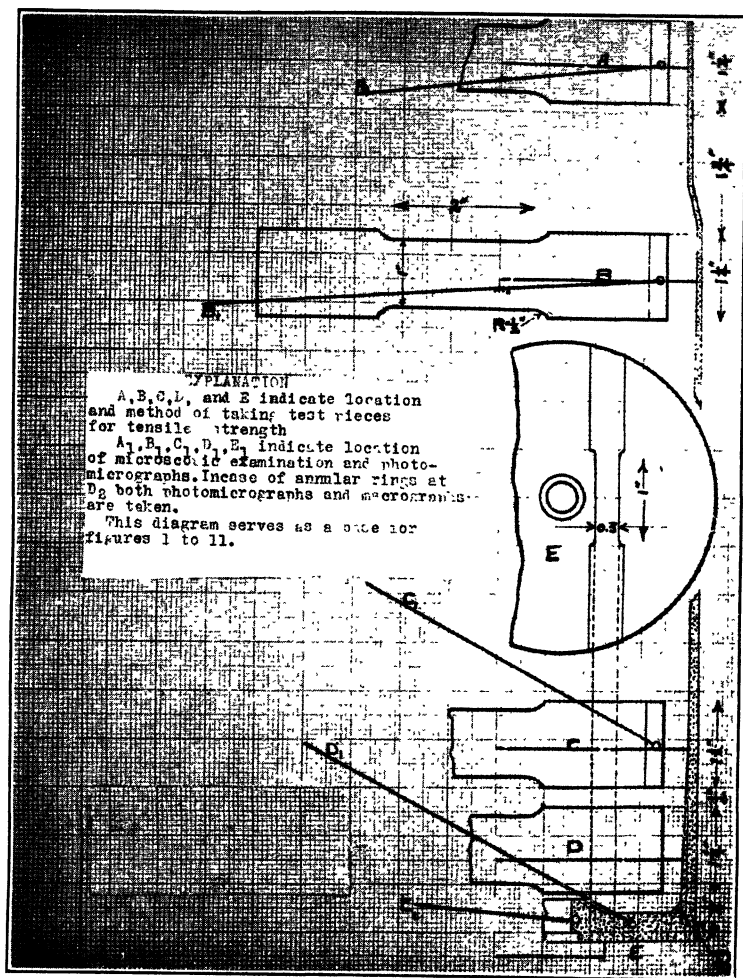
FIG. 12.—METHOD OF TAKING TESTS OF 75-MM. CARTRIDGE CASES.

650° C.), the grain size should be 0.070 to 0.080 mm. in diameter. If, however, the time is too long or the temperature higher than 650° C. (1200° F.), larger grains will result. If the tools were so designed that no work or very little was done on, say, the head during a redraw, that portion would not recrystallize and the grains would continue to grow.

This will mean a lower tensile strength and will affect the grain size

from the next working and annealing. The rate of heating will also affect the grain size. Jeffries[3] finds that the faster the heating rate the smaller is the grain size for any temperature, and the slower the heating rate the greater is the grain size. These account for variations in
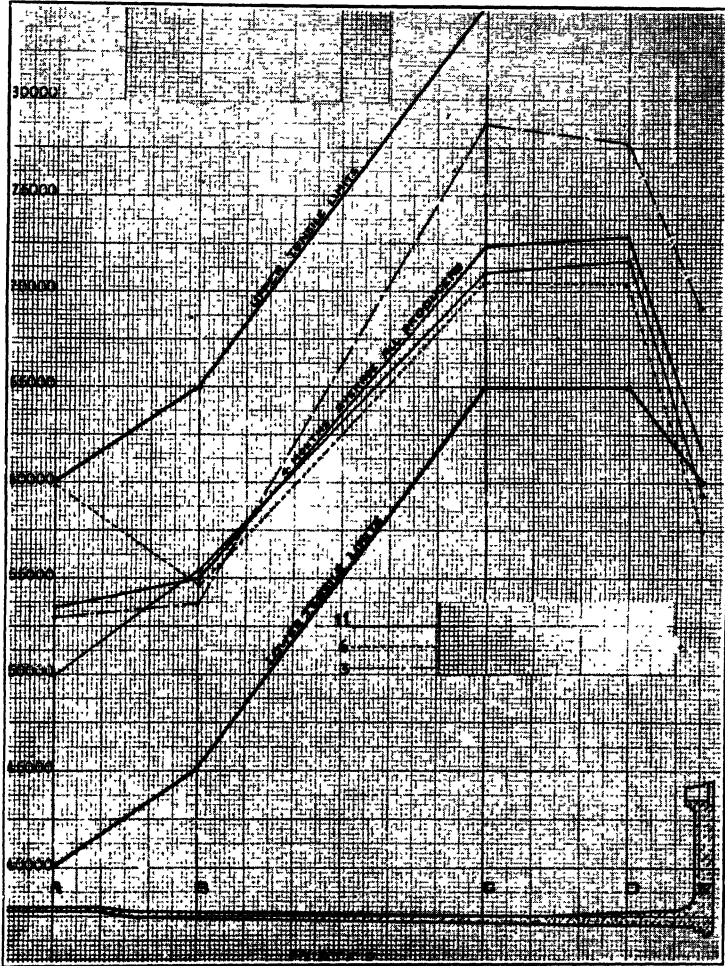
FIG. 13.—TENSILE STRENGTH OF 3-IN., 15-PDR., ANTIAIRCRAFT CASES, AS MADE BY MANUFACTURERS NOS. 3, 6, and 11.

the cases of different manufacturers illustrated in the accompanying photomicrographs.

For example, the photomicrographs and the tensile-strength results of sections E_1 and E, in Figs. 8 and 9, which represent the heads of

[3] Discussion of paper on Recrystallization of Cold-worked Alpha Brass on Annealing. *Trans.* (1916) **54,** 660.

cases by two manufacturers, show a difference in grain size and the tensile strength of the section with the smaller grain is 76,500 lb. per sq. in. and of the other 68,500 lb. per sq. in.   It is likely that the metal of section D_1, Fig. 8, is the result of recrystallization from annealing after
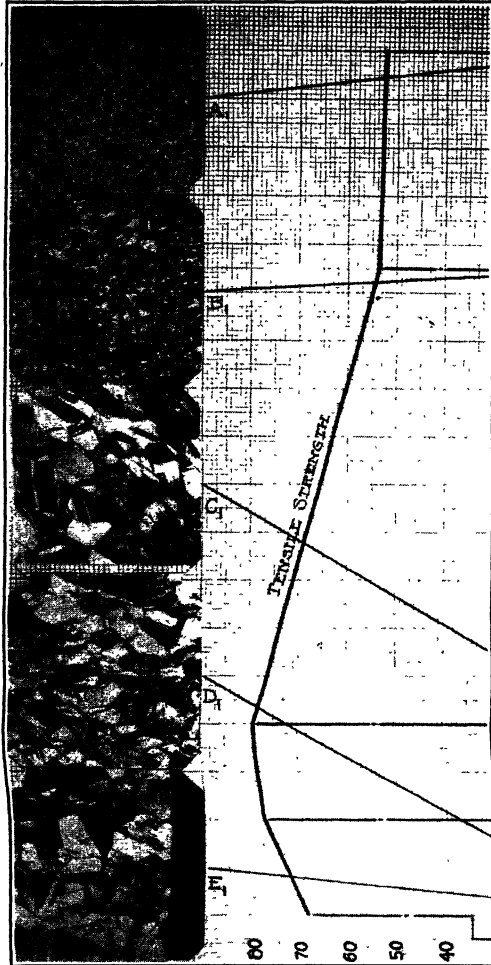
FIG. 14.—STRUCTURE AND TENSILE STRENGTH OF 3-IN., 15-PDR , CASE AS MADE BY MANUFACTURER NO. 3.

considerable working.   If this working amounted to 35 per cent. reduction, the condition shown would have resulted from a 600° C. (1110° F.) annealing for $\frac{1}{2}$ hr.   If, instead, a very slight or no reduction had taken place, the grains would have continued to grow, instead of recrystallizing, in the annealing and would have produced the result shown in D_1, Fig. 9.   Further, if a temperature of 700° C. (1290° F.) were

used in the annealing of metal after reducing it 39 per cent., the condition of section D_1, Fig. 9, would be expected. It is, therefore, evident that the grain size, tensile strength, and hardness are dependent on the amount of reduction previous to the annealing, the temperature of the annealing, and the grain size from the previous anneal.
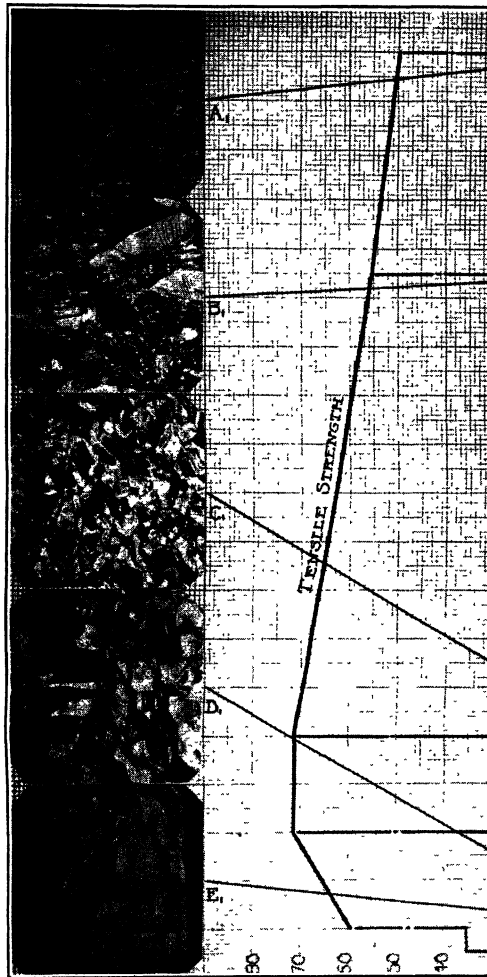
FIG. 15.—STRUCTURE AND TENSILE STRENGTH OF 3-IN., 15-PDR., CASE AS MADE BY MANUFACTURER NO. 11.

If a portion of a cartridge case is worked to any extent, that portion will recrystallize in annealing at 650° C. (1200° F.) for ½ hr. or more, and the resultant grain size will be approximately the same for all degrees of working. If slight or no working had taken place, recrystallization in the 650° C. (1200° F.) annealing would not occur and an increase in

the original grain size might be expected.   Mathewson and Phillips[4] state: "In order that the annealing effect may be felt at a given temperature, it is clear that the mechanical destruction of the original grain must have been sufficiently pronounced to produce fragments inferior in size to the recrystallized grain characteristic of this temperature."
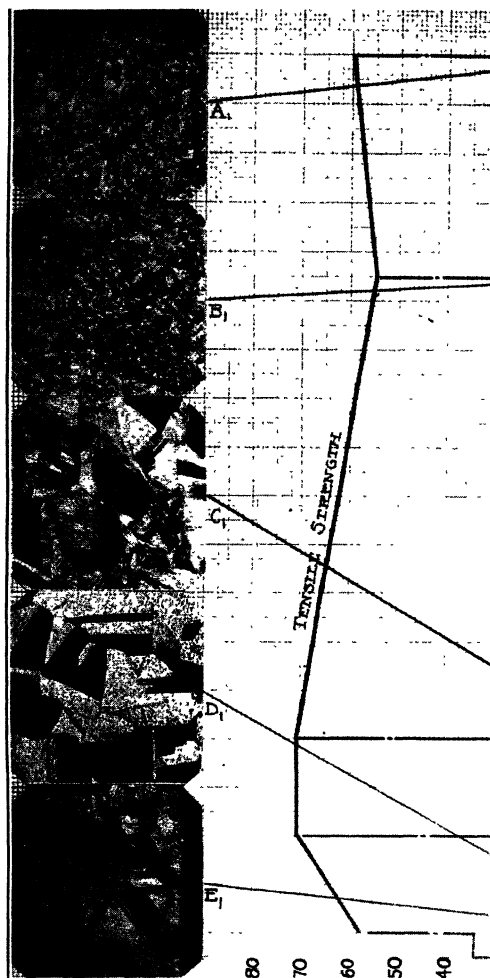
FIG. 16.—STRUCTURE AND TENSILE STRENGTH OF 3-IN., 15-PDR., CASE AS MADE BY MANUFACTURER No. 6.

A low-temperature treatment, less than 200° C. (390° F.) of heavily worked unannealed brass, produces a distinct hardening.   Bassett and Davis[5] found that heat-treating brass that had been reduced 39 per

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[4] Mathewson and Phillips: *Op. cit.*
[5] Bassett and Davis: *Op. cit.*

cent. in working, to 200° C. (390° F.) increased the Brinell hardness from 146 to 154. Similar results are shown in Figs. 29 and 30, which illustrate the results from mouth annealing in a nitrate bath at various temperatures; the difference in the heat conducted from the bath to the metal 6 in. from the mouth of the case was sufficient to produce a distinct
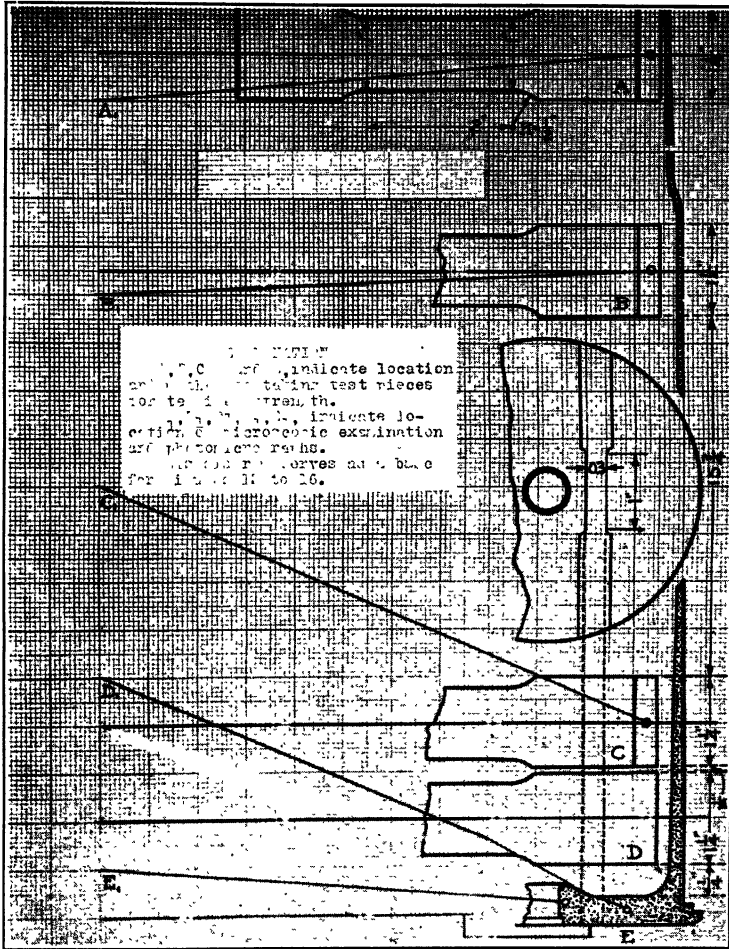
FIG. 17.—METHOD OF TAKING TESTS OF 3-IN., 15-PDR., ANTIAIRCRAFT CASES.

increase in tensile strength in those subjected to a temperature of 450° C (850° F.) over those subjected to 425° C. (800° F.).

The effect of differences in annealing temperature of the metal at the mouth is also shown in Figs. 29 and 30. A temperature of 425° C. (800° F.) was not sufficient to complete the annealing and only partial

recrystallization resulted, while 480° C. (900° F.) produced nearly complete recrystallization and what is now considered an ideal condition for the *A* and *B* sections.

The effect of working and annealing is further shown in Figs. 32 to 35. First, these show that variations in the amount of working produced
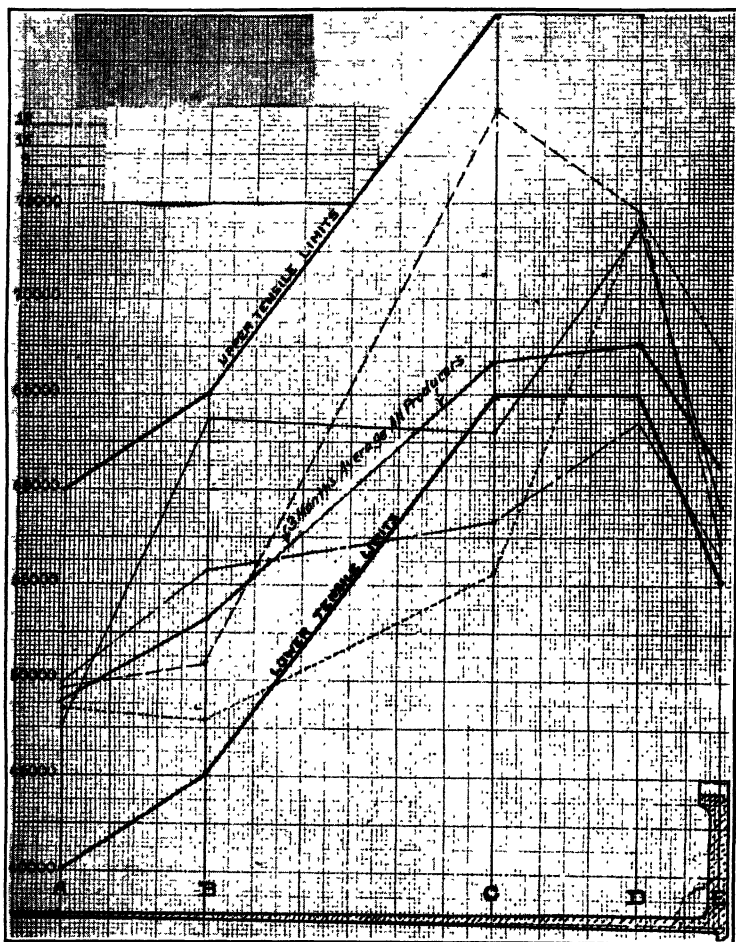
FIG. 18.—TENSILE STRENGTHS OF 4.7-IN. GUN CARTRIDGE CASES AS MADE BY MANUFACTURERS NOS. 2, 8, 15, AND 18.

a very beneficial difference in the *C* section; in the case of less working the tensile strength was not forced as high, while in the case of greater working the tensile strength was probably increased and then reduced by the greater susceptibility to anneal because of this work, though this cannot be detected in the photomicrograph. Second, the variation in the mouth anneal shows that at a temperature of 460° C. (860° F.), the metal

that had been reduced 0.012 in. showed crystallization almost complete in the *A* section and just starting in section 1. At a temperature 50° higher, much less recrystallization is apparent on the less worked metal and is not seen at all in section 1. Finally, when the work was increased, recrystallization was complete in section *A* and well started in section 2.
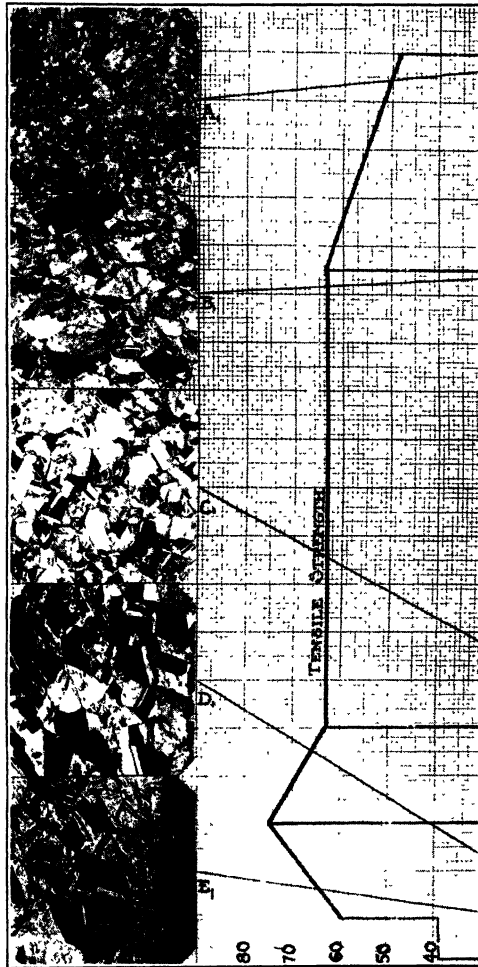
FIG. 19.—STRUCTURE AND TENSILE STRENGTH OF 4.7-IN. CASE AS MADE BY MANUFACTURER No. 15.

Extensive work is not detrimental to the product and the anneal of such metal is accomplished more readily than metal worked to a less extent. It also would appear that hard pinching and working would tend to avoid local strain and to produce a uniformity of work through the metal.

The mouth annealing practice of the various manufacturers was as follows:

Company Number	Type of Furnace	Amount of Case Heated, Inches		Temperature, Degrees C.	Time of Anneal	
					Min.	Sec.
		37-mm. Cartridge Case				
16	Rotary gas................	2		650–675		40
14	Nitrate bath.............	1⅛		525	1	15
10	Rotary gas................	no data				
		75-mm. Cartridge Case				
1	Nitrate bath	9		454	5	0
7	Rotary gas................	6¼		480	1	20
5	Nitrate bath.............	6		435–450	2	0
9	Rotary gas................	9		415–440	2	8
12	Nitrate bath	7	(before taper)	455	1	30
		2	(after taper)	390	1	0
8	Rotary gas................	6		490	3	35
2	Rotary gas................	10		480	1	0
4	Rotary gas................	2½		425	1	20
13	Nitrate bath.............	6		480	1	0
		4.7-in. Cartridge Cases				
15	Rotary gas..........	10		495	2	45
9	Rotary gas...............	12		455–480	2	10
2	Rotary gas...............	12		480	1	0
		3-in. 15-pdr. Cartridge Cases				
6	Nitrate bath.............	14	(1st taper)	445		
		4½	(2d taper)	440	1	0
11	Nitrate bath.............	14	(1st taper)	390	3	
		4	(2d taper)	425	2	
3	Nitrate bath.............	14	(1st taper)			
		6	(2d taper)			

MERCURIC TESTING

Season, or corrosion, cracking of brass has long been a puzzling phenomenon and has caused much trouble in both small-arms and artillery cartridge cases, which frequently develop cracks after being stored for a time. Such cracks have occurred in both loaded and unloaded cases and in all conditions of exposure and protection. Most of these

cracks occurred in the mouths of the cases but serious cracks have occurred in the body or head. Chemical, physical, or microscopic tests do not indicate in any way the liability to season cracking. The cartridge-case specifications are partly incorrect in the statement that: " Microscopic
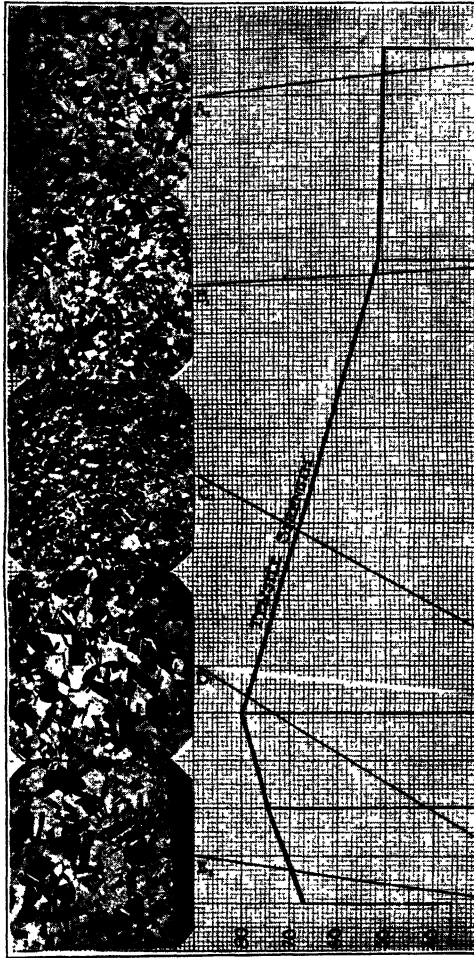
FIG. 20.- -STRUCTURE AND TENSILE STRENGTH OF 4.7-IN. CASE AS MADE BY MANUFACTURER No. 9.

test will consist of sectionalizing, polishing, etching, and examining microscopically the two cases selected to determine whether the various mechanical operations and subsequent heat treatments have been such as to leave the crystalline structure of the material in proper condition for obturation and storage. "

It is believed that the season crack is the result of time corrosion and

weakening of the metal until the stress exceeds the tensile strength. Corrosion obtained in the mercuric test produces "season" cracks in a few minutes or hours.

The Ordnance Department has specified that "the mercuric test will
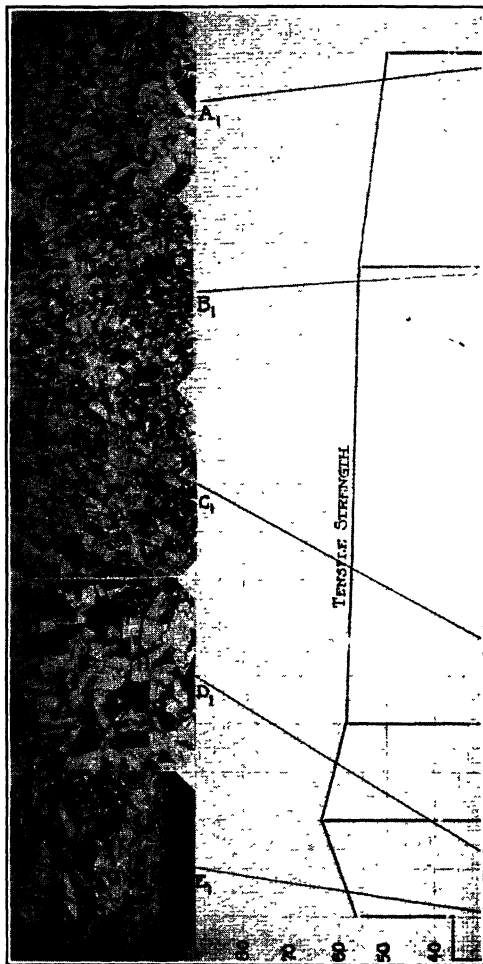
FIG. 21.—STRUCTURE AND TENSILE STRENGTH OF 4.7-IN. CASE AS MADE BY MANUFACTURER NO. 18.

consist of the immersion for 4 hr. in a $1\frac{1}{2}$ per cent. solution of a mercuric salt." Some operators prefer to use mercurous nitrate instead of mercuric chloride; apparently there is no reason why this should not be permitted though some results indicate that the mercuric-chloride is the more searching test. The mercurous-nitrate test consists of the immersion of the case in approximately 10-per-cent. solution for 15 min. Either test is sufficiently severe to indicate the behavior of the case in

storage or irregularities in manufacture. The mercuric-chloride solution is more readily prepared than the mercurous nitrate, but 4 hr. are required for making the test whereas only 15 min. are required for the mercurous-nitrate test.
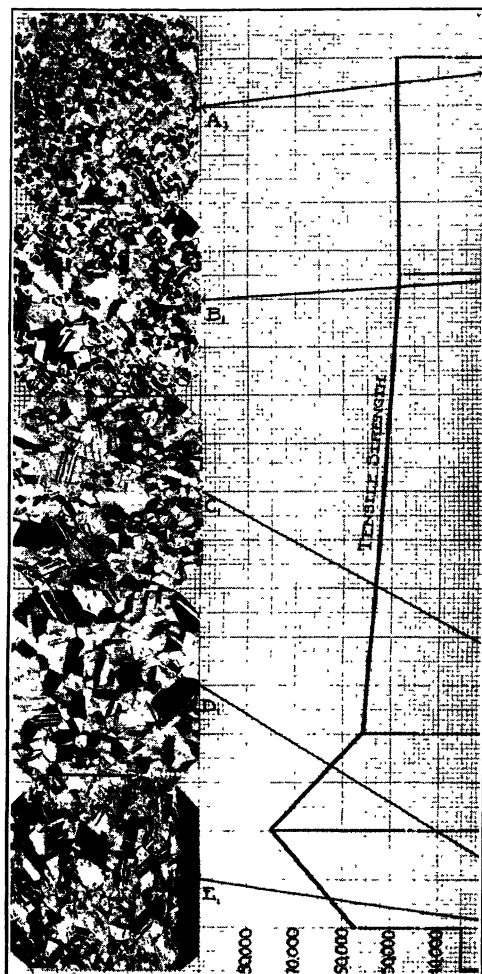
FIG. 22.—STRUCTURE AND TENSILE STRENGTH OF 4.7-IN. CASE AS MADE BY MANUFACTURER NO. 2.

The solutions for making these tests are made up as follows:

Mercurous-nitrate Solution.—Dissolve 200 gm. of mercury in 110 cc. of nitric acid (sp. gr. 1.42), with the aid of heat, so long as there is any action, then gradually add water. As yellow basic mercurous nitrate forms add small amounts of dilute (2:1) nitric acid to dissolve the basic salt, and then add more water (adding small amounts of the dilute nitric

acid as needed) until the total volume is 2500 cc.; then add a small amount of mercury to maintain the solution in the mercurous state.

When using this solution, which is approximately a 10-per-cent. solution containing free nitric acid, the cartridge case is immersed in
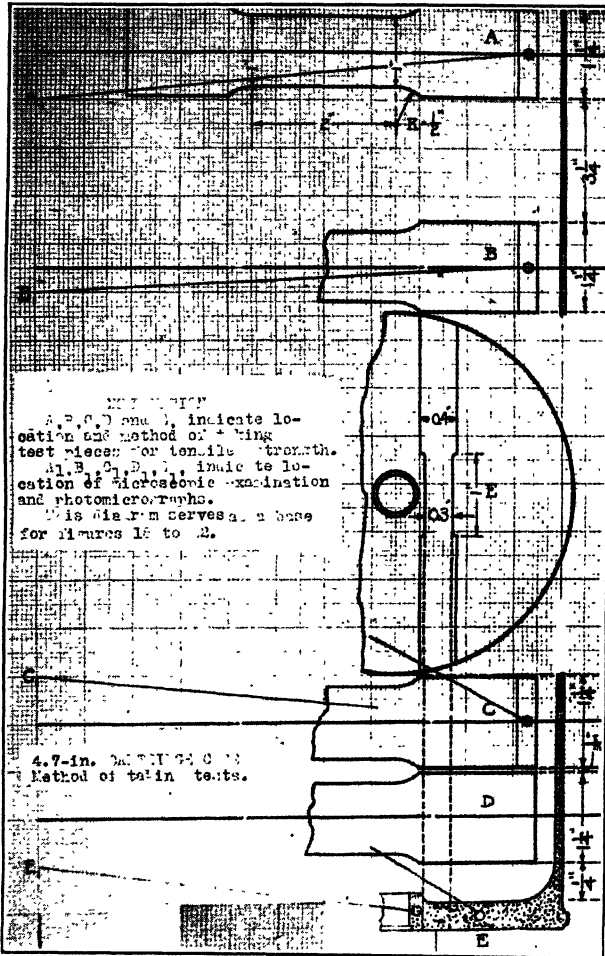
FIG. 23.—METHOD OF TAKING TESTS OF 4.7-IN. CARTRIDGE CASE.

the bath at room temperature for 15 min. and is then removed and examined for cracks.

Mercuric-chloride Solution.—Dissolve 1½ gm. of powdered corrosive sublimate in 100 cc. of water at room temperature. Immerse the test piece in this bath for 4 hr., then remove and examine for cracks.

Nothing is specified about stirring the solution during the immersion

of the cartridge case, the temperature of the solution, maintaining the concentration of the solution during the immersion, and the volume of the solution. Stirring, increased temperature, and maintenance of concentration would increase the severity of the test, which does not seem necessary. Neither is it necessary to note the effect on the case
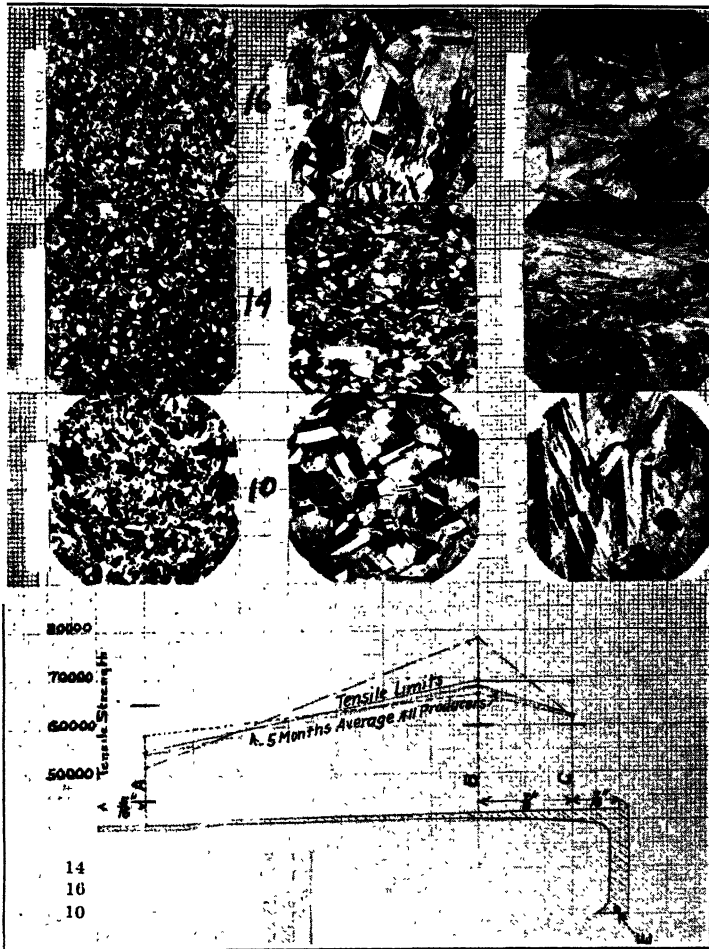
Fig. 24.—Structure and Tensile Strength of 37-mm. Cases as Made by Manufacturers Nos. 10, 14, and 16.

except during and immediately after its removal from the solution. Cartridge-case instructions require that any cracks resulting within 24 hr. be reported.

Early in the war, the mercuric test proved rather severe for cases coming from inexperienced or careless contractors, who protested against this and the physical tests determining the acceptance of the cases.

This protest was due entirely to the inexperience of manufacturers, lack of knowledge of the principles of brass working, determination not to try to overcome mercuric cracks, or the desire to pass product already manufactured. Continued effort to improve the processes of manufacture has resulted in so great an improvement in cartridge cases that
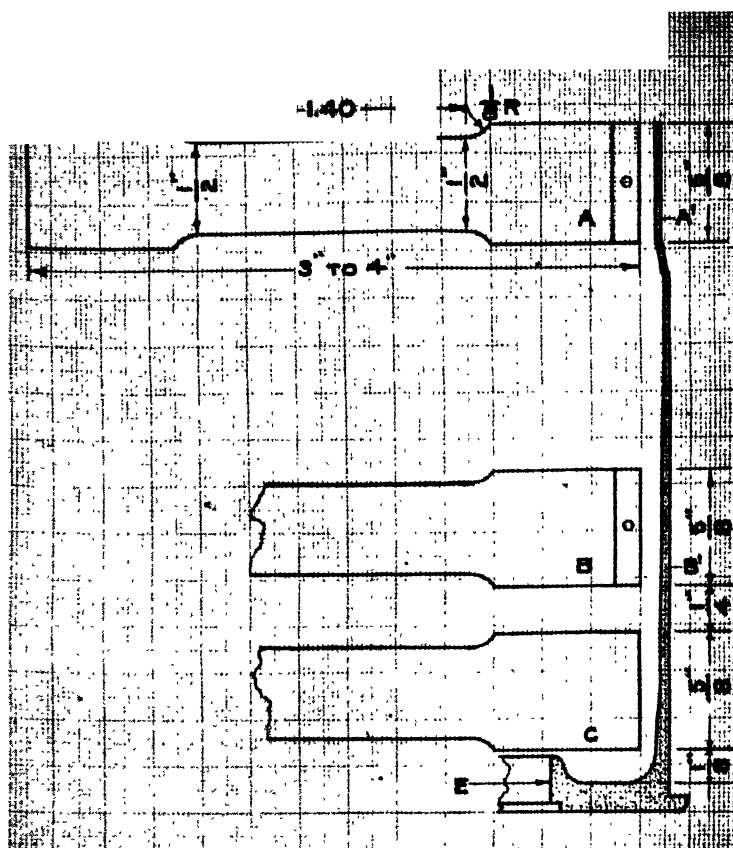
FIG. 25.—METHOD OF TAKING TESTS OF 37-MM. CARTRIDGE CASES.

nearly all manufacturers, including those who protested at first, now believe that this is a most important method of testing.

The first real contribution to the prevention of season cracking appeared, in 1914, in a lecture by Prof. E. Heyn,[6] who showed how strains in the metal are developed by unequal working of neighboring portions to the extent of causing cracks immediately or in time. He then discusses the removal of these strains by heat treating at temperatures of from

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[6] Internal Strains in Cold-wrought Metals, and Some Troubles Caused Thereby. *Jnl. Inst. of Metals* (1914) **12**, 3.

100°–230° C. (210°–450° F.), thus removing the cause of these cracks. These temperatures are much below the recrystallization point and produce no apparent change in the microscopic or physical condition of the metal.

FIG. 26.—STRUCTURE AND TENSILE STRENGTH OF 1.457-IN. SUBCALIBER GUN CARTRIDGE CASE AS MADE BY MANUFACTURER NO. 10.

A direct application of Professor Heyn's method to cartridge cases was made by W. B. Price,[7] who found that cases heated for 1 hr. or more at 260° C. (500° F.) did not crack, though all cases of the same lots that were not heat-treated developed serious cracks when immersed for 4 hr. in a 1½-per cent. solution of mercuric chloride.   Tensile-strength,

[7] *Proc.* A. S. T. M. (Part II, 1918) **18,**

elongation, and microscopic tests revealed no change in the metal from this treatment. It is, therefore, apparent that a low-temperature heat treatment will remove the cause of season cracking and the good cartridge cases thus treated will not deteriorate in storage.

FIG. 27.—STRUCTURE AND TENSILE STRENGTH OF 240-MM. TRENCH-MORTAR CARTRIDGE CASE AS MADE BY MANUFACTURER NO. 15.
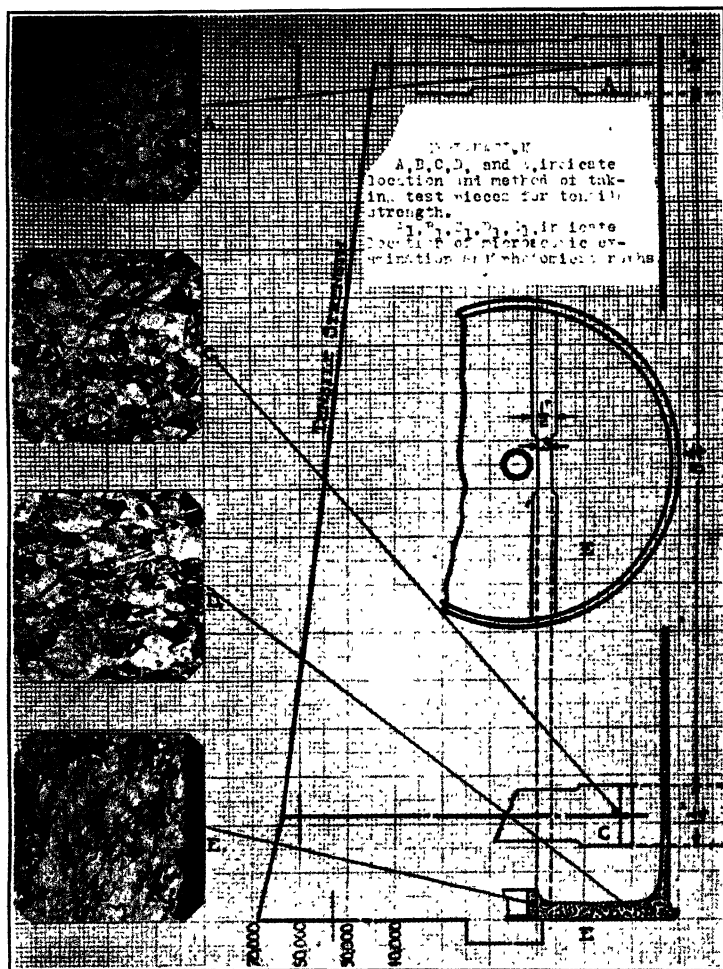
Some manufacturers, inspectors, and other ordnance people claim that a hard case will develop season cracks but that a soft case will not; this is an erroneous assumption. As yet no relation between physical properties and season cracking is known to exist, neither do photomicrographs indicate conditions under which cracks will occur. A season crack is the result of stresses and may occur in both underworked and overworked

cases. Cracks have occurred in the C, D, and E sections when the tensile strength of these sections was as low as 35,000 lb., and have not occurred when the tensile strength was over 90,000 lb.

As a case may be hard or soft from either work or annealing, it is
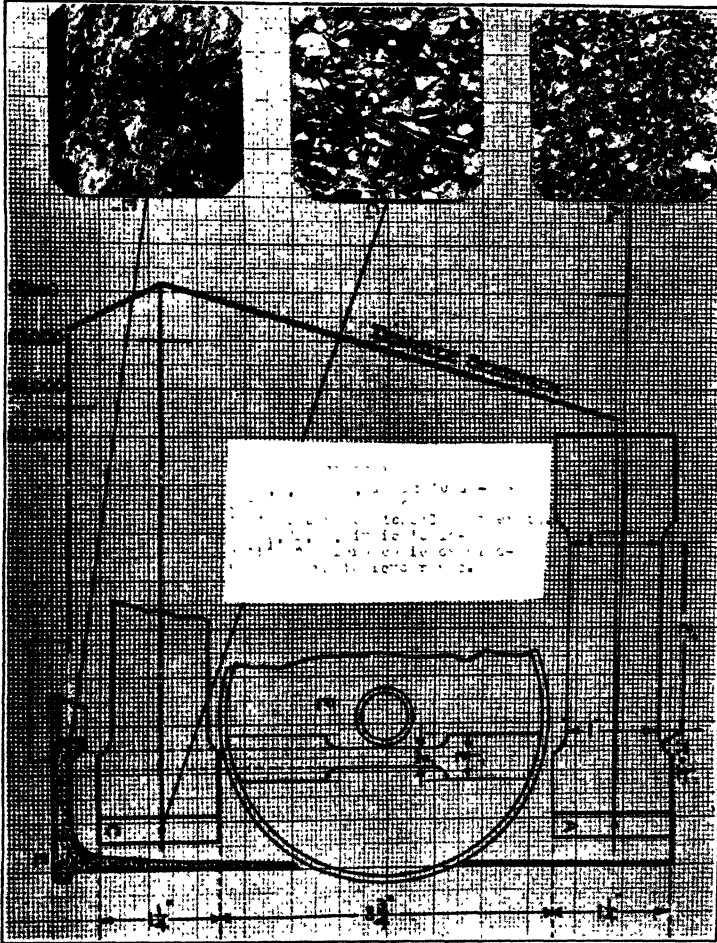
FIG. 28.—RESULTS OF TESTS ON 2.95-IN. CARTRIDGE CASE AS MADE BY MANUFACTURER No. 16.

essential that both work and annealing be considered when the condition of the finished cartridge case is studied. The results of a study of the relation of stress and failure in structural brasses show[8] conclusively that failures occurred only where stress existed.

During the last two years all manufacturers who made an earnest

* P. D. Merica and R. W. Woodward. The Failure of Structural Brasses. *Metal Industry*, N. Y. (Nov. and Dec., 1915) **13**, 459, 495.

attempt to meet the mercuric test were successful, having but a few minor cracks.   Manufacturers who did not meet this test failed because they did not apply simple remedies or refused to change their practice.   An infrequent failure in the mercuric test is to be expected but continued cracking is inexcusable and a reflection upon the manufacturer.
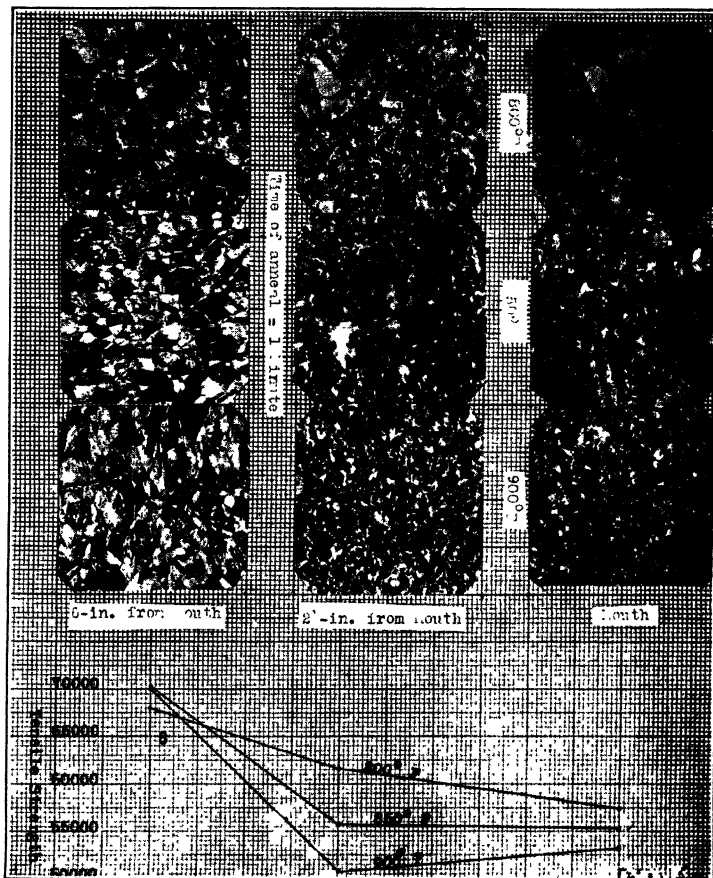
EFFECT OF VARIATIONS IN TEMPERATURE OF THE MOUTH ANNEALING OF 75-MM. CARTRIDGE CASES.

The mercuric test probably results in an amalgamation of the grain boundaries of the metal more rapidly than the grains themselves, thus producing lines of weakness throughout the metal.   If the stress in any part exceeds the stress in a neighboring part, a crack develops as soon as the amalgamation has developed sufficiently to reduce the tensile strength below the stress.   In storage a similar, but slow, corrosion takes place, so that the cracks are developed in a few months or a year.   Instead of

corrosion, the weakening may be due to some change in the amorphous condition between the grains, but as the amalgamation of the mercury test seems to attack the grain boundaries first, this does not affect the reliability of the result of this test.

The mercuric test, besides being an indication of the ability of the
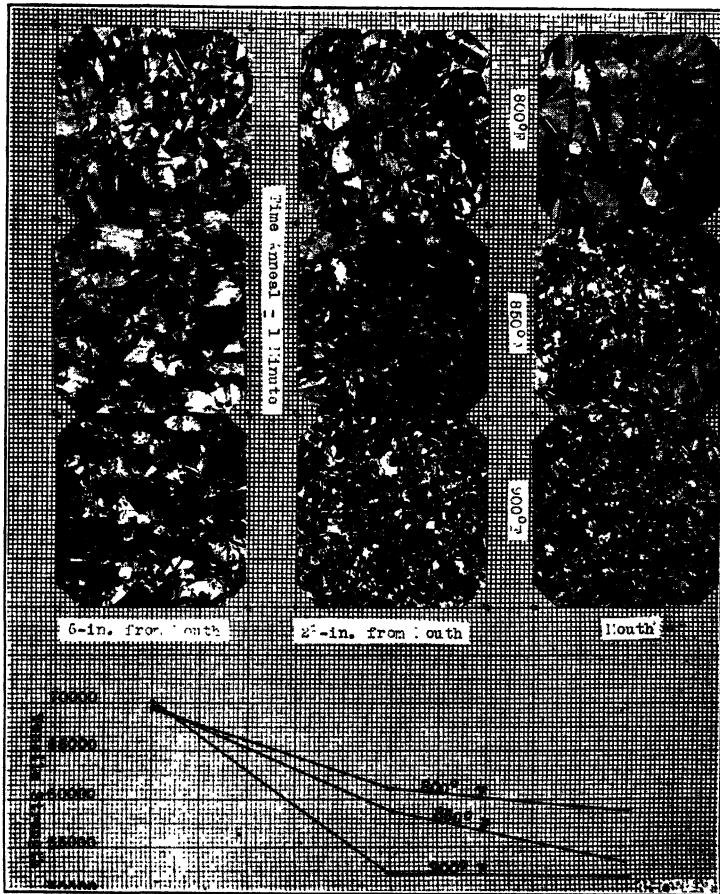
FIG. 30.—EFFECT OF VARIATIONS OF TEMPERATURE OF MOUTH ANNEALING OF 75-MM. CARTRIDGE CASES.

finished case to stand storage, is an excellent control test on the process of manufacture. If at some point in this process a change takes place that sets up serious stresses in the metal of the finished case, mercuric cracks occur. Of course, for determining conditions other than strain other physical tests are necessary. Even a slight wear in properly designed tools or the guides may cause cracks. If any conditions result in more work in a part of a case than in a contiguous part, stresses will

be set up between those parts, which may be sufficient to cause a crack. This was shown by W. B. Price[9] in the case of a cartridge case one side of the mouth of which was thicker than the side diametrically opposite, because of improper cupping tools. The difference in thickness of the

DISK, Brinnell 4⁵.5
Annealed 750°C.70 min.

...PACE OF DISK
HARDNESS ON THE FIN-
ISHED CASE

Cases were drawn from high and low Brinell disks and given identical treatment throughout process of manufacture.

DISK, Brinell 71
Annealed 500°C.68 min.

FIG. 31.

wall caused the thick side to receive more work in the redraw than the thin side. All cases that were eccentric cracked on the thin side, if subjected to the mercuric test, while non-eccentric cases did not crack. Photomicrographs were taken diametrically opposite one another. The one taken from the thick side showed some recrystallization, while no change was evident in the thin side. This shows that the greater the

⁹ *Trans.* (1919) **60,** 455

reduction in brass the lower is its recrystallization point. In this instance all the eccentric cases were made acceptable by increasing the mouth anneal from 425° to 480° C. (800° to 900° F.), when the mercuric cracks were avoided. An increase in the annealing temperature is not necessarily the cure in all instances, as the tensile strength may not be high enough
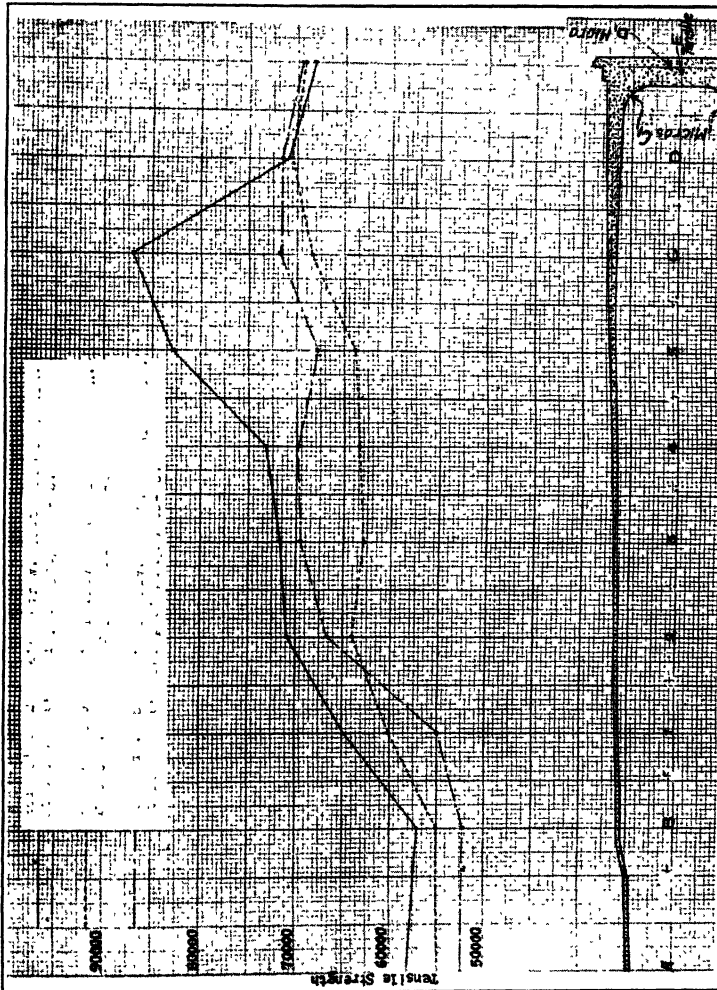
FIG. 32.—CHANGES IN FOURTH REDRAW AND MOUTH ANNEALING OF 75-MM. CASES.

to permit this. Further trouble was avoided by overcoming the eccentricity and continuing the mouth anneal at 480° C. (900° F.).

In Figs. 32 to 35, the results from change in work and mouth annealing are shown. The first process of manufacture gave a much higher tensile strength in the C section than in the contiguous portions of the case, as shown by the solid line in Fig. 32. An increase in the work and the

annealing produced a case with a more uniform working of contiguous parts and less cracking resulted, as shown by the dotted lines. This change greatly lessened the mercuric-crack troubles.

Local strains might be caused by dirt or chips producing scratches or
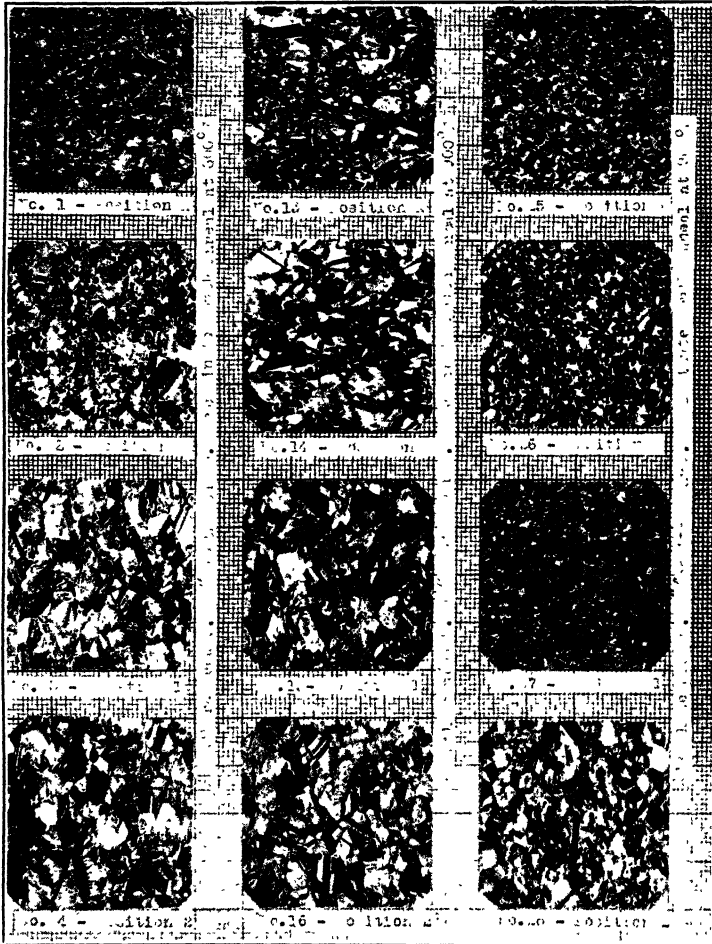
lines of intense work. In the heading operation used by one manufacturer, the metal surrounding the primer hole was under compression and that at the flange under tension. Severe cracks in the head and body resulted from the mercury test, so that it was necessary to change the operations so that the last heading operation produced uniform conditions throughout. The procedure for overcoming season cracks will vary

with the process of manufacture, but some combination of the following is bound to remove the difficulty:

1. Increase the work during the last redraw, thus increasing the effect of the annealing, which is made at the same temperature; higher tensile strength may result and the case thus be too hard and split in the gun.
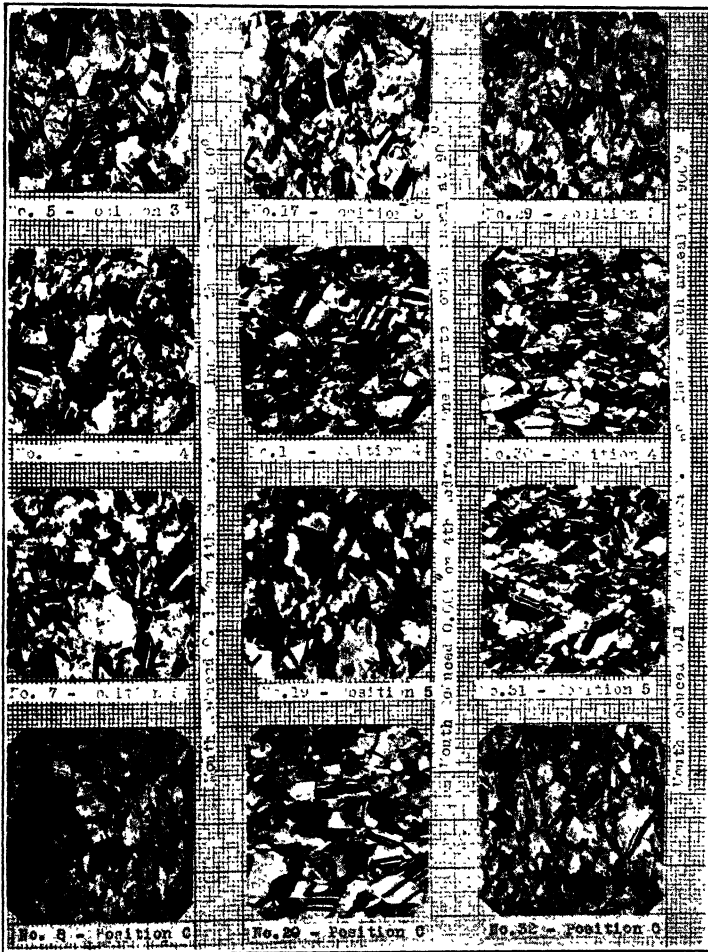
2. Increase the temperature or time of the annealing; this will lower the tensile strength and is not permissible if the case produced already has a low tensile strength.

3. Increase both the work and the annealing; it may be possible to eliminate much of the strain with little change in tensile strength.

4. Redesign the drawing and heading operations so as to remove marked inequalities of working.

5. Give the finished case a low-temperature heat treatment.

The first four properly applied and maintained will prevent most of the season cracking but will not prevent cracking caused by irregularity
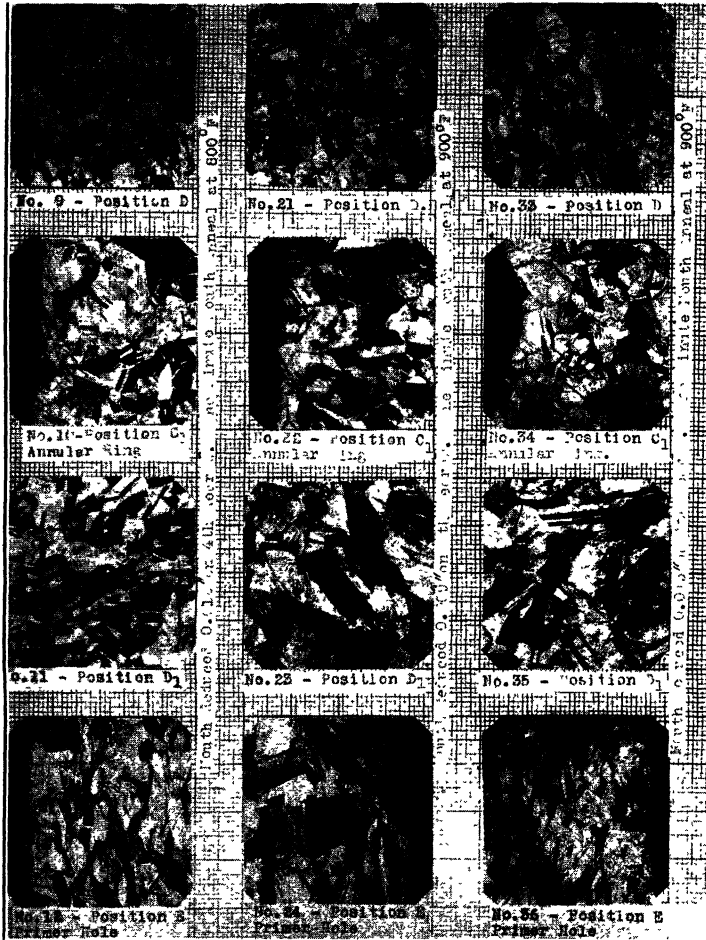
FIG. 35.

in tools and shop practice.   These irregularities will be inversely proportionate to the care taken in the manufacture, but a few cases will always be made that will not stand the mercuric test.   It would, therefore, seem that the addition of the low-temperature heat treatment to the finished cases is desirable.   Such heat treatment can be readily and cheaply applied, and as the temperatures are so low that automatic

devices may be used in the annealing furnaces, the cost of upkeep and operation will be small.

W. B Price[10] says: "A heat treatment of 260° C. (500° F.) for 1 hr. does not cause a recrystallization even on the hardest part of the shell and yet is sufficient to relieve all internal stresses and prevent the case

240-mm Trench Mortar Case, Annular Ring

6-in. Howitzer Case Annular Ring

Tangents AF and CD inlie to limits beyond which annular rings could not extend in acceptable cases.
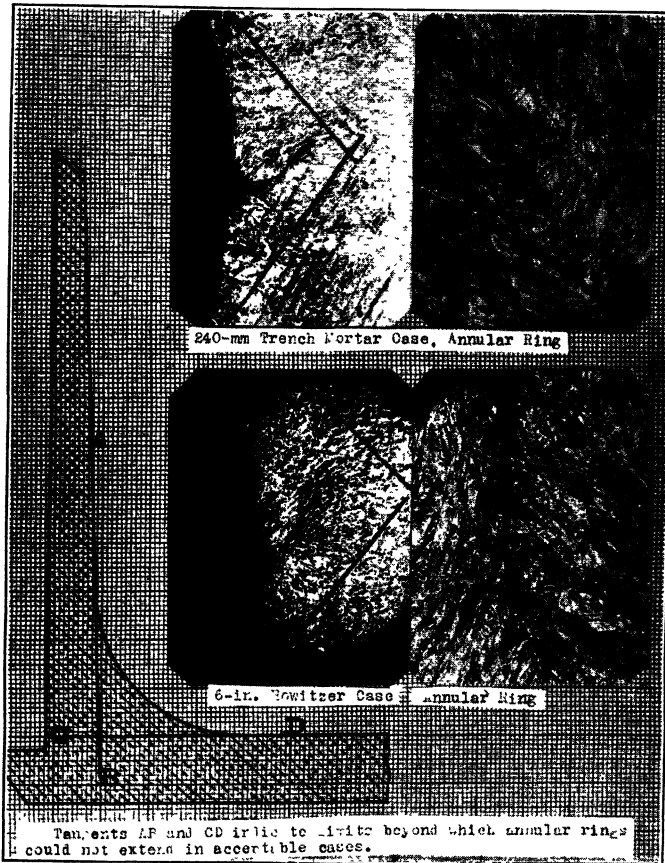
FIG. 36.

from cracking on the mercuric-chloride test." This supports the findings of Heyn[11] and is supported by the ordnance results.

The largest producer of 75-mm. cases furnishes the following data: "The annealing of the cups consists of a 90-min. period in a temperature of 620° C. (1150° F.). After the first redraw, the cups were subjected

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[10] The Prevention of Season and Corrosion Cracking of Brass Artillery Cases by Special Heat Treatment. *Proc.* A. S. T. M. (Part II, 1918) **18.**

[11] Heyn: *Op. cit.*

for 90 min. to a temperature of 600° to 620° C. (1110°–1150° F.); and after the second and third redraws they were subjected each time for 75 min. to a temperature of 600° to 610° C. (1110°–1130° F.)."

The products of combustion from the furnaces in which a temperature of 600° C. (1110° F.) is maintained would, if properly handled, furnish much, if not all, of the heat necessary for the heating of a furnace for final annealing to 260° C. (500° F.).   Also, the 90-min. period required
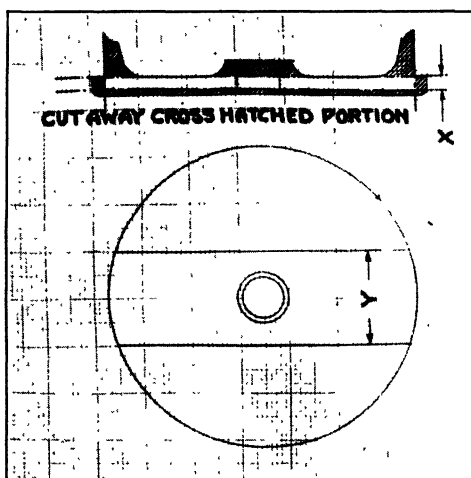
FIG. 37.—LOCATION OF TENSILE TEST FOR PRIMER-HOLE METAL.

Cartridge Case	Thickness Head, Inch	Thickness X Test Specimen, Inch	Width Y Test Specimen, Inch
2.95-in. mountain gun.............	0.2 ±0.01	0.135±0.005	1.00
3-in. field, 1902..................	0.2 ±0.01	0.135±0.005	1.00
3-in. Mark I.....................	0.32 ±0.01	0.255±0.005	1.00
3-in. Mark II....................	0.32 ±0.01	0.255±0.005	1.00
75-mm. field, 1916...............	0.305±0.01	0.210±0.005	1.25
4.7-in. gun, 1906.................	0.40 ±0.01	0.335±0.005	1.00
4.7-in. Mark I...................	0.40 ±0.01	0.335±0.005	1.00
4.7-in. howitzer..................	0.30 ±0.01	0.210±0.005	1.00

for annealing would permit of an equal time in the final furnace without slowing down production.   Mechanical conveyors would work readily at 260° C. (500° F.) so that little labor would be involved in the additional equipment.   The first cost is the main consideration and this would not be excessive because of the comparatively low temperatures involved.

It would seem that annealing at 425° C. (800° F.) for 1 min., after which cracking occurred on the thin side of the case, would do more than a low-temperature anneal for 1 hr., especially as this temperature

is sufficient to produce visible recrystallization on the thick side. Experience[12] indicates that the low-temperature anneal through the longer time may be more effective than the higher temperature anneal through a shorter time, though some recrystallization does occur.

BALLISTIC TESTING

At the present time, the acceptance of artillery cartridge cases depends almost entirely on their ability to pass the proof firing test. This test consists in firing at 12 per cent. excess powder pressure three cases selected
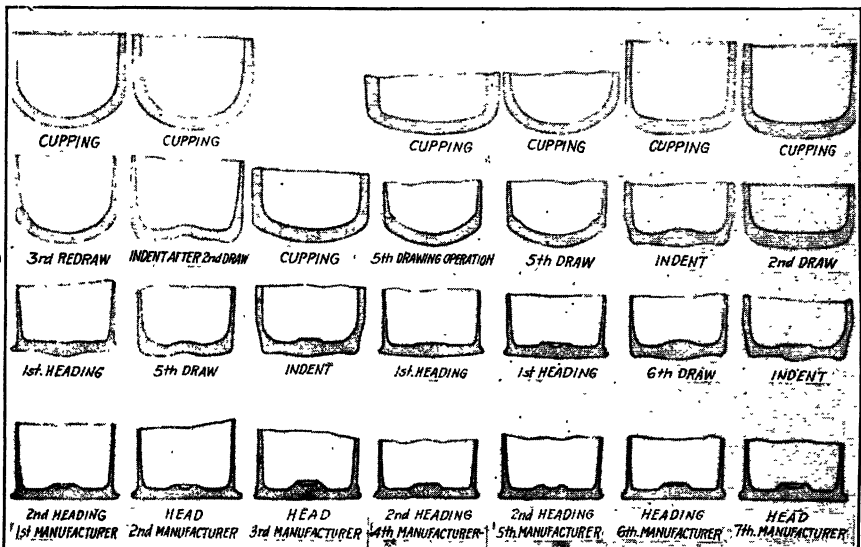
FIG. 38.—METHODS OF WORKING HEAD METAL DURING DRAWING AND IN FINAL HEADING OPERATIONS.

at random from each lot (five for 75-mm. cases), which consists of from 5000 to 25,000 cases. During the war (2353) lots were submitted to ballistic tests, and all were finally passed as satisfactory. Out of this number, 123 lots failed, so that it was necessary to make retests, which consisted in the proof firing or five more cases (as a rule) from the same lot. Of the failures 72.8 per cent. were caused by metal conditions, which were outside of limits set by the Inspection Division; 11.7 per cent. were dimensionally or visually defective and should not have been submitted to the proving ground for tests; 11.7 per cent. were reported as failures and later released as it was found the failures were caused by faulty guns or faulty resizing dies. This last percentage does not include a large number of failures that were reported but withdrawn before record

[12] Merica and Woodward: *Op. cit.*

was made. In one case, after fourteen lots of 75-mm. cases and nine lots of 4.7-in. cases were reported as failures, it was discovered that the failure was caused by improper resizing. Only 3.8 per cent. of the failures could not be explained metallurgically or mechanically; that is, the failures were such that the reports of the metallurgical branch and visual and gaging inspection did not indicate that such failures should

FIG. 39.—METHOD OF TAKING TESTS ON 4.7-MM. ANTIAIRCRAFT CARTRIDGE CASE.

occur. As only three cases out of 25,000 are selected for metallurgical tests and only five cases are selected for ballistic tests, it is likely that those selected in either case were not representative of the lot.

No lots of cartridge cases were finally rejected, as upon ballistic retest all were accepted. It is, therefore, evident that a considerable quantity of questionable material was accepted. Also a number of lots of cases did not meet the suggested metallurgical specifications, but

passed the ballistic tests. Inasmuch as 72.8 per cent. of the failures were caused by metallurgically defective cases, according to suggested specifications, and only 3.8 per cent. were within those specifications, it is evident that the acceptance of material on ballistic tests alone means the acceptance of questionable material. Further, the fact that such a
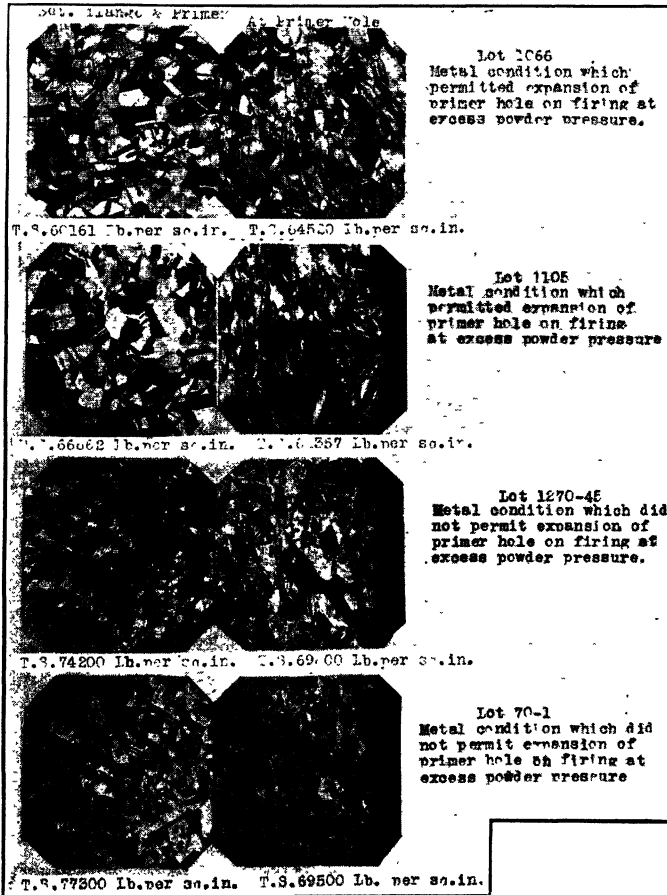
FIG. 40.—PRIMER-HOLE CONDITIONS OF 75-MM. CASES AS MADE BY MANUFACTURER No. 13.

large number of failures caused by faulty guns and resizing have been temporarily recorded, makes it apparent that the ballistic test may be decidedly unfair; at any rate it is uncertain.

Nearly all of the manufacturers produced cases that were within prescribed limits and were generally free from cracks after the immersion in mercuric-chloride solution. The results shown in the accompanying illustrations justify the employment of physical tests as meanes of determining acceptance and rejection. These tests also give far more

information as to the quality, life, and behavior of the cartridge case than any ballistic tests that may be applied.

The cost and time consumed are also much less. Physical and metallographic testing of a lot of cases in the laboratory will require the time of one man in the machine shop for 8 hr., two men in the physical
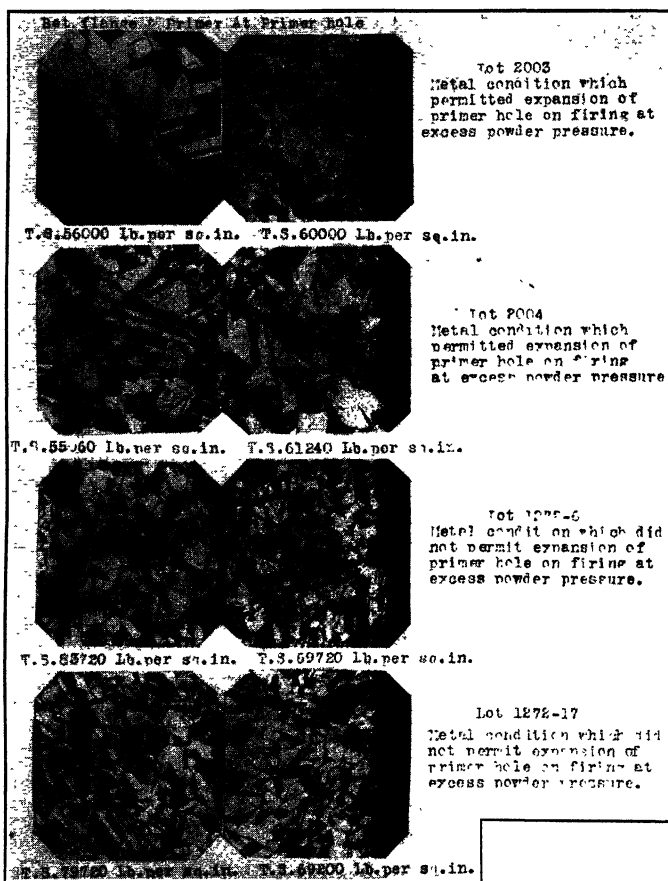
FIG. 41.—PRIMER-HOLE CONDITIONS OF 75-MM. CASES AS MADE BY MANUFACTURER NO. 5.

testing laboratory for 3 hr., and one man in the metallographic laboratory for 4 hr., or a total of 18 man-hours. Allowing $1 per hour for this labor cost and 150 per cent. for overhead expense, materials, etc., the cost of testing a lot of cases will be $45.

The cost of the firing tests required on a lot of cases, including cost of explosive, proof projectile, resizing of case, and overhead, is estimated, by the proving ground, as follows: 37-mm. cases $23, 75-mm. cases $140, 4.7-mm. cases $280, 240-mm. cases $480. The time required for this

testing, after the cases are received at the proving ground, is not much less than that required for the metallurgical tests. To this time must be added the time required for shipping the cases to the proving ground and the time that must elapse between the proof-firing and the receiving at the manufacturing plant of a report of the results.
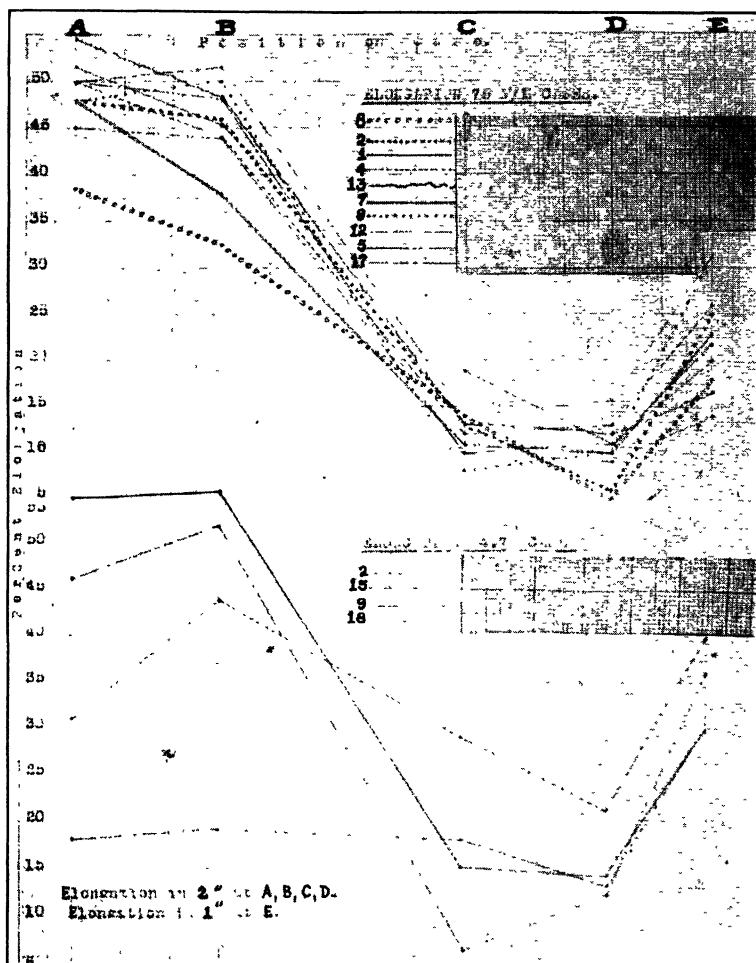
FIG. 42.—ELONGATION OF CASES AS MADE BY VARIOUS MANUFACTURERS.

The establishment of strict metallurgical specifications will impose no hardship on the manufacturer; in fact, these specifications will be to his advantage as he will then have something definite to which to work. It is not contended that all ballistic tests of cartridge cases should be done away with, as such tests are desirable on doubtful material and as all product that is not specified by experience requires these tests.

MANUFACTURING DIFFICULTIES

The most important of the manufacturing difficulties encountered were: expansion of the primer hole, occurrence of annular rings, and appearance of visual defects in the metal. The expansion of the primer hole is an old difficulty. Old cartridge-case specifications required mandreling of the primer hole to toughen the metal, so that the primer
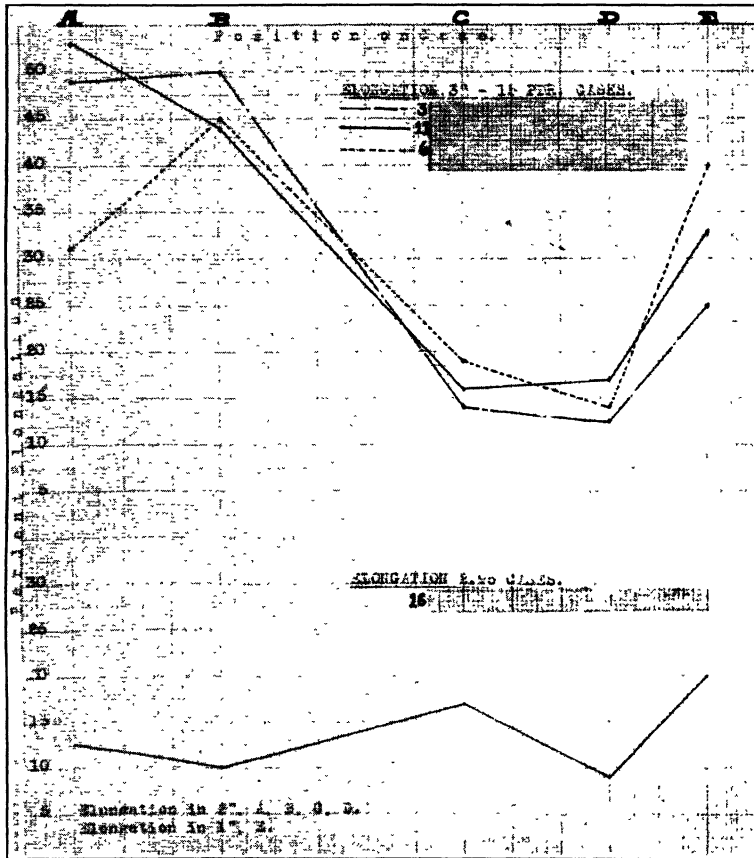
FIG. 43.—ELONGATION OF CASES AS MADE BY VARIOUS MANUFACTURERS.

hole would not expand during firing. It was thought by some that the mandreling was absolutely necessary to provide sufficient tensile strength. The extra operation did not appeal to manufacturers contemplating quantity production, so many of them resorted to other methods of working the head. Indenting the cup or head of the case between the draws was a rather popular method of obtaining this strength, but like the mandreling this required an extra operation and as it came early in the process its effect on the finished head was not marked.

Some manufacturers adopted various methods of causing the metal to flow in the heading operations, thus doing away with indenting and mandreling. Sections E in Figs. 22 and 11 illustrate the metal of the head with mandreled primer hole; section E, in Figs. 20 and 3, illustrate the condition where no mandreling took place. A comparison of the
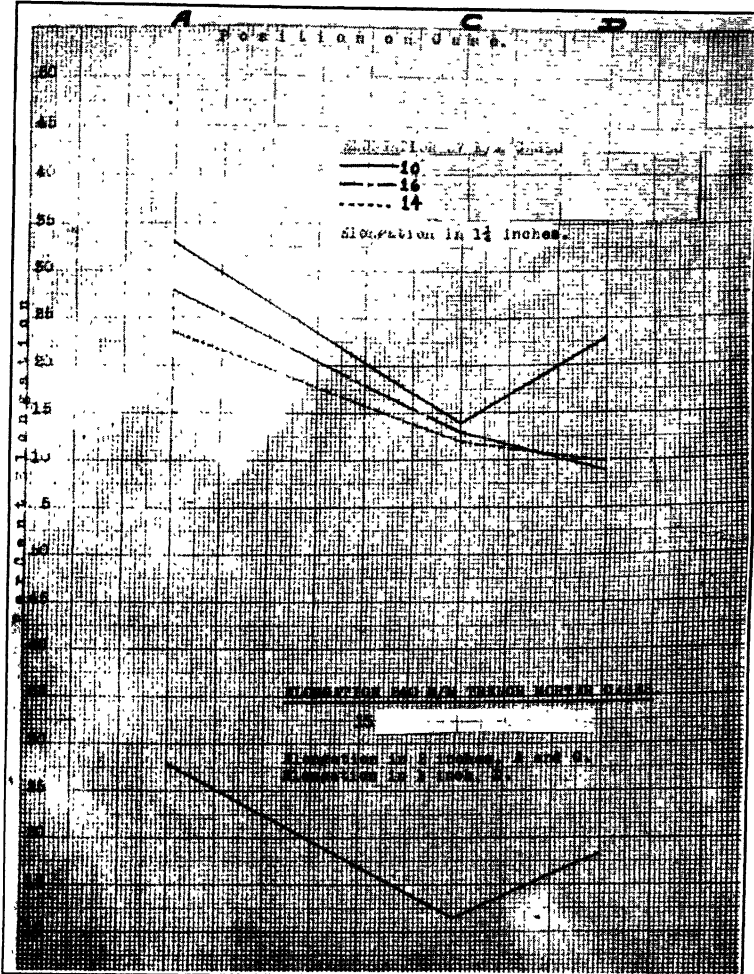
FIG. 44.—ELONGATION OF CASES AS MADE BY VARIOUS MANUFACTURERS.

tensile strengths and photomicrographs shows that all the methods give the desired results. The various methods of working the head metal are shown in Fig. 38.

The latest specifications state that any method of mandreling of indenting that will sufficiently harden the head to prevent expansion or the primer hole may be used. Many manufacturers who use neither

mandreling nor indenting are obtaining just as satisfactory cases as those who use these methods. The trouble experienced from the expansion of primer holes was caused probably by the low tensile strength in the D and E sections or the excess powder pressure used in the ballistic tests.

The photomicrographs shown in Figs. 40 and 41 are taken from sections of cases that did and that did not expand at the primer hole. The difference in the structure is not as apparent as in the tensile strengths. A high tensile strength of the metal in the D section, or heavy portion of the head, is just as essential as a high tensile strength in the metal immediately surrounding the primer hole. Many of the failures may have been due to the excess powder pressure used at the proving grounds, as other cases from the same lots passed the tests at lower powder pressures. However, it would seem advisable to maintain the higher tensile strengths in order to provide a sufficient factor of safety. Experience has shown that it is easily possible to make cases of sufficient tensile strength to meet the higher powder pressures and still have a product that will stand storage.

Sections of cases made with various heading operations are shown in Fig. 38. There is very little difference in either the microstructure or the tensile strength of cases made by the various methods. Operations in the first drawing stages have scarcely any effect on the finished case, if the work during these operations has been sufficient to cause recrystallization in the annealing process between draws. A temperature of 650° C. (1200° F.) for 1 hr. will completely anneal the brass and remove the effect of any indent or head working in the previous steps in the process. A number of manufacturers use indenting operations, but these are followed by annealing, which removes the effect of the working of the metal by this operation. The greater working of the head metal where an indent is used is brought about by the final heading from the indent instead of from the cup form. No annealing follows the final heading operation.

Annular rings, metal folds, or cold shuts in the curved part of the cartridge case, between the head and the wall, were a source of trouble in a few instances at the beginning of the war. These rings are entirely due to improperly designed tools; changes in the design of these and in the manufacturing processes soon overcame the difficulty. Fig. 36 illustrates cracks that were found in early manufacture. The Ordnance Department arbitrarily specified that cases in which the annular rings extended beyond tangents drawn to the curved portion of the case from the inside of the wall and the head should be rejected. As these rings are the result of wrongly designed tools and have nothing to do with the metal condition, no further discussion is necessary here.

Spills, blisters, and scratches in the finished case are due to various causes and appear in widely different sizes and conditions, so many

questions as to how they should affect the acceptance of cases have arisen. The Ordnance Department attempted to formulate rules as to what should be accepted but the variations are such that acceptance must rest largely on the judgment of the inspector. Smooth, sound disks, free from defect, should draw into smooth, sound cases and the manufacturer should study his process if this result is not obtained. Good casting and rolling conditions produce proper disks and, therefore, defects in cartridge cases should not occur.

If a blister, blowhole, or spill occurs in the disk, it will be elongated and drawn out, forming a long slender line visible on the inside of the case, if near the surface; if under the surface, it will be hidden from sight. It is likely that cartridge cases that show a poor surface contain similar defects beneath the surface; this is certainly true if the defects occur other than on the surface of the disk. The outside of the case is smoothed over in the redrawing dies so that most defects are obliterated, as far as the eye is concerned.

If, in one of the redraws, the metal is scratched, this scratch will be drawn out so as to appear in the final product very much as the result from a blister in the disk.

If the redrawing tools are so designed that the metal on the outside of the cup is advanced much beyond the metal of the inside, or vice versa, the rough edges of the disk will be drawn into the case, producing in the mouth of the case staggered cracks that cannot be removed in the trimming operations. In a few instances these have resulted in a complete breaking away of the mouth.

Scratches, pits from embedded metal or dirt, dents from furnace tongs, or other defects are also objectionable, as they produce local working and strain. This strain may not be sufficient to cause season cracking or cause rejection of the product, but all of these tend toward poor product and should be avoided. The shop practice and not the metal is usually at fault when these defects appear and, therefore, the remedy usually is to be found on the premises.

Rough handling of the cases in either the finished or unfinished state, dents or folds are apt to cause trouble. If a case is dented, particularly when annealing does not follow the next working, the metal is very apt to be overworked by such dents either in that or a following working, causing cracks. One manufacturer was allowing carelessness in the handling of the hooks used for drawing the cases from the annealing furnace, so that many sharp dents occurred, which resulted in many rejections because of radial cracks developing in later steps in the process. In a few instances, thin cracks showed first on firing in the ballistic test. Dents from throwing the cases against each other will produce similar results.

If too high an annealing temperature is used, particularly preceding

the last redraw, a coarse granular structure will result leaving a rough, or orange-peel, effect in the final case. While this might not be sufficient to cause a failure of the case in the ballistic test, it produces unsafe and questionable material, which should not be accepted.

Defects are thus caused by poor manufacture and defective disks; and though some are bound to occur, care should be taken to distinguish between those that are serious and those that are not serious. It is necessary that projections or roughness be removed from cases but the repairing by digging and scraping out defects or scratches is questionable. This practice weakens the case through the removal of metal and little information is gained other than that showing on the surface. No doubt, during the war many "visually defective" cases were rejected that should have been accepted. Foreign practice permits extensive repair by the brazing of very defective cases and while this may be going too far it is an indication that the United States inspection is too rigid.

PHOTOMICROGRAPHS AND CHARTS

The accompanying illustrations were selected from a large mass of material that accumulated during the war. Most of these charts and photomicrographs have been explained, but they are mentioned here to facilitate the study of the plates.

Fig. 1 represents the comparative tensile-strength tests of typical 75-mm. cartridge cases selected from each manufacturer. A diagram at the bottom shows the location on the case of the test pieces and the ordinates indicate the tensile strength in pounds per square inch. The upper and lower limits used by the Inspection Division of the Ordnance Department are shown by the heavy lines. It will be noted that the tensile strengths of all these bars fall within these limits and that the greatest variation occurs in the C and D sections. This is to be expected as it is on these sections that the least annealing is done after the heading operations, and because of the variation in the amount of work in the different processes. The heavy line in the center indicates the average tensile strength of all lots of 75-mm. cases produced during 4 months.

The method of taking test pieces of 75-mm. cartridge cases is shown in Fig. 12. At A, B, C, D and E are shown the size and location of test pieces and at A_1, B_1, C_1, D_1, and E_1 the location of the microscopic examination. This examination was made on pieces cut from the grip ends of the test pieces A, B, and C. Test piece E and section D_1 show the properties of the head metal. Section E_1 shows the metal condition at the primer hole. There is no tensile-strength test piece for the primer-hole metal and no microscopic examination was made at D. The microscopic examination of A_1, B_1 and C_1 is made on the surface of the case, and of E_1 and D_1 on sections. Figs. 2 to 11 show photomicrographs at

37.5 diameters and the tensile strength of sections A, B, C, D, and E of typical 75-mm. cartridge cases of various producers.

Figs. 13 to 17 show tensile-strength and micrographic data on 3-in. 15-pdr. antiaircraft cases. Figs. 18 to 23 apply to 4.7-in. gun cartridge cases. Fig. 24 gives data regarding the 37-mm. case. The tensile strength of sections A, B, and C and photomicrographs of sections A_1, B_1 and E_1 are shown. The short heavy lines drawn above and below the tensile-strength lines indicate the tensile-strength limits given by the Ordnance instructions. In section B, cases from different manufacturers show considerable variation. The average tensile strength of all lots for 5 mo. is shown, together with those of typical product. Fig. 25 shows the locations, size, and method of taking tests on the 37-mm. cartridge case.

Fig. 26 shows the results of tests of the 1.457-in. subcaliber gun cartridge case. The short heavy horizontal lines indicate the tensile-strength limits given in the instructions. Fig. 27 gives the results of tests of the 240-mm. trench-mortar cartridge case. Fig. 28 gives results of tests of the 2.95-in. cartridge case.

Figs. 29 and 30 show the effect of variations in temperature of the mouth annealing of 75-mm. cartridge cases. The effect of annealing for 1 min. at 425° C. (800° F.) and 480° C. (900° F.) is shown at the mouth, $2\frac{1}{2}$ in. from the mouth, and 6 in. from the mouth. The condition shown from the 480° C. (900° F.) anneal is considered ideal; the 460° C. and 480° C. anneal increased the tensile strength 6 in. from the mouth over the same zone at the 425° C. anneal.

Fig. 31 shows the results of tests on a very soft and a very hard disk and the cases resulting from these disks after passing through identical conditions in the process of manufacture. The hard disk (71 Brinell) is at the bottom and the structure of its case at the right; the soft disk (42.5 Brinell) is shown at the top, and the structure of the case at the left. The tensile strengths of the two cases are shown in the center; the structure and tensile strength of both cases are very similar.

In Figs. 32 to 35 are shown the results obtained by changes in the fourth redraw and in the mouth annealing of 75-mm. cartridge cases, all other conditions remaining constant. Fig. 36 illustrates, by photomicrographs at 5 and 37.5 diameters, annular rings that occurred in some cases made at the beginning of the war. The diagram at the left and the lines on the photomicrographs at 5 diameters are drawn tangent to the fillet of the case where it forms the head and the wall. Annular rings extending beyond these lines caused the rejection of the lot.

Fig. 37 shows the method of testing the tensile strength of the case at the primer hole of various size cases. This method was suggested in the instructions to inspectors but was not required. At the top is shown

the location of the test piece; the table at the bottom indicates the dimensions of the test piece for the different sizes of cases.   Fig. 38 shows the various methods of working the head metal during the drawing and in the final heading operations.

Fig. 39 illustrates the location, size, and method of taking test pieces on the 4.7-mm. antiaircraft cartridge case.

In Figs. 40 and 41 are shown the microstructure of the cases that expanded at the primer hole on firing, also of the cases that did not expand. The tensile strengths are also given.

In Figs. 42, 43, and 44 are given the elongation of metal in the typical cases of various sizes.   The elongation of sections A, B, C, and D is given as the per cent. in 2 in., while the elongation of section E is the per cent. in 1 inch.

CONCLUSIONS

1. Tensile-strength requirements are now sufficiently established by the experience during the war to warrant their being placed in specifications and used in deciding the acceptance or rejection of cases.

2. The mercuric test has been met by reliable manufacturers, is a good test for control of the manufacturing process, is a sure indication of the behavior of the case in storage, and indicates a metal condition that cannot be detected by chemical, physical, or microscopic tests.   It should, therefore, be made a definite requirement in the specifications for cartridge cases.

3. Microscopic examination should be made of a portion of the cases submitted for tensile-strength tests, also of samples of all questionable material.   Photomicrographs are essential as a record and for purposes of comparison; their number will depend on conditions.

4. Ballistic tests alone are not a sufficient or proper test of cartridge cases.   Except on material that is questionable, metallurgical tests taken with proper dimensions and visual inspection are sufficient and more reliable than ballistic tests.

5. The present requirements and specifications for cartridge-case disks are satisfactory.

6. Indications are that hardness tests, made by the baby Brinell machine without cutting up the cartridge case, might be substituted for the tensile-strength test.   Such tests would increase the sample, decrease the work, and save the case.

7. The low-temperature anneal (below recrystallization temperatures) seems so certain of overcoming all season cracking that its employment in the manufacturing of cartridges should be seriously considered.

8. More judgment should be used in rejecting cartridge cases because of slight visual defects.

DISCUSSION

G. E. DALBEY, Waterbury, Conn.—Is the mercuric-chloride test adaptable to 60–40 brass with a small amount of lead? In the case of tubing of other alloys would it show the probability of cracking after considerable drawing?

WM. H. BASSETT, Waterbury, Conn.—The mercurous-nitrate test has become a standard test for brass rods in general and is being used on almost all of the copper alloys to determine the liability toward season cracking.

G. E. DALBEY.—Is it reliable for tubing made of brass and copper?

WM. H. BASSETT.—I do not know of its being applied to copper tubing, or as a regular test to brass tubing. A large amount of testing has been done on rods and shells and to some extent on brass tubing with mercurous nitrate. On rods to be used for construction purposes, the mercurous nitrate test has been used in comparison with weathering tests; they have given the same results with the exception that the mercurous-nitrate test seems to be rather more severe. All of the samples that broke in the weathering test failed in the mercurous-nitrate test, but some of the samples that failed in the mercurous-nitrate test did not fail in the weathering test. Undoubtedly the mercurous-nitrate test would be useful for tubing and could be applied to almost any of the material made from the copper-zinc alloys.

The cracking or breaking of the material is due to internal strains set up by cold-working operations acting in conjunction, usually, with some corrosive action. The mercurous-nitrate test is, of course, a rapid corrosion test and attacks particularly between the grains or crystals. When the material fails on account of internal strains the breaking is generally brought about partly by the direction of the strain. Strains that act in a lengthwise direction will not cause the cracking of tubes or shells, while strains in the direction of radial expansion are quite certain to cause failures.

C. H. BIERBAUM, Buffalo, N. Y.—Two reagents have been spoken of, mercuric chloride and mercurous nitrate, which has also been spoken of as mercuric nitrate; is mercuric nitrate used at all?

WM. H. BASSETT.—I think not.

S. TOUR.—As to the question whether or not season cracking results from excessive work, it has been quite definitely shown that season cracking is not due to excess work but to strains caused by variations in work. A piece may have had but very little working and be in a·strained condition. In certain cartridge cases referred to in the paper under discussion,

season cracking occurred in sections having a tensile strength of only 30,-000 lb. per sq. in., representing slightly worked material, whereas sections having a tensile strength of 80,000 to 90,000 lb. per sq. in. did not season crack. In other words, the amount of working is not the determining factor in season cracking. Strain in the metal, whether it has been heavily or slightly worked, will result in season cracking.

WM. H. BASSETT.—Our experience is the same. The amount of work is not an essential factor in the season cracking. It is rather the direction of the strains and the condition of the strained body. The material may be worked until it has a very high tensile strength, the working may be very severe, and yet season cracking may not occur. On the other hand, if the working leaves the material under compression so that the stresses tend to produce radial expansion, the metal may readily split. Cracking that takes place in annealing is usually called "fire cracking" because it is caused by rapid expansion in heating. The grain shown by fire cracking is similar to that shown by season cracking. The cracks usually proceed between the crystals or grains, but it is not uncommon to have tubes fire crack in a manner such that some of the grains are ruptured. We suppose that this is because the fire cracking takes place more rapidly and in this way some of the grains are broken across as they are in a ruptured specimen produced by physical strain rapidly applied as in a testing machine.

Physical Properties of Cartridge Brass

By C. Upthegrove* and W. G. Harbert, Ann Arbor, Mich.

(Rochester Meeting, June, 1922)

During the past year considerable work dealing with physical properties of cartridge brass was done at the University of Michigan in coöperation with the Ordnance Department of the U. S. Army. This paper gives some of the data bearing on the effect of the anneal, previous to the final reduction or rolling, on the physical properties, particularly the hardness.

Various phases of the subject of physical properties of cartridge brass have been presented by different investigators. Mathewson and Phillips[1] refer to the work of Grard. in 1909, as the first systematic presentation of the relation between temperature and time of anneal, changing of physical properties and microstructure, and the previous reduction by cold rolling. They discuss the relation between time and temperature of anneal and the previous reduction by cold rolling; also the relation between physical properties and grain size of annealed brass. The degree of deformation previous to the annealing was found to determine the temperature at which recrystallization and corresponding changes in physical properties would take place. Bassett and Davis,[2] in establishing the relation between grain size and Brinell hardness, found the hardness for low-temperature annealing to be dependent on the previous degree of reduction, while for anneals above a certain temperature the hardness after the anneal was independent of the degree of reduction. Actual differences in hardness, due to the degree of reduction, disappear only on annealing at 600° C. or above; though the differences for metal subjected to different degrees of reduction and annealed at temperatures between 350° and 600° C. were found to be relatively small. In their determination of the relationship of grain size to Brinell hardness, Bassett

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* Assistant Professor of Metallurgical Engineering, University of Michigan.
[1] Recrystallization of Cold-worked Alpha Brass on Annealing. *Trans.* (1916) **54,** 608.
[2] Comparison of Grain-size Measurements and Brinell Hardness of Cartridge Brass. *Trans.* (1919) **60,** 428.

and Davis used 70–30 brass that, previous to its final reduction, was annealed at 650° C.   F. G. Smith, investigating grain growth in alpha brass,[3] subjected the brass to anneals at 450°, 550°, 650°, 750°, and 850° previous to the final reduction or deformation.   As a result of his investigation, he concluded that the temperature of recrystallization and the resultant grain size of the final anneal are dependent on the temperature of the anneal previous to the deformation.   This conclusion indicates that both the deformation previous to the final anneal and the anneal previous to the deformation determine the physical properties of cartridge brass.

In the preparation of the cartridge brass four bars were cast of the following composition:

	No. 1	No. 2	No. 3	No. 4
Copper, per cent................	68.42	68 09	68 54	68.45
Lead, per cent.................	0 03	0 02	0.02	0.02
Iron, per cent..................	0 02	0 03	0.02	0 03
Zinc, per cent.................	31.53	31.86	31.42	31.50

The bars were overhauled direct from the casting shop, rolled and annealed as follows:

BAR NO.	ROLLED TO, INCH	ANNEALED AT, DEGREES C.	ROLLED TO, INCH	ANNEALED AT, DEGREES C.	ROLLED TO, INCH
1	0.516			650	0.325
2	0.560	550	0 434	650	0.273
3	0 560	550	0.348	650	0.213
4	0.560	550	0.267	650	0.168

It will be noted that the temperature of the anneal (650° C.) previous to the last rolling, and the degree of reduction,, 37.1 per cent., was the same for all four bars.   The gages 0.325, 0.273, 0.213 and 0.168 in. are such that the reductions necessary to bring all the bars to 0.134 in. were equivalent to reductions of 8, 6, 4, and 2 numbers B. & S., or 58.8, 50.9, 37.1, and 20.2 per cent., respectively.   For the purpose of annealing previous to the final reduction, each bar was divided into three parts giving three lots A, B, C, each containing parts of bars 1, 2, 3, and 4.

Temperatures of 550°, 650°, and 750° C. were chosen for the anneals previous to the final reduction, as representative of the maximum, average, and minimum annealing temperatures.   In carrying out the annealing operations there was, however, some variation from these temperatures so that actual annealing temperatures were probably nearer to 560°, 570°, 650° and 750° C.   The difference in structure caused by the anneals previous to the final reduction is shown in Figs. 1, 2, and 3.   Following the anneals at 570°, 650° and 750° C. the bars were all rolled to 0.134 in.   Following the reductions, the bars were cut into 10- by 1-in. strips.

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[3] Grain Growth in Alpha Brass.   *Trans.* (1920) **64**, 159.

Bar	Annealed, Degrees C.	Rolled		Reduction	
		from Inch	to Inch	B. & S.	Per Cent.
1A	570	0.325	0.134	8	58.8
2A	570	0.273	0.134	6	50.9
3A	570	0.213	0.134	4	37.1
4A	580	0.168	0.134	2	20.2
1B	650	0.325	0.134	8	58.8
2B	650	0.237	0.134	6	50.9
3B	650	0.213	0.134	4	37.1
4B	650	0.158	0.134	2	20.2
1C	750	0.325	0.134	8	58.8
2C	750	0.273	0.134	6	50.9
3C	750	0 213	0.134	4	37.1
4C	750	0.168	0.134	2	20.2

From this point the annealing treatment was similar to that used by Bassett and Davis, the bars being annealed at 50° intervals from 200° to
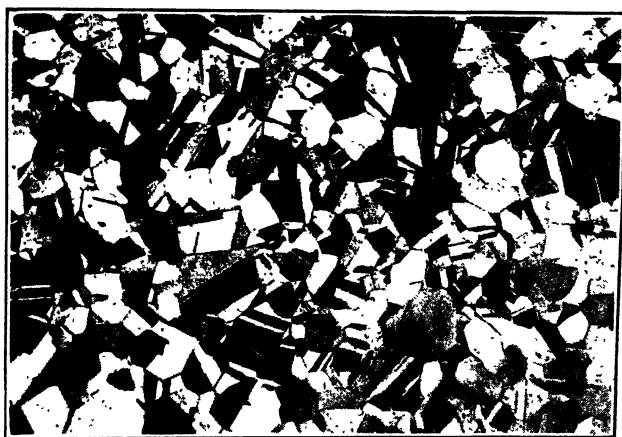
FIG. 1.- CARTRIDGE BRASS ANNEALED AT 570° C. PREVIOUS TO FINAL REDUCTION. BRINELL 56.6.   ×76.

750° C. and at 25° intervals from 250° to 325° C.   A 3-in. piece was cut from each bar, for microscopic examination and Brinell tests.   The bars and 3-in. pieces were then arranged in bundles of six, as 2A, 2B, and 2C bars, and annealed in an electric furnace.   The pieces for the Brinell test were so placed as to occupy the same relative position in the furnace as the centers of the test bars; the hot junction of the thermocouple also was at the center of the bundles.   The samples were brought to the desired temperature, held 30 min., and quenched in water.

Brinell hardness tests, using a 500-kg. load and 10-mm. ball, were made on each 3-in. piece. As two test bars were used in the tensile-strength tests, the Brinell hardness numbers for any one anneal are the
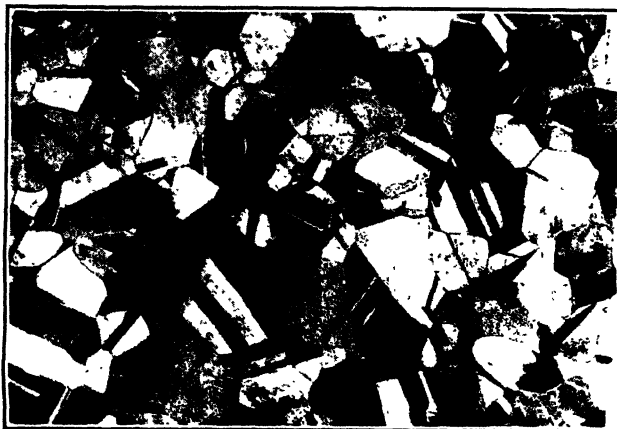
FIG. 2.—CARTRIDGE BRASS ANNEALED AT 650° C. PREVIOUS TO FINAL REDUCTION
BRINELL 50.3.   × 75.

average of twelve impressions. In the case of the bars annealed previous to a reduction of 58.8 per cent. at 570° and 650° C., and for which there appears to be little difference in hardness, the hardness numbers represent the average of five bars or strips.
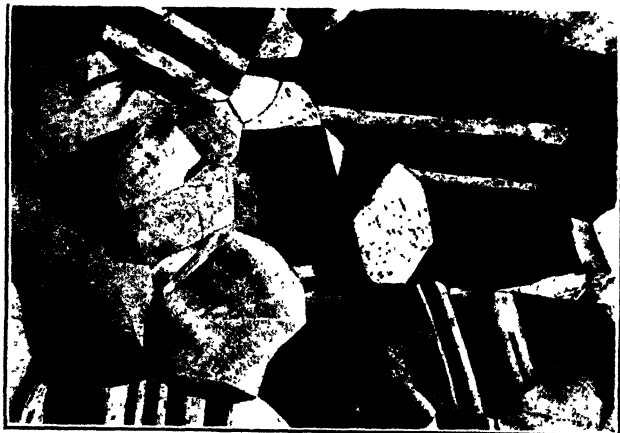
FIG. 3.—CARTRIDGE BRASS ANNEALED AT 750° C. PREVIOUS TO FINAL REDUCTION.
BRINELL 45.7.   × 75.

In Figs. 4 to 8, Brinell hardness numbers are plotted against the annealing temperatures. In Fig. 4, the effect of the deformation or degree of reduction is shown, the temperature of the anneal previous to the reduction remaining constant. In Figs. 5 to 8, the effect of the anneal,

TABLE I.—*Brinell Hardness*

Anneal After Rolling, Degrees C.	Anneal Before Rolling, Degrees C.	Bar 1 Reduction 58.8 Per Cent.	Bar 2 Reduction 50.7 Per Cent.	Bar 3 Reduction 37.1 Per Cent.	Bar 4 Reduction 20.2 Per Cent.
None	570	183.0	172.7	150.9	122.3
	650	182.0	166.6	146.4	108.9
	750	171.2	163.6	140.5	103.1
200	570	191.3	179.0	161.1	121.2
	650	189.0	174.5	154.9	109.2
	750	182.0	172.0	142.3	101.7
250	570	184.0	175.0	152.9	117.0
	650	181.0	171.0	147.9	111.5
	750	176.0	169.0	136.6	103.0
275	570	168.0	151.5	143.7	115.5
	650	166.0	147.2	141.5	110.0
	750	161.5	144 7	129.0	102.5
300	570	97.7	108.2	116.5	108.2
	650	93.6	107.8	119.4	104.5
	750	98.0	110.7	119.3	98.4
325	570	92.5	91.0	99 5	105.4
	650	92.5	89.7	104.0	99.6
	750	93.2	90.5	109.2	92.2
350	570	92.4	84.8	81 3	97.3
	650	89.4	81.4	80.5	95.8
	750	86.1	83 1	83.0	90.2
400	570	80.1	81.9	73 8	66.5
	650	79.7	80.9	73.2	66.5
	750	77.8	80.2	67.0	69.6
450	570	68.6	68.9	70 3	65.8
	650	68 8	68.2	68.9	65.2
	750	69.1	68.5	65.3	65.9
500	570	66.1	66.3	60.8	61.7
	650	69 3	68 4	60.8	61.1
	750	67.6	67.7	62.6	59.0
550	570	61.7	58.9	60.6	61.0
	650	62 3	57.8	61.2	61.1
	750	63.4	57.8	61.7	58.7
600	570	53.9	50.2	50.3	51.6
	650	53.8	50.9	49.6	51.6
	750	52.8	51.9	51.3	51.6
650	570	49.2	48.1	48.7	49.1
	650	49.5	47.8	48.3	49.2
	750	49.0	48.1	48.5	49.6
700	570	46.7	47.4	44.6	45.2
	650	46.4	47.7	44.7	43.9
	750	47.3	48.0	44.7	43.7
750	570	43.4	42.5	43.6	44.5
	650	43.2	43.1	43.3	43.9
	750	43.6	43.3	43.9	43.8

previous to the final rolling or reduction, is shown, the degree of reduction remaining constant.   For low-temperature annealing, the hardness of
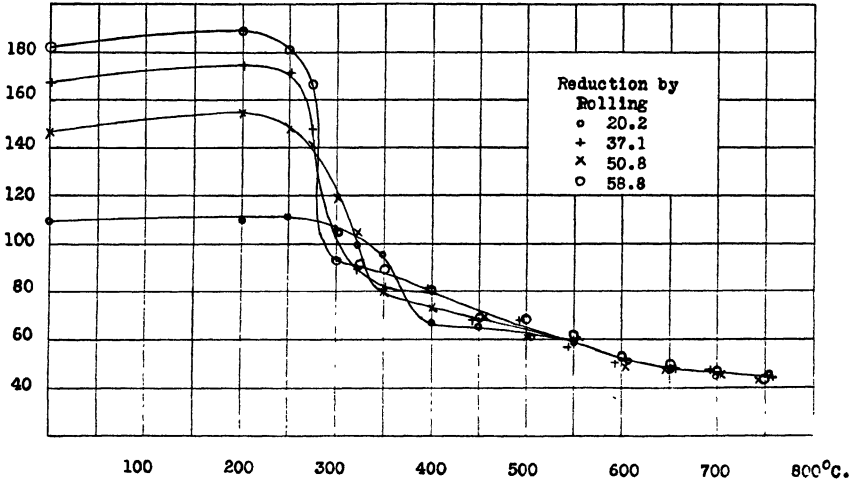
FIG. 4.—BRINELL HARDNESS OF CARTRIDGE BRASS ON ANNEALING AFTER REDUCTIONS BY ROLLING OF 20.2, 37.1, 50.8, AND 58.8 PER CENT.   ANNEALED PREVIOUS TO FINAL REDUCTION AT 650° C.

the brass is shown to be influenced by both the degree of reduction and the temperature of the anneal previous to that reduction.   The difference in hardness due to the degree of reduction disappears on annealing between
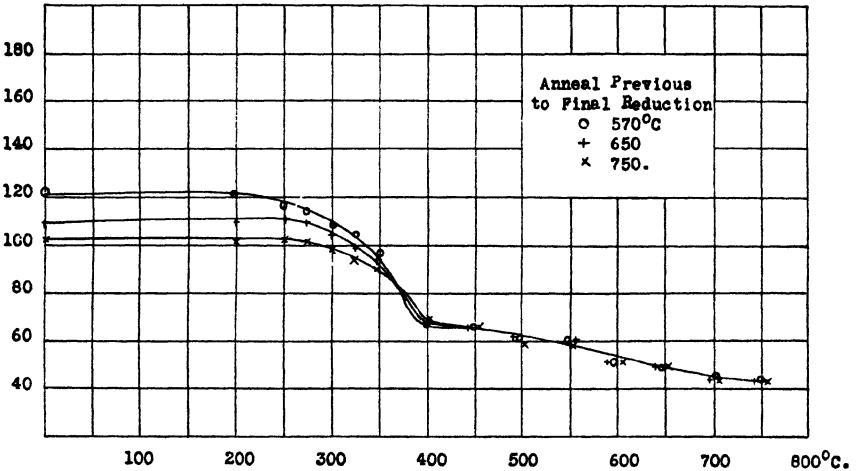
FIG. 5.—BRINELL HARDNESS OF CARTRIDGE BRASS ON ANNEALING AFTER A REDUCTION BY ROLLING OF 20.2 PER CENT.   ANNEALED PREVIOUS TO FINAL REDUCTION AT 570°, 650°, AND 750° C.

600° and 650° C.   For reductions of 20.2 and 37.1 per cent., the differences in hardness due to the anneal previous to the reduction disappears

at 500° C.; while for reduction of 50.8 and 58.8 per cent., the differences disappear at 300° C.
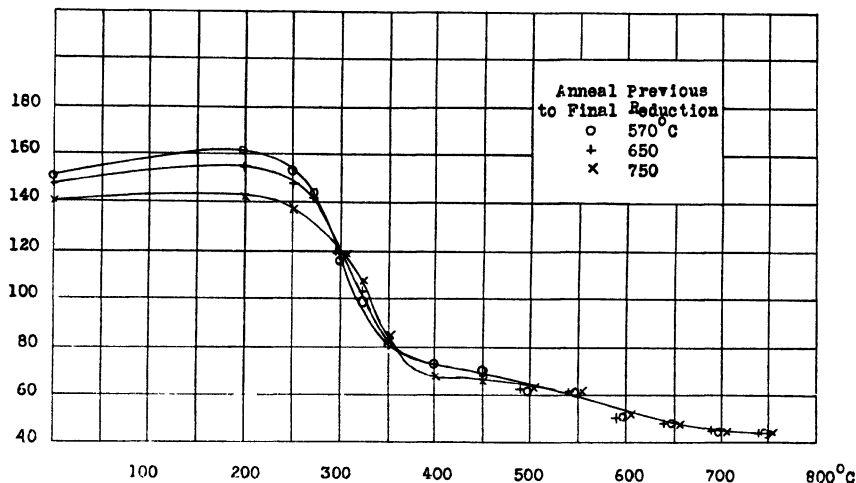
FIG. 6.—BRINELL HARDNESS OF CARTRIDGE BRASS ON ANNEALING AFTER A REDUCTION BY ROLLING OF 37.1 PER CENT. ANNEALED PREVIOUS TO FINAL REDUCTION AT 570°, 650°, AND 750° C.

It will be noted that the influence of the anneal previous to the rolling is more marked for the lower degrees of reduction. For the reduction of 58.8 per cent., there is little or no difference between the hardness ob-
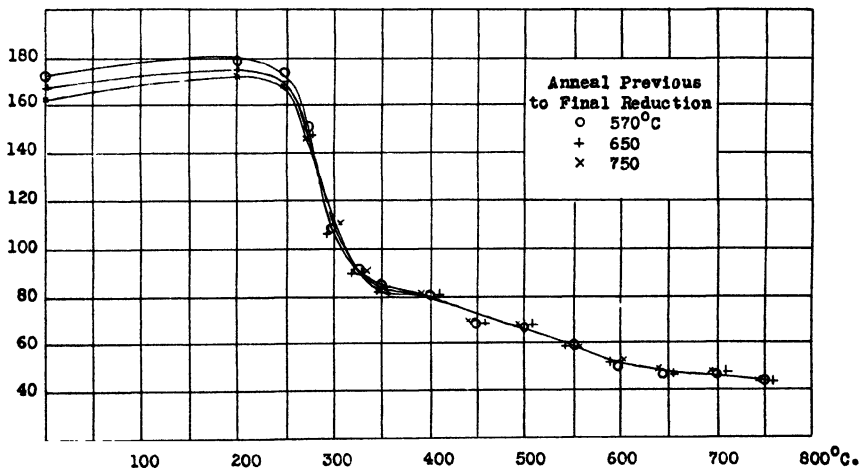
FIG. 7.—BRINELL HARDNESS OF CARTRIDGE BRASS ON ANNEALING AFTER A REDUCTION BY ROLLING OF 50.8 PER CENT. ANNEALED PREVIOUS TO FINAL REDUCTION AT 570°, 650°, AND 750° C.

tained on the metal previously annealed at 570° and 650° C., although an appreciable difference is shown for the metal previously annealed at 650°

and 750° C.   For the heavier reduction or very hard rolling, the influence of the anneal previous to rolling will disappear for temperatures of 650° C. or below.   The variation in grain size for annealing temperatures of 500° to 650° C. is shown by Bassett and Davis to be 0.040 to 0.080 mm.; and for annealing temperatures of 650° to 800° C., 0.080 to 0.225 mm.   With very heavy reductions, the Brinell hardness for low-temperature anneals will be independent of the anneal previous to rolling unless that anneal results in comparatively large variations in grain size.

Differences in Brinell hardness due to the temperature of the anneal previous to the deformation are shown to be of a lesser degree than those obtained for different degrees of reduction.   As only a limited range of
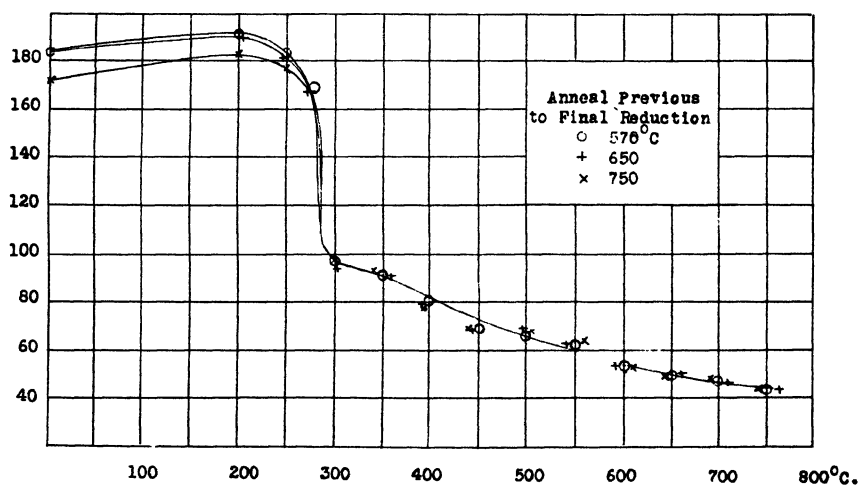
FIG. 8.—BRINELL HARDNESS OF CARTRIDGE BRASS ON ANNEALING AFTER A REDUCTION BY ROLLING OF 58.8 PER CENT.   ANNEALED PREVIOUS TO FINAL REDUCTION AT 570°, 650°, AND 750° C.

temperature was used for the anneals previous to rolling, extending of the temperature range from 570°–750° to 500°–800° C., particularly for the lower reductions, would result in differences more comparable to those obtained by varying the degree of reduction.

The influence of the previous anneal on tensile strength and elongation was found to be similar to that on the Brinell hardness, though to a smaller degree.   Tensile-strength tests were made on a 50,000-lb. Riehle Bros. machine using test pieces 0.5 by 0.134 in.   In Table 2 are given results of tensile-strength tests for low-temperature anneals; all tests were in duplicate.

When annealed at low temperatures, after reduction by rolling, the tensile strength and elongation are influenced by the anneal previous to the reduction.   This influence is more marked with the lower reductions and is practically eliminated with a reduction of 58.8 per cent.

TABLE 2.—*Tensile Strength and Elongation of Cartridge Brass Annealed Previous to Rolling*

Anneal, Degrees C.	Annealed at 570° C.		Annealed at 750° C.	
	Tensile Strength, Lb. per Sq. In.	Per Cent. Elongation in 2 In.	Tensile Strength, Lb. per Sq. In.	Per Cent. Elongation in 2 In.
Reduction, 58.8 per cent.				
None	79,800	5.2	78,900	5.3
200	81,375	5.2	78,925	7.0
250	76,000	8.5	74,700	12.0
275	71,600	12.7	72,700	14.2
300	49,875	47.5	48,825	47.3
325	50,100	48.3	50,200	48.0
350	46,550	50.0	46,325	52.0
Reduction, 50.8 per cent.				
None	74,475	7 0	71,800	7.7
200	73,650	7 5	71,725	10.2
250	74,750	7.7	72,300	8.5
275	65,550	19 7	66,800	18.0
300	50,660	41 5	54,500	41.5
325	48,800	47 2	49,925	46.5
350	45,475	49 7	44,850	53.0
Reduction, 36.6 per cent.				
None	63,150	11.2	60,000	19.5
200	63,400	15 5	58,650	19.0
250	60,750	19.2	58,550	25.0
275	62,400	20.7	59,525	26.5
300	53,100	35 5	54,400	33.7
325	51,325	42 0	53,875	37.2
350	44,250	52 5	46,600	51.2
400	44,150	57.7	43,200	61.7
Reduction, 20.2 per cent.				
None	51,800	32.5	48,700	39.2
200	52,300	30.3	47,500	44.0
250	51,275	33.5	48,050	....
275	50,250	36.7	48,700	43.0
300	51,200	37.0	47,750	39.5
325	51,125	40.7	46,850	46.2
350	48,200	43.5	47,750	45.5
400	44,650	52.5	42,300	53.5

Results of tensile-strength tests on metal annealed, previous to rolling, at 650° show little difference from those obtained on metal annealed at 570°, indicating that variation in the temperature of the anneal previous to rolling will have an appreciable influence on the tensile strength and elongation only when that variation results in marked change in grain size.

CONCLUSIONS

For low-temperature anneals, following reduction by rolling, the Brinell hardness of cartridge brass is appreciably influenced by the anneal previous to the rolling.

This influence decreases as the degree of reduction in rolling is increased.

For low-temperature anneals following reduction by rolling, the tensile strength and elongation are influenced, though to a lesser degree than the Brinell hardness, by the anneal previous to the rolling.

For cartridge brass annealed at temperatures of 650° C., or above, after reduction by rolling, the Brinell hardness, tensile strength, and elongation are independent of the anneal previous to the rolling.

It is desired to acknowledge with thanks the assistance of F. G. Smith, of the American Brass Co., in the preparation of the cartridge brass.

DISCUSSION

C. H. MATHEWSON,* New Haven, Conn.—The difference in temper between fine-grained low-annealed metal and coarse-grained high-annealed metal is perpetuated more or less after cold working. Therefore, it would seem necessary, if this difference in temper is to be equalized, that the cold-worked material should be thoroughly annealed. An anneal that would not completely relieve the strained hardened condition, could hardly equalize the temper difference; that fact has been brought out very clearly in this investigation.

* Professor of Metallurgy, Sheffield Scientific School, Yale University.

Arsenical Bearing Metals

BY HAROLD J. ROAST* AND CHARLES F. PASCOE,† MONTREAL, QUE.

THE object of this investigation was to compare the arsenical antimony-lead alloy with some of the regular bearing-metal alloys. With this end in view, the following tests were made:

1. Chemical analyses to show the composition.
2. Thermal analyses.
3. Macroscopic examination of fractures.
4. Microscopic examination.
5. Tensile tests.
6. Crushing tests.
7. Scleroscope tests.
8. Brinell tests, at temperatures from 80° to 400° F.

Tests 3 to 8 were conducted on both sand-cast and chill-cast specimens. For the purposes of this paper the alloys have been numbered from 1 to 6; their composition is as follows:

NUMBER	LEAD, PER CENT.	ANTIMONY, PER CENT.	TIN, PER CENT.	COPPER, PER CENT.	ARSENIC, PER CENT.
1	82.64	12.95	4.40	0.01	none
2	27.54	9.94	59.87	2.65	none
3	77.65	20.75	none	0.13	1.47
4	82.95	16.85	none	0.20	none
5	83.78	15.35	none	0.05	0.82
6	0.15	7.87	84.13	7.85	none

The analysis, in the case of the antimony, was made by solution in sulfuric acid and titrated with permanganate. For tin, the ferric-chloride method was used; for copper, the alkaline separations, and the thiocyanate methods; for arsenic, volatilization of arsenic and titration with iodine. The lead, with the exception of alloy 6, was taken by difference.

With the exception of No. 5, the alloys were taken as received from the manufacturer. About 40 lb. were melted in a graphite crucible and recast into small ingots to insure uniformity of composition; samples

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* Manager, Testing Dept., James Robertson Co., Ltd.
† Consulting Metallurgist.

for analysis were taken from these remelted ingots. No. 5 was made up by the authors and remelted as in the other cases. A gas-fired crucible furnace holding a 40-lb. graphite crucible was used; it was easily regulated as to temperature and condition of flame, whether oxidizing or reducing. All melts were kept at or below 1000° F. and no delay occurred between the melting of the alloy and the casting. The size of crucible was varied according to the amount of metal required for the particular test.

THERMAL ANALYSES

For the thermal analyses, about 650 gm. of alloy was melted in a small graphite crucible, which was practically filled with the metal and brought to the required temperature; it was then removed to a suitable stand and a 900° F. thermometer inserted in a covering tube of electrically welded sheet iron that fitted the thermometer closely, thus permitting the easy removal of the thermometer from the solid mass at the end of the experiment. Owing to the thinness of the covering tube, practically no lag
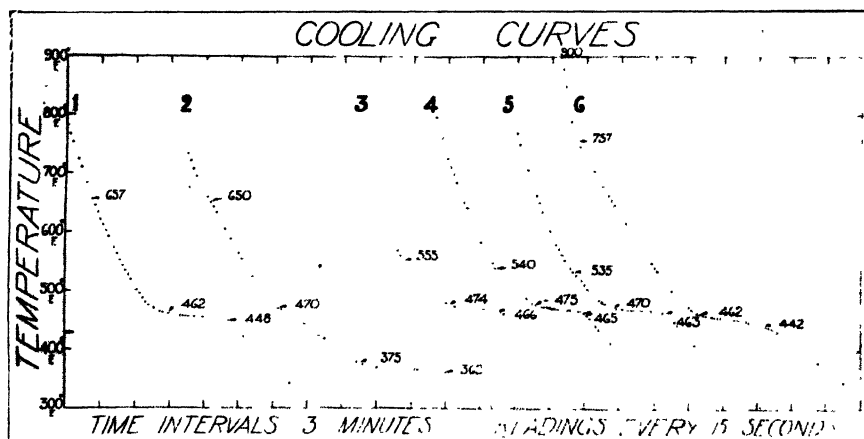
FIG. 1.

occurred. The thermometer was capable of being read to within 1° F. The crucible was covered with an asbestos cover, through which the protection tube was inserted. Readings were taken at intervals of every 15 sec., and the thermal curves shown in Fig. 1 were obtained.

The curves show that the addition of arsenic has little effect on the curve. With exception of alloy No. 2, the eutectic point varies but slightly; the decided lowering of the eutectic point in No. 2 must be borne in mind when considering the value of this alloy as a bearing metal. It is perhaps remarkable that the eutectic point of alloys of such variable constitution as Nos. 1, 3, 4, 5, and 6 should be at so nearly the same temperature; or in other words, that the softening point of all these alloys,

or the danger point so far as the heating of the bearing is concerned, should be so little affected by such great changes in composition. Further, a comparison of alloys 3, 5 and 4 shows that the addition of arsenic has practically no effect on the first precipitation of crystals, or the eutectic point, notwithstanding the decided change in structure, which will be commented upon in connection with the micrographs. Curve 3 shows that the precipitation started at a higher temperature than 4 and 5, which is in conformity with the increase of antimony in this alloy. Inasmuch as this investigation was primarily intended to deal with arsenic as a constituent of bearing metals and the constitution of the other babbits has been the subject of much discussion, we have not considered the constitution of alloys 1, 2, and 6 to any extent.

FRACTURE EXAMINATION

For the purpose of fracture examination about 23 lb. of metal, including the bar, gate, and riser, were cast, a pattern similar to that shown in Fig. 2 being used. We have found this pattern most useful for casting test bars, not only for white metal but for the brasses and bronzes,
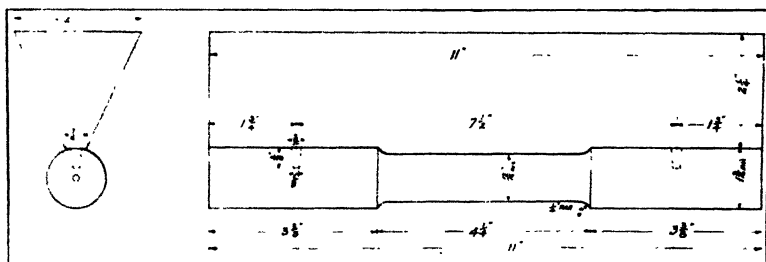
FIG. 2.—TEST-BAR PATTERNS.

including manganese bronze. Examination of the chill fractures was made on the chill-cast test bars and Brinell pieces, as described later. These were cast, at 800° F., into molds heated to 212° F., with the exception of No. 6, which was cast at 900° F. The fractures of the alloys varied considerably, especially in the case of the slowly cooled sand-cast

FIG. 3.—SAND-CAST BARS; ACTUAL SIZE.

specimens, as shown by Fig. 3. It will be noticed that a small quantity of arsenic has an extraordinary refining effect. The coarse crystalline fracture of the arsenic-free alloy No. 4 is reduced to practically a silky fracture with the entire elimination of the large plate-like crystals. This

difference of fracture or grain size is the most noticeable effect of the addition of arsenic and the grain produced is much finer than that of any of the other alloys. The appearance of the samples is far more convincing than the photographs; repeated melting and casting has no appreciable effect on the fineness of the fracture.

MICROSCOPIC EXAMINATION

Pieces were taken from the center of the sound part of the sand-cast bars, for the sand-cast specimens, and small cylinders, as used for the crushing tests, were cast for the chill specimens. The casting temperature for alloys 1 to 5 was 800° F. and 900° F. for No. 6; as before, the chill molds were heated to 100° F. All the photomicrographs were taken at a

FIG. 4.—CRYSTAL STUDY SAND-CAST BARS. × 70.

magnification of 70 diameters and are therefore strictly comparable. The samples were etched with dilute hydrochloric acid, with the exception of the crystal study, Fig. 4, which was etched with ferric chloride to eliminate the ground mass and so accentuate the crystal. An examination of samples from the blocks used for Brinell tests, as described later, showed practically the same structure as the chilled cylinders from which the photomicrographs shown in Figs. 5 and 6 were made. For the same reason as is given for the curves, the following general statement concerning alloys 1, 2, and 6 will suffice.

In Fig. 5, No. 1 shows the usual cubical crystals of tin-antimony SnSb, or gamma, compound, embedded in the softer lead-antimony eutectic, which has a distinctly crystalline tendency. It is interesting to compare this lead-antimony eutectic with the eutectic in No. 4, which has presumably a similar composition. No. 2 has the tin-copper crystals as a primary constituent together with the cubic crystals of the tin-antimony compound embedded in a complex matrix, which apparently

No. 1.

No. 2.
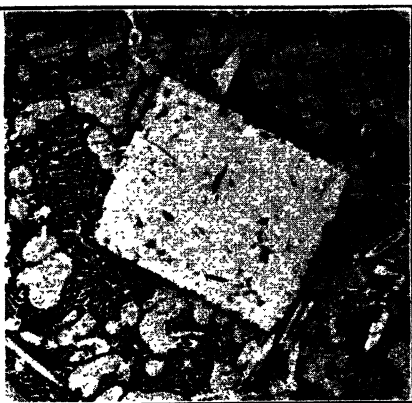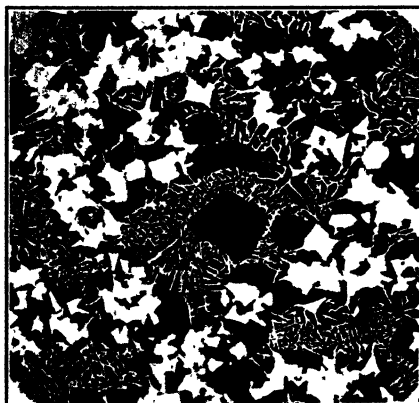
No. 3.
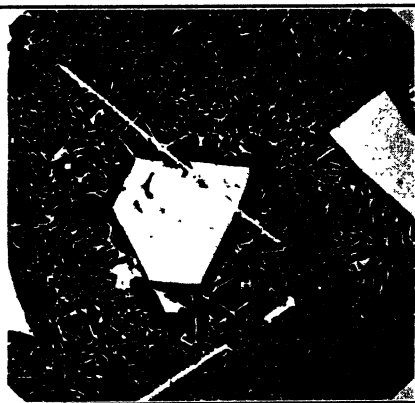
No. 4.
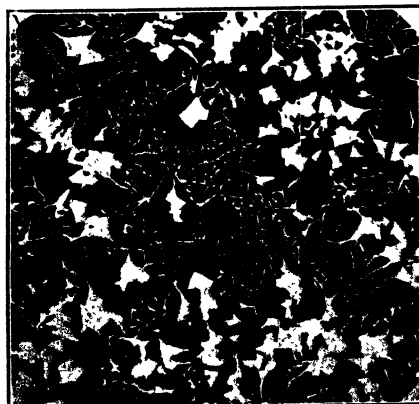
No. 5.

No. 6.

FIG. 5.—SAND-CAST BARS. × 70.

No. 1.

No 2.
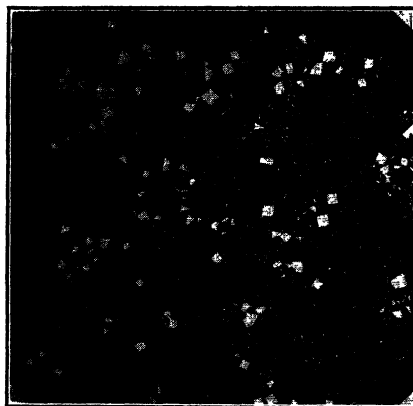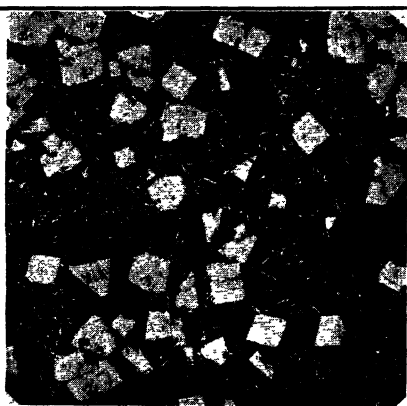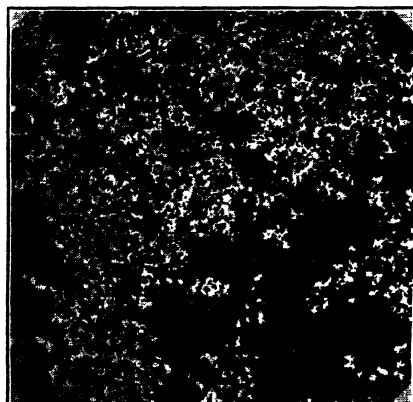
No. 3.

No; 4.
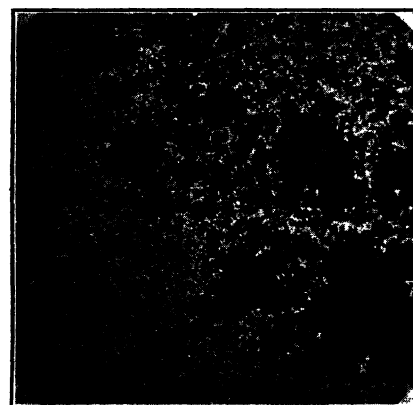
No. 5.
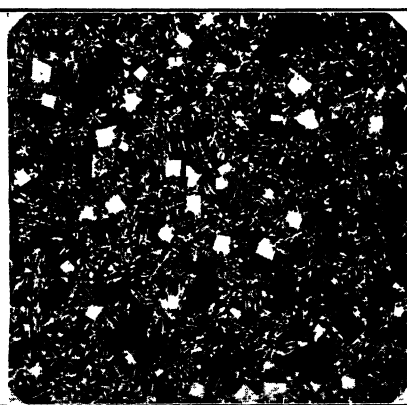
No. 6.

FIG. 6.—CHILL-CAST BARS. $\times$ 70.

consists of crystallites, which are embedded in an eutectic.   No. 6 gives greater evidence of the tin-copper crystals followed by the cubic tin-antimony compound embedded in a matrix exceedingly rich in tin. Under high magnification, this matrix does not show any sign of the usual eutectic structure, although it has been suggested that it may be an eutectic consisting of tin with 1 per cent. of copper.   The most striking difference in alloys 3, 4, and 5 is the marked effect of the arsenic on the antimony crystals and the tendency to transform the cubic crystals of antimony into needle-like plates having a curved tendency.   The antimony in the eutectic of antimony and lead has a tendency to segregate away from the primary crystals, leaving dark structureless masses consisting probably of lead.   This is also true of alloy 4 but to a less degree.

FIG. 7.—CRYSTAL STUDY OF SAND-CAST BARS.   × 850.

The copper content of 0.20 per cent. in alloy 4 shows clearly, as the Cu_2Sb compound having its characteristic violet color, in the acicular crystals From the amount present it would seem that practically all the copper is thus accounted for.   The still smaller amount of copper present in the arsenical alloys exists apparently as the same Cu_2Sb compound in the center of some of the antimonial crystals, indicating that the copper compound was the first constituent to solidify and formed, in some cases, nuclei for the antimonial crystals.   This copper compound is not readily visible under the low power but is obvious at a higher magnification, as shown by Fig. 7.   From our observations, the action of the arsenic is concentrated on the antimony crystals, compelling them to crystallize out in smaller units, and therefore larger numbers, and also distorting them, as before mentioned.

The refinement of the fracture and therefore of the grain size may reasonably be expected to have an influence on the physical constants, as shown by the tensile and hardness tests..   With this in view the following tests were made:

TENSILE-STRENGTH TESTS

The alloys were heated to 800° F., with the exception of No. 6, which was cast at 900° F., and poured into a sand mold similar to Fig. 2. The cooling was slow and every opportunity given for the final re-solution of the various eutectics. The bars were turned to 0.798 in. diameter and 4 in. between centers. The temperature of the testing laboratory was about 80° F. In the case of the chill-cast samples, the alloys were cast at the same temperature as those in the sand but into iron molds heated to about 212° F., the samples being cast vertically and turned down to the dimensions just given. The results of the tensile-strength tests are as follows:

	SAND CAST, LB. PER SQ. IN.	CHILL CAST, LB. PER SQ. IN.		SAND CAST, LB. PER SQ. IN.	CHILL CAST, LB. PER SQ. IN.
1	8,620	8,950	4	5,520	6,130
2	8,820	11,000	5	7,030	7,980
3	6,160	7,280	6	10,900	13,980

The elongation and reduction of area were not recorded, being very slight.

The results given suggest that the arsenical alloys Nos. 3 and 5 are less affected by casting conditions than the high-tin alloys Nos. 2 and 6, which are very susceptible to temperature conditions.

CRUSHING TESTS

Crushing tests were made on both sand-cast and chill-cast samples poured under the same conditions as the test bars. The height of the cylinders after machining was approximately 2.5 in. and the diameter 1.25 in. Loads of 1000, 5000, 10,000, and 15,000 lb. were applied with the results given in Table 1.

TABLE 1.—*Results of Crushing Tests*

Number of Alloy	Original Height, Inches	Reduction in Height, Inch, after a Pressure of				Original Diameter, Inches	Increase in Diameter, Inch, after a Pressure of			
		1000 Lb.	5000 Lb.	10,000 Lb.	15,000 Lb.		1000 Lb.	5000 Lb.	10,000 Lb.	15,000 Lb.
				Sand-cast Bars						
1	2.508	*nil*	0.005	0.038	0.158	1.2480	*nil*	0.0016	0 0112	0 0588
2	2.525	*nil*	0.006	0.015	0 070	1.2511	*nil*	0.0004	0 0021	0.0169
3	2.530	*nil*	0.007	0.049	0.206	1.2490	*nil*	0.0010	0 0070	0 0960
4	2.512	0.004	0.012	0.112	0.392	1.2519	*nil*	0.0014	0.0232	0 1131
5	2.520	0.004	0.010	0.045	0.206	1.2498	0.0012	0.0018	0.0112	0 0742
6	2.500	*nil*	0.005	0.009	0.054	1.2453	*nil*	0.0005	0.0017	0 0140
				Chill-cast Bars						
	2.528	*nil*	0.006	0.038	0.138	1.2498	*nil*	0.0011	0.0108	0.0404
	2.540	0.002	0.006	0.015	0.120	1.2409	*nil*	0.0006	0.0022	0 0295
	2.535	*nil*	0.007	0.029	0.124	1.2528	*nil*	0.0011	0.0073	0.0354
	2.535	*nil*	0.007	0.075	0.353	1.2500	*nil*	0.0010	0.0200	0.1136
	2.528	*nil*	0.006	0.039	0.151	1.2500	*nil*	0.0012	0.0098	0.0443
	2.514	*nil*	*nil*	0.005	0.021	1.2512	*nil*	0.0000	0.0008	0.0052

After this test, increasing loads were put on the sand-cast cylinders until an apparent maximum was reached; this apparent maximum was:

Alloy No.	Lb. per Sq. In.	Alloy No.	Lb. per Sq. In.
1	21,980	4	19,330
2	16,580	5	20,880
3	19,770	6	24,430

Tests were made with the Shore scleroscope on the chill-cast samples and the same test pieces were used for Brinell tests, with the following results:

Alloy No.	Scleroscope Results	Reduced to Normal Readings	Alloy No.	Scleroscope Results	Reduced to Normal Readings
1	13, 12, 13, 13		4	9, 9, 9	5
2	14, 12, 14, 12		5	9, 9, 9, 9	5
3	9, 9, 10, 9		6	19, 19, 20, 19	11

Our experience, based on these figures and others not recorded, indicated that for this class of alloy the scleroscope did not provide a means of differentiating their hardness. For instance, alloys 3, 4, and 5 show no difference by scleroscope while showing distinct differences under Brinell, tensile, and crushing tests.

TABLE 2.—*Results of Brinell Tests*

Alloy No.	80° F. 500 Kg.	100° F. 500 Kg.	212° F. 250 Kg.	300° F. 125 Kg.	400° F. 125 Kg.	80° F. 500 Kg. *	Sand Cast, 80° F. 500 Kg.
1	21.0	16 5	10.5	8.6	4.6	18.6	18.6
2	19.3	23.8	11.9	8.5		25.9	14.3
3	21.0	15.9	13.6	10.7	6.5	17.8	20.1
4	17.2	12.8	7.6	7.1	3.8	14.3	15.9
5	19.3	15.3	13.0	9.0	5.2	17 8	
6	31 2	29.7	15 6	11.2	7.8	32.6	28.4

* After heat treatment.

Brinell tests were made on a 10,000-lb. Olsen machine using a special casting to hold the 10-mm. ball and weighing the applied load. It was thought that by this method a more accurate result could be obtained than with the Brinell machine generally used. The metals were heated in a bath of linseed oil by means of an electric heating unit and stirred constantly. A thermometer was used for temperature and the load was applied for 30 sec. The test blocks were cast, at a temperature of 800° F. for alloys 1 to 5 and at 900° F. for alloy 6, in rectangular iron molds 2½ by 4 by 1 in. These blocks were turned in a lathe, the under, or chill side, being used for the Brinell test. The blocks were found to be very uniform. It will be noticed that the arsenical alloys 3 and 5 are harder than the non-arsenical alloy 4; and that as the temperature rises 3 and 5 get harder than 1 and 2. A summary of the tensile, crushing and hardness tests, all on chill-cast samples, is given in Table 3.

TABLE 3.—*Summary of Test Results on Chill-cast Bars*

Alloy No.	Tensile Strength at 80° F., Lb. per Sq. In.	Reduction in Height Crushing at 10,000 Lb., at 80° F., Inch	Hardness	
			at 80° F.	at 212° F.
1	8,950	0.039	21.0	10.5
2	11,000	0.015	19.3	11.9
3	7,280	0.049	21.0	13.6
4	6,130	0.112	17.2	7.6
5	7,980	0.045	19.3	13.0
6	13,980	0.009	31.2	15.6

SUMMARY

Of the six alloys taken for comparison, the high-tin base is undoubtedly the hardest, the strongest, and the most resistant to crushing loads. On the other hand, all the alloys stand 1000 lb. per sq. in. without deforming, and all deform but slightly at 5000 lb.; in other words, none would be expected to squeeze out of a bearing at ordinary loads. Leaving out No. 6, the arsenical alloys maintain their hardness as the temperature rises better than the other alloys. Structurally, the arsenical alloys have by far the finest grain while having hard crystals embedded in a softer but tough matrix. The arsenical alloys can be poured at a lower temperature than No. 6 and run more readily at, say, 700° F. than any of the other alloys. Further, they show less sign of oxidation. In the matter of cost of components, the arsenical alloys (not taking into account the special cost of alloying in the case of the latter) will be less than the high tin alloys but more than alloys 1 and 4. All of these tests have been carried out from the practical rather than the academic point of view and several lines of experimental work are suggested.

The authors hope to make further investigations when opportunity offers. These results have been made public in the hope that they may save, in some degree at least, duplication of effort, and form a basis for present discussion and further work.

The thanks of the authors are due to The James Robertson Co., Ltd., Montreal, for facilities furnished and for permission to publish the figures obtained.

DISCUSSION

JESSE L. JONES, Pittsburgh, Pa. (written discussion).—One of the chief objections to the arsenical lead babbitts has been their lack of toughness. When such a babbitt is used for lining a flanged bearing, the babbitt on the flange is liable to crack off, even if well anchored. The rest of the lining is also liable to crack if any pounding occurs. Where peening is

necessary to expand a lining into the anchor holes, the arsenical lead babbitts probably would be unsatisfactory.

The maximum arsenic allowed by the A. S. T. M. Tentative Specification B23–18T for an alloy of 15 per cent. antimony and 85 per cent. lead is 0.25 per cent. The Brinell hardness given for this alloy is 17.0 at 70° F.and 9.9 at 212° F. Compared with the figures given for the No. 5 alloy, it would seem that arsenic has a decided hardening effect.

It is a common belief among users of babbitt that the presence of arsenic in either the babbitt or the bronze shell prevents satisfactory adherence of the lining. Recently, some castings were made from an alloy of 80 per cent. copper, 10 per cent. lead, and 10 per cent. tin, with the addition of 0.5 per cent. of arsenic, instead of 0.5 per cent. of phosphorus. The castings were tinned with "half and half" solder and babbitted with an alloy of 78 per cent. lead, 8 per cent. tin, and 14 per cent. antimony; the adhesion of the babbitt to the bronze was excellent.

The babbitt known as No. 4 Hardware Babbitt consists of 77 per cent. lead, 20 per cent. antimony, 1 per cent. copper, 1 per cent. arsenic, 1 per cent. tin. U. S. Patent 874866, Dec. 24, 1907, granted to Chas. J. Reed, of Philadelphia, Pa., specifies an alloy 85 lb. lead, 10 lb. arsenic and 1 to 5 lb. copper. For certain purposes, the inventor says, the copper may be omitted.

WILLIAM A. COWAN, New York, N. Y. (written discussion).—The object of the paper is stated to be a comparison of arsenical alloys with regular bearing-metal alloys, but the summary gives the impression that the authors attempted to prove that the arsenical alloys are the better; however, this has not been accomplished. Data on a sufficient number of alloys have not been presented; in fact, it is possible to make a proper comparison between similar alloys with and without arsenic only in the case of alloys 4 and 5, and these may be affected, also, by small differences in composition, particularly in respect to copper.

Arsenic imparts a finer grain to antimonial-lead alloys, but it also tends to make them more brittle and friable and less malleable and plastic. The statement that, as the temperature rises, the arsenical alloys maintain their hardness better than other bearing metals is not fully borne out. Sufficient data to uphold it are not given and in the tests that can be compared, as given in Table 2, the only figure which would seem to prove this is Brinell hardness of 7.6 for alloy No. 4 at a temperature of 212° F. This figure should be rejected as not consistent with determinations of hardness of this alloy at other temperatures. Between the Brinell hardness of 12.8 at 100° F. and 7.6 at 212°, there is a drop of 5.2 points; whereas, between this figure at 212° and 7.1 at 300° there is a drop of only 0.5 point. This is entirely unreasonable and inconsistent with the other tests on this and the other alloys. In fact, it has been shown that, in general, the relation between decrease in hard-

ness and increase in temperature can be represented approximately by a straight line between room temperature and the solidification point of the alloy in question. It will be seen, by comparing the Brinell hardness tests, in Table 2, for alloys 4 and 5 that the decrease in hardness of the two alloys is practically parallel when this figure of 7.6 is rejected.[1]

As hardness decreases the more rapidly with a reduction of the solidification point of the alloy under consideration, it is purposeless to point out that alloys 3 and 5, representing arsenical alloys, do not decrease in hardness as rapidly with a rise in temperature as does alloy 2. The presence of arsenic, of course, has nothing to do with this.

In preparing the specimens for photomicrographing, it is probable that equilibrium of the different structural components could better have been attained by a slower cooling than even sand casting or annealing will give. For instance, an alloy of the composition of specimen No. 1 should, if in complete thermal equilibrium, show no separation of crystals of antimony-tin compound; it would be composed of eutectoid with only a slight excess of crystals of antimony. Specimen No. 2, if in complete equilibrium, would probably show some attack of the cubic crystal of the antimony-tin compound by the surrounding liquid, giving an eutectic formation at the reaction point P, as described in the thermal equilibrium study of lead-tin-antimony alloys by Dr. William Campbell.[2]

In the determination of cooling curves, probably a more accurate location of critical points could be obtained by using a thermocouple and by a differential or inverse-rate method.

The statement in the summary that the arsenical alloys are composed structurally of hard crystals embedded in a softer but tough matrix is misleading. The matrix of the lead-antimony alloys, with and without arsenic, is essentially the same; it is composed of the lead-antimony eutectic, which is less tough than is the ground mass of bearing metals containing tin. In the case of the lead-tin-antimony alloys, low in tin, this ground mass is lead-tin-antimony eutectoid. In the case of tin-antimony-copper alloys, the ground mass is antimony-tin solid solution; in both cases the matrix is tougher than the antimony-lead eutectic alone. None of the alloys under consideration differ in the fact that the matrix is softer than the crystals embedded therein.

One of the advantages in the use of lead-base bearing metals containing tin, over straight antimonial-lead alloys, is that the addition of tin adds toughness and malleability to the alloy. The addition of arsenic to antimonial lead affects these properties in exactly the opposite direction. This is objectionable in bearing metals, for alloys for such use should have

[1] In this connection, see paper on similar subject: Some Properties of White Metal Bearing Alloys at Elevated Temperatures, by John R. Freeman, Jr., and R. W. Woodward. Bureau of Standards *Tech. Paper* 188.

[2] A. S. T. M. *Proc.* (1913) **13**, 630–665.

sufficient malleability or plasticity to flow out under excessive pressure and conform to any irregularities in fit or alignment of the shaft. In this connection it would be interesting to note the condition of the test pieces as to cracking after application of the maximum load in the tests given at top of page 743.

The paper brings out the well-known differences in structure and properties between antimonial lead and other lead-base or tin-base bearing metals, but does not prove that antimonial lead containing arsenic is as good a bearing metal as one without arsenic; the latter element, in fact, imparts some properties that are detrimental.

The paper is valuable in reporting experimental work that has been done and it would be desirable for the authors to carry out the other investigations suggested.

WILLIAM K. FRANK,* Pittsburgh, Pa.—In the manufacture of rail-road-car journal bearings, we have found that the non-adherence of babbitt linings to the bronze backs is usually due to arsenic content in the babbitt. Commercial antimonial lead, in most cases, contains arsenic, and in the purchase of this material we specify that this should be below 0.25 per cent., which, in our opinion, is the upper limit for arsenic where good adherence is necessary. Some time ago we rejected a lot of anti-monial lead containing about 86 per cent. lead, 13 per cent. antimony, and 1 per cent. arsenic. The seller would not agreed that arsenic caused loose linings, so we lined twenty bearings with the antimonial lead, as received; these showed eighteen loose linings. With a mixture of half antimonial lead and half new lead and antimony, in the same proportions, nine loose linings resulted in twenty samples. With an all new lead and antimony mixture, only one loose lining showed in twenty samples. As arsenic was the only impurity present in appreciable amount, all present agreed that the arsenic was responsible for the trouble.

WILLIAM M. CORSE,† Bayonne, N. J.—Would the addition of arsenic to other metals produce the same effect, as to refining of grain, that it apparently does in this combination of alloys? In other words, is this an inherent property of small additions of arsenic in any of the combinations with fairly high copper?

WILLIAM K. FRANK.—In some experiments with arsenic in bearing bronze of approximately 79.75 per cent. copper, 10 per cent. tin, 10 per cent. lead, and 0.25 per cent. arsenic, we found no such marked refining of the grain size as noted in the babbitts.

HAROLD J. ROAST.—As to the refining effect of arsenic in other alloys, though I have had a limited experience with arsenic alloys, I should say

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† Monel Metal Products Corpn.

that where tin is present in white-metal alloys arsenic has the opposite effect. We get the well-known bird's-nest crystals, and a very coarse crystal is produced by a very small amount of arsenic when tin is present. It does not necessarily follow that arsenic will refine the grain of metals generally because it refines the grain in certain special white metals.

We have not attempted to prove that arsenical alloys are necessarily better than others; careful reading of the summary will show that the facts found by the practical tests have been stated without recommending arsenical alloys as being superior. This paper gave the result of tests made largely according to works practice under conditions that did not permit of exhaustive investigation.

The statement that arsenic tends to make antimonial-lead alloys more brittle and friable raises the question as to whether the alloys are made too brittle. Examination of the compression-test cylinders showed clearly that the arsenical alloys squeezed down without showing brittleness and, subsequent inquiries by the writers have elicited the fact that, in Canada, arsenical alloys, of the type referred to in the paper, have given very satisfactory results in rolling-mill practice. This would seem to discredit the idea of the alloy being of a brittle nature.

In regard to the rejection of the figure for Brinell hardness of 7.6 for alloy 4 at a temperature of 212° F. we would say that this figure, in common with all the Brinell figures, is the result of several tests on each piece giving close agreement. Although theoretically only eutectoid may be present with a slight excess of crystals of antimony, such conditions will not be found under a normal rate of cooling and, as already stated, the object of the authors was to carry out the investigation within reasonable limits of practical conditions. Furthermore we doubt if the use of a thermocouple would have given greater accuracy than the thermometer method, which readily gave readings to 1° F. It is of course obvious that the inverse-rate method would give a more startling representation of the thermal activity, but the time at the disposal of the investigators was not favorable to a duplication of methods when the practical results had been obtained.

We cannot see that "the statement in the summary that the arsenical alloys are composed structurally of hard crystals embedded in a softer matrix" is misleading. It is a plain statement of fact and is intended to be read as such; we do not wish to convey that this is a unique condition in the case of arsenical alloys. In regard to the cracking of the test pieces after application of the maximum load, we would say that the 2½-in. cylinders composed of the arsenical alloys showed less sign of cracking, and therefore of brittleness, than the other alloys. In testing, the cylinders were squeezed down by the maximum load to a height of about 1¼ in.

The statement that copper makes for a refinement of grain in anti-

.mony and lead alloys is undoubtedly correct, but it does not have nearly the same refining effect as the arsenic; in other words, the refining effect could not be caused by the slight differences in the copper content of alloys 4 and 5. The refinement of grain with arsenic holds good over a large range of temperature. If an alloy were cast at 750° and also at 1000°, it would be difficult to distinguish a difference in the grain, with the eye.

Without going into details of the figures, it would seem to me, notwithstanding certain inequalities of the figures in connection with Brinell hardness, that undoubtedly alloy 2, for example, is much more adversely affected by increase in temperature, considering that it goes from 19.3 at 80° to 8.5 at 300°, after which it was too soft for handling, whereas the arsenical alloy is reduced from 19 to 9 and is quite solid at that temperature, which of course is borne out by the curves.

In regard to the various alloys under discussion, the reason for not following an annealing process, or something tantamount to it, is suggested in the summary.

Some Electrical Properties of Nickel and Monel Wires

By M. A. HUNTER,* F. M. SEBAST,† AND A. JONES,† TROY, N. Y.

(Rochester Meeting, June, 1922)

THIS paper gives the results of an investigation of the specific resistance and temperature coefficient of the electrical resistance of nickel and its important alloy, monel metal. For most of the pure metals, the temperature coefficient of resistance is approximately 0.004 per degree centigrade at 20° C. Nickel and iron, however, are exceptions. This property is important as it gives us good material for the construction of thermometers and other instruments involving electrical control.

A summary of information on this subject has been published by the Bureau of Standards.[1] The present experiments antedate this publication by several years and as the information obtained is not contained in it, the results of the experiments are published as additional information.

ELECTRICAL PROPERTIES OF PURE NICKEL

It is generally conceded that the lowest values for electrical resistance are obtained in the purest metal. In the case of nickel, the best material obtainable is electrolytic nickel, which contains approximately 99.80 per cent. nickel, 0.15 per cent. iron, and 0.05 per cent. copper. Any cobalt that may be present is included in the nickel figure. If the electrolytic sheet has been deposited without intermission, it is possible to cut a section from the sheet and roll it into wire without melting. By this means a nickel wire may be obtained that is uncontaminated by any of the impurities always taken up by nickel during melting.

Two samples of nickel wire obtained from different electrolytic nickels had the following electrical resistance and temperature coefficient of electrical resistance:

SAMPLE	MATERIAL	SPECIFIC RESISTANCE	TEMP. COEFF. PER DEG. C.
No. 1	Electrolytic nickel....................	7.55	0.00559
No. 2	Electrolytic nickel....................	7.60	0.00553

* Rensselaer Polytechnic Institute. Director of Research, Driver-Harris Co.
† Russell Sage Laboratory, Rensselaer Polytechnic Institute.
[1] U. S. Bureau of Standards, *Circular* 100 (March, 1921).

The specific resistance is given in microhms-cms., and the temperature coefficient per degree at 20° C. over the range from 20° to 50° C. The first value given is equivalent to a value of 0.00629 per degree C. at 0° C.

It is not feasible, however, to make nickel wire on a commercial scale by this method. For production in quantity, the metal must be melted, cast into ingots, forged, and drawn into wire. The effect of melting, therefore, was investigated. A sample of electrolytic metal was melted under ideal conditions in an alundum crucible in the Arsem vacuum furnace. This material, when worked down to wire, had a specific resistance of 7.87 and a temperature coefficient of 0.00541. This shows that, even under ideal conditions, the melting process results in a deterioration of the electrical properties of the material.

EFFECT OF IMPURITIES

It is considered good practice, in the melting of nickel, to add manganese to the molten bath in order to increase the forgeability of the ingots, so that the three principal impurities in commercial nickel wire are cobalt, iron, and manganese. The effect of these was found by adding 1 per cent. of each to electrolytic nickel in successive experiments. The melts were made in alundum crucibles, in vacuum, and gave the following results:

MATERIAL ADDED	SPECIFIC RESISTANCE	TEMPERATURE COEFFICIENT
Cobalt...................................	8.38	0.00521
Iron......................................	8.82	0.00490
Manganese...............................	9.41	0.00459

The addition of each of these constituents lowers the electrical qualities of the material, manganese, however, having the greatest effect. When the metal is melted in air, the chance for the absorption of impurities is materially increased. An electrically heated furnace was constructed with a carbon resistor, and 5 lb. of nickel was melted in it under a glass slag in a clay-lined crucible. When the metal was melted, manganese or aluminum was added. The results obtained were as follows:

	MATERIAL ADDED	TEMPERATURE COEFFICIENT
No. 1	0.25 per cent. manganese..............	0.00460
No. 2	0.63 per cent. manganese..............	0.00452
No. 3	1.00 per cent. manganese..............	0.00357
No. 4	0.50 per cent aluminum..............	0.00368
No. 5	1.00 per cent. aluminum..............	0.00285

Melt No. 1 gave a porous casting, as too little manganese was added to deoxidize the melt completely. Melts Nos. 2 and 3 were sound. In

melts Nos. 4 and 5, the metal was treated with aluminum immediately before pouring. The cast bars had an excellent appearance but the metal was finely granular and not fibrous. Such additions of aluminum are of no value in melting nickel where a high temperature coefficient is required in the material. The use of aluminum (and magnesium) must be confined to the very small additions used for final deoxidation.

As the quality of the nickel is affected by this method of melting, it was reasonable to suppose that the time taken to melt the metal should influence the material to a considerable degree. Accordingly, a succession of melts were made. For the first melt, the furnace required 45 min. to reach the melting point of nickel. The last melt was cast 12 min. after the introduction of the cold charge. The nickel used was electrolytic sheet, to which 0.75 per cent. manganese was added in each case. The following results were obtained from the separate melts:

MELT	SPECIFIC RESISTANCE	TEMPERATURE COEFFICIENT
No. 1 (45 min.)	9.75	0.00448
No. 2	8.80	0.00462
No. 3	8.67	0.00484
No. 4	8.64	0.00484
No. 5 (12 min.)	9.12	0.00490

An analysis of this last material showed the following constituents: Nickel, 99.22 per cent., manganese, 0.59 per cent., iron, 0.14 per cent., copper, 0.03 per cent., carbon, 0.00 per cent.

These experiments indicate the conditions that must be observed to secure a nickel wire with the highest possible temperature coefficient of electrical resistance. The nickel must be the finest available. The additions of manganese must be as small as is compatible with subsequent forgeability. The time taken to melt should be a minimum, in order that the molten material shall be exposed for as short a period as possible to the effect of furnace gases. If these conditions are met, a high-grade nickel wire can be readily produced.

COMMERCIAL ALLOYS OF NICKEL

Malleable nickel ingots are produced in several grades. The most important are grades A, C, and D nickel.[2] In view of the fact that nickel has a high temperature coefficient at low temperatures and also a high resistance to oxidation at high temperatures, it was thought advisable to investigate the temperature coefficient of the material at high tem-

[2] See Bureau of Standards *Circular* 100

perature.   To this end, the resistances of samples of these various materials were measured over the range from 20° to 1000° C.

Monel metal, an alloy containing 67 per cent. nickel, 28 per cent. copper and 5 per cent. other metals (chiefly iron, manganese and silicon) is similar to nickel in that it has a high temperature coefficient, so a sample of this material was subjected to the same investigation.   Exact analyses of the materials used are not at present completed, but will be
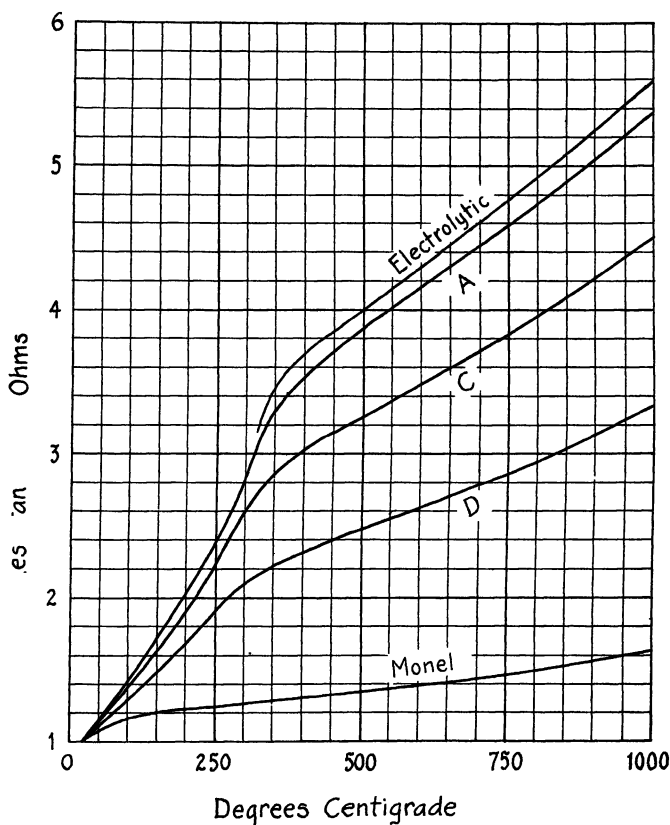
FIG. 1.

added to the report.   The resistances of the various materials at different temperatures, based on 1 ohm at 20° C., are given in Table 1, the values being taken from Fig. 1.   The breaks in the individual curves show the transformation point of nickel in each alloy.   For grade A, the transformation point is approximately 350° C., for grade C., 320° C., for grade D, 275° C., and for monel metal below 100° C.

In further investigating the transformation point of monel metal, the resistance of a sample was taken over the range from 20° C. to 160° C.
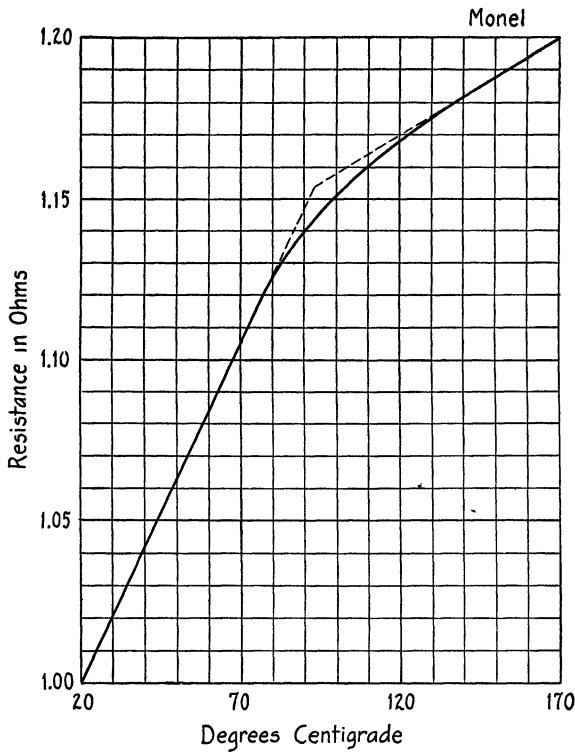
FIG. 2.

TABLE 1.—*Resistances of Nickel and Monel Metal at Different Temperatures*

Temperature, Degrees C.	Nickel			Monel Metal
	Grade A	Grade C	Grade D	
20	1.00	1.00	1.00	1.000
100	1.43	1.36	1.29	1.157
200	2.06	1.90	1.68	1.217
275			2.04	
300	2.83	2.58	2.10	1.254
325	3.07	2.74		
350	3.28	2.85		
400	3.52	3.02	2.31	1.294
500	3.87	3.26	2.46	1.339
600	4.15	3.49	2.62	1.384
700	4.44	3.61	2.78	1.434
800	4.73	3.96	2.94	1.488
900	5.04	4.22	3.13	1.550
1000	5.38	4.50	3.32	1.626

The observed values are plotted in Fig. 2, from which the values given in Table 2 are taken. Fig. 2 shows that the transformation point is 93° C.

TABLE 2.—*Resistance of Monel Metal at Different Temperatures*

Temperature, Degrees C.	Resistance, in Ohms	Temperature, Degrees C.	Resistance, in Ohms
20	1.0000	100	1.1508
30	1.0210	110	1.1598
40	1.0420	120	1.1685
50	1.0629	130	1.1753
60	1.0840	140	1.1830
70	1.1053	150	1.1890
80	1.1255	160	1.1942
90	1.1400	170	

The specific resistances of the samples investigated were:

MATERIAL	SPECIFIC RESISTANCE AT 20° C., OHMS PER MIL.-FT.
Grade A nickel	64
Grade C nickel	84
Grade D nickel	117
Monel metal	268

These values, together with the values given for the temperature coefficient of each metal, will give the specific resistance of the material at any temperature.

The composition of the wires mentioned in this paper are as follows:

	CARBON, PER CENT.	MANGANESE, PER CENT.	SILICON, PER CENT.	NICKEL, PER CENT.	COPPER, PER CENT.	IRON, PER CENT.
Grade A nickel	0.08	0.08	0.03	98.85	0.24	0.65
Grade C nickel	0.22	2.10	0.18	96.15	0.40	0.89
Grade D nickel	0.08	4.75	0.13	94.10	0.16	0.74
Monel	0.16	1.50	0.11	68.10	27.66	2.40

SUMMARY

1. Pure nickel with a low specific resistance and high temperature coefficient may be produced by melting electrolytic nickel with the addition of a minimum amount of manganese. The time for melting is a material factor in the production. The effect of various impurities is also indicated.

2. The variation of resistance with temperature for various grades of commercial nickel and of monel metal has been investigated. From the resistance-temperature curves, the transformation points have been obtained.

Experiments with Sherardizing

By Leon McCulloch,* East Pittsburgh, Pa.

When clean iron and metallic zinc dust, protected from the air, are heated below the melting point of zinc, the iron takes on a coating that has excellent protective value. This coating is a brittle alloy of zinc and iron, and the process is called *sherardizing*. Although the zinc particles are covered with oxide and cannot touch the iron, an extremely small amount of the zinc vaporizes and the vapor, reaching the iron, alloys with it. Other zinc then vaporizes to replace that which has been removed from the atmosphere. This process continues indefinitely at a decreasing rate, for the coating already formed permits zinc to diffuse through it to combine with the iron beneath. In this way, the coating grows outward from the surface of the iron, as the bark grows from a tree.

The method of growth was clearly seen while sherardizing small cubes of cast iron, at about 450° C., in a small glass-stoppered bottle enclosed in a rotating drum. The dust contained about 3 per cent. iron, which prevented caking, and coatings ½ in. thick were grown in a few hours. After a heavy layer had been formed, the cubes were again treated for a shorter time. The layer last formed was seen to lie next to the iron, instead of at the outer surface, showing that the growth was from the surface of the iron outward. The coatings split at the edges and corners, leaving the iron exposed there. Flakes of graphite from the iron were lifted upward and embedded in the coatings.

The composition of sherardized coatings varies from the outer surface inward, as would be expected from the mode of growth. That is, the iron content increases slowly at first, then more rapidly as the iron is approached. At the surface the least amount of iron possible is 6 per cent., as the experiments described below have shown.

A study was made of the effect of iron in zinc dust on the process and on the resulting coatings. Glass bulbs, holding the small pieces to be sherardized, together with enough zinc dust to make them two-thirds full, were heated in a small, rotating electrically heated and controlled furnace. The bulbs had 6-in. (15 cm.) necks, of ⅜-in. (9.5 mm.) tubing, plugged

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with asbestos and zinc dust to absorb oxygen that otherwise would diffuse inwardly and oxidize the contents of the bulbs.

The dusts contained iron in amounts varying from 0 to 26 per cent. They were prepared by mixing 200-mesh zinc in the desired proportions with powdered electrolytic iron of the same fineness. These mixtures, before using, were heated until the iron particles had been thoroughly alloyed. The dusts containing less than 20 per cent. iron were no longer magnetic, so that magnetic separation of the iron was impossible. Only oxidized particles or free iron could then be taken out. (In these dusts, pieces of very thin iron foil were placed and heated for long periods, until it was thought that they had taken up all the zinc possible. In order to hasten the action, the foil before using was converted into an alloy with 50 per cent. zinc, by heating to redness in a dust containing 30 per cent. iron.) The alloyed strips after the long heating in these dusts were carefully freed from dust and were analyzed. The results were as follows:

IRON IN DUST. PER CENT.	PER CENT. IRON FOUND IN RESULTING ALLOYS AFTER				
	2 DAYS 365° C.	3 DAYS 415° C.	5 DAYS 415° C.	14 DAYS 415° C.	45 DAYS 415° C.
0	6.1	6.0		6.0	6.15
3		6.3		6.0	
5					6.10
7					7.81
9					11.13
10			14.1	15.5	
11					14.9
14				20.2	
18				21.7	
22				27.7	
26				43.6	

From the curve between iron in the dusts and iron in the resulting alloys, certain conclusions can be drawn:

It is evident that no part of a sherardized coating can contain less than 6 per cent. iron, as heating even for 45 days fails to lower the amount. When the dust contains less than 6 per cent. iron, the resulting alloy contains 6 per cent. As the iron is increased from 6 to about 10 per cent. the composition of the alloy changes proportionally. This indicates the existence of the solid solution n, varying in composition from 6 to 10 per cent. iron. When the iron in the dust exceeds 10 per cent., the composition of the alloy changes rapidly to about 20 per cent. And when 20 per cent. iron in the dust is exceeded, the iron in the alloy rises abruptly to a high value; the compound $FeZn_3$ is thus indicated.

These results are in general accord with the well-known equilibrium diagram for the zinc-iron alloys.[1] There is this material difference:

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[1] G. H. Gulliver: "Metallic Alloys," 336. Ed. 3. London, 1919. Griffin & Co., Ltd; Von U. Raydt and G. Tammann: *Zeit. Anorg. Chem.* (1913) **83**, 257–266.

These results show iron to be insoluble in zinc·at temperatures below the melting point, while the diagram indicates a solid solution containing about 1 per cent. iron.

The indicated constituents of a sherardized coating are the following: The solid solution n, varying in composition between 6 and 10 per cent. iron; the compound $FeZn_3$; and possibly the 20 per cent. solution of zinc in iron described by Raydt and Tammann. Next to the iron is the solid solution of zinc in iron, in amounts too small to be detected; then there is a very thin layer of the compound $FeZn_3$, and beyond this the relatively thick layer of solid solution n, on which the protection of the iron depends. $FeZn_3$ can have little protective value, for it readily rusts when placed in water. For example, coatings formed in dust containing 11 per cent. iron immediately rusted in salt spray, while coatings formed in dust with 9 per cent. iron showed good resistance.

The iron content in a dust should be low, but there is some question as to what the limits should be. "Dross" dust containing iron is cheap and the percentage of iron increases with use, as zinc is removed while the iron remains. Iron in dust is useful as it raises the melting point and prevents fusing or caking when a high temperature is used to give rapid deposition. Iron, however, rapidly lowers the rate of deposition, making it necessary to employ higher temperatures, as the following figures show:

Iron in Dust, Per Cent.	Temperature to Deposit 0.1 g. Zinc per Hour, Degrees C.
0	375
5	400
7	450
10	Above 600 (?)

From these figures, it is concluded that the iron content should not exceed 7 per cent. An analysis of the dust should include the metallic iron. The important thing is the ratio of the metallic iron to the metallic zinc, rather than the percentage of iron to the total dust.

Life tests show that there is a minimum weight of zinc per square inch necessary for good coatings; to test the value of sherardizing, the weight of zinc should therefore be determined. There are various proposed methods for doing this. One approximate way is by the Preece or copper-sulfate test.

A new method depends on the fact that a boiling 10 per cent. solution of ammonium chloride removes zinc from sherardized coatings rapidly, but does not dissolve iron until the zinc is practically gone. This test may be applied in two ways: The coating is completely removed and its amount determined by loss in weight; or the weight of the coating is found approximately by noting the time for iron to appear in the boiling solution. At each 5 min. interval a drop of the solution is removed upon

a spot-plate and tested with potassium ferricyanide for the presence of iron. When the blue color appears, the coating is gone, and the time is a measure of the weight of zinc. A good coating should withstand this test for 10 or 15 minutes.

A comparison of short-time rusting tests, such as the salt spray, with tests in actual service, shows that rusting in the short-time tests does not necessarily mean a short life in service. The rapid rusting tests are, however, of value in judging the quality of a coating, when an actual determination of the weight of zinc is made at the same time. These two methods of testing should supplement each other.

DISCUSSION

FRED L. WOLF,* Mansfield, Ohio.—Marshall also found that if he sherardized at 370° or less, for a period of 3 hr., he had three iron-zinc layers. If his temperature or his time was increased beyond that point, he had five layers. It would be rather difficult to find the exact composition of those layers.

The author's theory is that the deposition is due to the formation of vapor, which is absorbed. Is it necessary that the pieces be in actual contact with the zinc at all times, or would it be possible to sherardize by having the samples in the air-tight zinc receptacle, and would particles that were not submerged in the zinc sherardize? I have made a few experiments, putting plain steels in an air-tight receptacle and heating them beyond the sherardizing temperature, but I have not found that the deposit went above the point of contact in dust.

LEON McCULLOCH.—To sherardize at a distance, it is necessary to have a very long time, or else do it in an evacuated vessel, so that the molecules of zinc have a free path and can diffuse rapidly to the iron. The atmosphere of the drum must be free of oxygen, or the zinc will be changed to oxide before reaching the iron. Usually it is the presence of oxygen that prevents the sherardizing of parts not actually in contact with the zinc dust.

WILLIS M. PEIRCE,† Palmerton, Pa.—In the last few months, I have made microscopical examinations of zinc coatings of various kinds, including some sherardized specimens. I have also done some work on the constitution of iron-zinc alloys, and I agree with the author that zinc cannot hold 1 per cent. of iron in solid solution. The author does not mention the compound $FeZn_7$, which contains 10.2 per cent. iron but holds zinc in solid solution up to a composition of approximately 7 per

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† Research Division, New Jersey Zinc Co.

cent. iron; this compound would correspond quite closely with the 6 per cent. alloy mentioned by him.

While I have not examined many specimens, I have not seen a sherardized coating that bore a strong resemblance to the solid solution of zinc in $FeZn_7$ containing approximately 7 per cent. iron. Galvanized coatings have a distinct layer of approximately that composition, but I have not seen a sherardized coating the main part of which I could identify as that alloy. I am, therefore, inclined to doubt whether the composition of a typical commercial sherardized coating is uniformly that high in iron from the surface of the coating to its junction with the iron base.

Has an analysis been made of a commercial sherardized coating stripped from the iron base with hydrochloric acid? That method of stripping sherardized coatings is simple and should be an easy method of determining the average iron content of a sherardized coating.

JESSE L. JONES,* Pittsburgh, Pa., (written discussion).—Sherardizing has been exploited more than it deserves. No better coating can be obtained than is produced by a molten bath of electrolytic or similar high-grade zinc; this gives a thin, hard, tough coating that will last a lifetime even under severe conditions of service. The sherardized coating, on the other hand, is always weak and brittle. If the work is well done, the coating usually has a long life; but in many cases the coating is of variable thickness, especially where there are recesses. In such cases the coating, to use an expression current with electroplaters, "does not throw well." With the hot galvanized coating, any bare spots on an article can be rubbed with sal-ammoniac by the galvanizer until well coated with zinc. Thin spots or defective places on sherardized articles cannot be detected by visual inspection or remedied. Where sherardizing is necessary, as in the case of threaded bolts, nuts, etc., much better results are obtained by using a zinc dust under 80 per cent. metallic zinc and maintaining a temperature in excess of 375° C., as intimated by the author.

O. W. STOREY, Madison, Wis. (written discussion).—From the first paragraph, it is evident that the author regards sherardizing as a vapor process. I do not agree with him; although zinc vapor has some action, part of the sherardizing action is caused by the actual contact of the zinc and the iron. The almost negligible vapor pressure of zinc at the temperature at which sherardizing may be done makes it impossible to believe that the action is entirely due to the zinc vapors. This matter has been discussed elsewhere.[2] It has been my experience that, on a

* Metallurgist, Westinghouse Elec. & Mfg. Co.

[2] O. W. Storey: The Sherardizing Process. *Met. & Chem. Eng.* (June 15, 1916) 14, 683.

commercial scale, it is impossible to sherardize at low temperatures unless the zinc dust is in actual contact with the iron. If there is the slightest cavity in the zinc dust, when a stationary process is used, the iron will not be coated.

The work of the author in determining the definite amount of iron in a sherardized coating is interesting and instructive, although misleading deductions may be made from this work. Sherardized iron has several layers of zinc-iron alloys and the layer richest in zinc will contain about 6 per cent. of iron. It must be borne in mind, however, that in actual sherardizing the presence of the iron base tends to increase the amount of iron in the sherardized coating, and the higher the temperature the smaller is the amount of the zinc-iron alloy high in zinc. With low-temperature sherardizing, the 6 per cent. alloy will form the bulk of the coating, whereas with high-temperature sherardizing the 6 per cent. alloy will form a small part of the coating. The presence of the base iron is the important feature in the commercial process and the deduction must not be drawn that a zinc dust low in iron will give a coating that contains but 6 per cent. iron. The sherardized article may have a thin superficial layer of relatively pure zinc on the outside, the formation of which may be caused by mechanical inclusions. An interesting fact is that pure zinc cannot be sherardized; that is, a deposit of zinc cannot be formed on pure zinc when introduced into the sherardizing drum.

In discussing the zinc-iron alloys, the author does not mention the equilibrium diagrams of Wologdine and Von Vegesack. These diagrams differ somewhat and for that reason the exact nature of the zinc-iron alloy formed by sherardizing is in doubt. It is my belief that a true compound of iron and zinc is formed containing from 7 to 8 per cent. The exact formula for this compound is not known, although it is somewhere in the neighborhood of $FeZn_{10}$, this may be deduced from the diagram of Wologdine. Experimentally, if iron is added to zinc, a microscopic examination shows that a hard crystalline material that separates from the pure zinc is formed. These crystals increase with the iron content and at about 7 to 8 per cent. occupy the entire field. The cystals have a high melting point and are the cause of the mushy consistency of dross when removed from galvanizing pots. If the iron content is increased to about 8 per cent., a high melting point alloy is formed corresponding about to the formula $FeZn_{10}$. If this alloy is poured into spelter molds and allowed to cool, it expands and rises much as bread during baking. The resulting alloy is a mass of loose crystals, which may be used for sherardizing. This is one method of manufacturing dross crystals for sherardizing.

When two metals alloy in the sherardizing process, metallic compounds are formed. If zinc is coated (in a manner similar to sherardizing) by other metals, compounds apparently are formed. This formation of

compounds is well illustrated when copper is sherardized. Two alloys of copper and zinc are formed, one high in copper and conforming to the formula CuZn and the other high in zinc and conforming to the compound $CuZn_2$. These two compounds, together with the copper base, illustrate the alloying action, for the high-zinc alloy on the outside is white and hard, and the alloy in the center brass-colored. When copper is "sherardized" in metallic antimony, two layers are formed, one being the purple $SbCu_2$. From these data, it would appear that a zinc-iron alloy in the form of a definite compound is formed rather than a solid solution, as indicated by the author.

My experiments on large-scale operations with dross crystals bear out the statements by the author that this dross may be used, but a higher temperature is necessary. While on a commercial scale the iron content of the dross crystals after continued use did not increase, which is contrary to the statement of the author, there are many other factors that must be considered in commercial work.

It has been my experience that when the iron content of a coating increases to about 15 per cent., the sherardized coating does not show sufficient resistance to rusting. Such a coating has a relatively thin layer of the author's 6 per cent. alloy protecting the $FeZn_3$, or high-iron alloy, forming the bulk of the coating. This high iron content may be due to high temperatures in the process or to a high iron content of the sherardizing material, which in turn requires a higher temperature. By using a low temperature with a zinc dust of high metallic content, or a clean dross dust, it is possible to obtain a coating that does not turn red with rust and is highly resistant to weathering. The coating, instead of turning rusty, gradually turns to a gray and finally to a deep brown black, indicative of a magnetic iron-oxide coating.

The work of the General Electric Co. along this line is of much importance and shows the necessity of using a zinc dust high in metallic zinc so that a low temperature may be used. It shows that the tendency is for the coatings to be porous with a low metallic-zinc content, and that a porous coating is less resistant. The inclusion of this zinc oxide in the coating also indicates that the sherardizing action is caused, in a large measure, by an actual contact alloying.

DAVID R. KELLOGG, East Pittsburgh, Pa.*—The point that should be emphasized is not the extent of vapor pressure nor the magnitude, but the rate at which it is maintained, because one may have an almost infinitely low vapor pressure and, granted that it could be maintained, one may expect a deposition of any magnitude desired.

The remarks about the composition of the dust recalled an experience we had when trying to determine the metallic zinc content of sherardizing

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dust. One man, who alone could get the same results twice, treated the zinc dust with ferric chloride at the proper temperature, which he determined by touching the flask, for 10 min., and then titrated with permanganate. This is a rather difficult procedure, as the highly colored ferric-chloride solution masks the color change of the permanganate. Our present method is to treat a $\frac{1}{2}$-gm. sample of the dust with 50 cc. of ferric-chloride solution (300 gm. per liter) for 10 min. at 60° to 70°. Then the flask and contents are cooled, 30 cc. of 30 per cent. H_2SO_4 added, and titrated with $K_2Cr_2O_7$, using the electrometric method of titration. We used a 250-cc. Florence flask, provided with a two-hole rubber stopper, in one hole of which is an inlet tube and in the other a Bunsen valve. Before adding the ferric chloride, we pass natural gas through the flask a few moments to prevent oxidation. This titration with dichromate is easy when the electrometric method is used, as colored solutions cause no difficulty and no judgment is required. The figure obtained by this titration must be corrected for the metallic iron present, which is obtained by dissolving $\frac{1}{2}$ gm. of the dust in H_2SO_4 and then titrating electrometrically with dichromate. Then $1\frac{1}{2}$ times the number of cubic centimeters used in the iron titration is subtracted from the figure obtained in the first titration, which gives the true metallic zinc equivalent.

WILLIAM H. FINKELDEY, Palmerton, Pa.—What effect has the size of the particles of zinc dust on the character or quality of the coating produced?

LEON McCULLOCH.—I have not made any experiments with zinc dust of various finenesses. I am convinced, however, that as long as the dust is within reasonable fineness, good sherardizing can be done with it. The main thing is to maintain the amount of zinc in the dust and maintain the proper proportion of iron to metallic zinc.

FRED L. WOLF.—We have found that the thicker coating does not necessarily mean better protection. If the coating was put on at around 370° C. and averaged $\frac{1}{2}$ to $1\frac{1}{2}$ oz. per sq. ft., we succeeded in holding it in the salt spray without rusting for 170 hr.; if the coating exceeded $1\frac{1}{2}$ oz. per sq. ft. the protection was no better; in other words, $1\frac{1}{2}$ oz. seemed to be about the point where the curve went the other way.

L. H. MARSHALL,* Washington D. C. (written discussion).—Sherardizing, from both the theoretical and the practical viewpoints, is a most interesting process and an inviting field for investigation. As the effects of many of the factors are problematic, new data, such as those contained in this paper, are welcome. Three of the statements, however, seem open to question. The author concludes that 6 per cent. is the smallest amount of iron that can possibly be present in any part of the

* Research Associate, Bureau of Standards.

sherardizing coating. This assertion is evidently based on the fact that 6 per cent. was the average iron content of the thin strips of iron-zinc alloy that were heated for a long time, up to 45 days, at 415° C. in iron-free zinc dust; the assumption apparently was made that the composition of the resulting iron-zinc alloy was uniform throughout. Such an assumption is not warranted unless some information derived by microscopic inspection, sectional analyses, or the like is at hand; the elimination of variations in composition by heat treatment alone is frequently impossible. Peirce states[3] that even very small amounts of the constituent $FeZn_7$ in a matrix of zinc remain unabsorbed after annealing at 400° C. for 110 hr. It may well be, therefore, that the samples produced by McCulloch were not uniform in composition but consisted of a series of layers of iron-zinc alloys of varying composition. If such is the case, the conclusion as to the smallest amount of iron possible in any part of a sherardized coat is unwarranted.

Exception might also be taken to the statement made (just before the table listing the variation in sherardizing temperature with the iron content of the dust) that the iron in zinc dust is useful; the author admits, though, that the presence of iron necessitates the use of higher sherardizing temperatures. The higher temperatures are undesirable because they tend to increase the amount of iron in the coat. This high iron content does not necessarily decrease the corrosion resistance of the deposit but, because of the brittle alloys formed, it does increase the flaking and chipping of the coating under impact or bending. Such a consideration is important in view of the rough handling many sherardized articles receive.

The practicability of the proposed method for determining the amount of deposited zinc by stripping in a boiling 10 per cent. ammonium-chloride solution is questionable until more data are available on the subject. It seems inconsistent, however, to use the appearance of iron in the stripping solution, as shown by the potassium-ferricyanide test, for the indication that the coating has been entirely dissolved, when the presence of iron in the coating itself has been admitted. It is quite improbable that the zinc alloyed with the iron would be entirely dissolved without any of the ferrous material going into solution. In fact, when the antimony chloride-hydrochloric acid stripping solution is used, the deposited zinc is entirely dissolved, as shown by analysis of the used reagent, but the alloyed iron accompanies the zinc. Consequently it is doubtful if a method that does not permit any iron to be dissolved would account for all the zinc deposited.

LEON McCULLOCH (author's reply to discussion).—Hydrochloric acid was not used during this work, but many coatings were removed for

[3] Studies on the Constitution of Binary Zinc-base Alloys. See page 767.

analysis by the ammonium-chloride method.   Coatings made in dust with less than 5 per cent. iron, varying in weight from 0.1 to 1.0 gm. zinc per sq. in., average 8 to 10 per cent. iron.   The thinner coatings and those made in dust higher in iron are higher in iron.   It is quite evident from these analyses that the main constituent of good coatings is the solid solution of $FeZn_7$ and zinc containing from 6 to 10 per cent. iron.   The author has made no inspection of these coatings with the microscope. Different structures might be expected, however, in this solid solution in sherardized and in hot galvanized coatings, for the conditions of formation are quite different in the two cases.

The essential thing, in sherardizing, is the existence of a solid solution through which the vapor of one metal can diffuse to combine with the other.   This was shown by sherardizing aluminum with zinc.   In this case zinc is soluble in the metal which it is coating, but no compound is formed.   (To sherardize aluminum, the temperature must be high, about 450° C., and a dust high in iron must be used, in order that zinc may not deposit rapidly enough to form the low-melting eutectic alloy.)

The uniformity of the alloys showing a minimum iron content of 6 per cent. even after prolonged heating with iron-free zinc dust has been questioned.   While an inspection under the microscope has not been made, there is little doubt that uniformity would be discovered.   If this reaction of zinc with iron were very slow, non-uniformity might be suspected.   But this action is quite rapid, so that the 6 per cent. limit to the iron is reached in a few hours.   The probability of non-uniformity is thus very small indeed.

It should have been stated that when the ammonium-chloride method of stripping or testing coatings is used, the iron in the coating remains upon the surface in a spongy form and does not appear in the solution until the zinc is practically gone.   This spongy iron is, however, readily brushed away in order to determine the coating by loss in weight.   The following solution also may be used cold for removing coatings for analysis: Ammonium chloride 35 gm.; ammonia (sp. gr. 0.90) 150 gm.; and water 325 gm.

This selective solution of zinc is similar to the action that takes place when a coating is corroded in service.   It is the spongy iron, left behind after some of the zinc has been removed, that in time gives to good sherardizing its well-known gray or black appearance.

Studies on the Constitution of Binary Zinc-base Alloys*

By W. M. Peirce,† M. S., Palmerton, Pa.

The present work has been done in an endeavor to correlate and complete the data on the constitution of alloys of zinc with other common metals, dealing exclusively, however, with the zinc-rich alloys in which the zinc content is between 90 and 100 per cent. This investigation has included the systems lead-zinc, cadmium-zinc, iron-zinc, copper-zinc, aluminum-zinc, nickel-zinc, manganese-zinc, cobalt-zinc, tin-zinc, and magnesium-zinc. The important ternary system zinc-lead-cadmium and the zinc corners of other ternary systems have been investigated in connection with this work but will not be dealt with here.

Zinc does not form a single homogenous zinc-rich solid solution over any considerable range of concentration with any of the metals mentioned. A limited range of solid solubility is said, by previous investigators, to occur in several cases and certain effects on the properties of zinc are attributed (with varying degrees of consistency among different writers) to the presence of certain other metals dissolved in zinc. The increasing use of rolled zinc and the possibilities offered by zinc as a casting alloy in certain fields have made it essential to determine precisely these equilibrium relationships.

In carrying out this investigation reliance was placed chiefly on two methods, conductivity measurements and microscopic analysis. The first method proved valuable in determining the existence and extent of solid solubility. The second method stands alone as a means of determining structural characteristics and supplements the first method in the measurement of solid solubility. The scleroscopic hardness of all conductivity specimens was measured partly for its value from the point of view of mechanical properties and partly as a check on the conductivity curves in indicating the limits of solid solubility. No hardness curves are included in this paper, for in no case were they important in arriving at the constitution of any system.

Conductivity measurements were in all cases made on cold-rolled strips 0.020 in. thick and 0.5 in. wide, both as cold rolled and after an

anneal of $\frac{1}{2}$ hr. at 150° to 200° C. to produce recrystallization. A Kelvin double bridge was used and measurements made at 20° C. The temperature was maintained by means of a well-stirred, water-jacketed bath of light oil. The temperature was measured to 0.05° C. and correction made accordingly. For the early part of the work, micrometer measurements were depended on to compute the cross-section but in later work weight, length, and density determinations were used in these computations. The over-all accuracy of these determinations was calculated to be about 0.25 per cent., and different test strips from the same coils usually checked well within this limit.

With regard to the methods employed in the preparation of micro-specimens, attention is drawn to the method of polishing and particularly to the etching reagent described in the paper by Mathewson, Trewin and Finkeldey.[1] This reagent is made by dissolving 20 gm. C. P. chromic anhydride and 1.5 gm. C. P. sodium sulfate (anhydrous) in 100 cc. water. It should be noted, however, that the chromic anhydride must be free from SO_4, a requirement which is not met by all so-called C. P. grades. Neglect of this precaution will result in stains. In every case this reagent has proved superior to other etching agents in developing the structure of zinc and in differentiating secondary constituents. In one or two cases, other reagents have been found useful in developing the zonal structure of non-homogeneous solid solutions.

LEAD-ZINC

The equilibrium diagram shown in Fig. 1 is based on a careful study of the methods and results of Matthiessen and Bose,[2] Spring and Romanoff,[3]
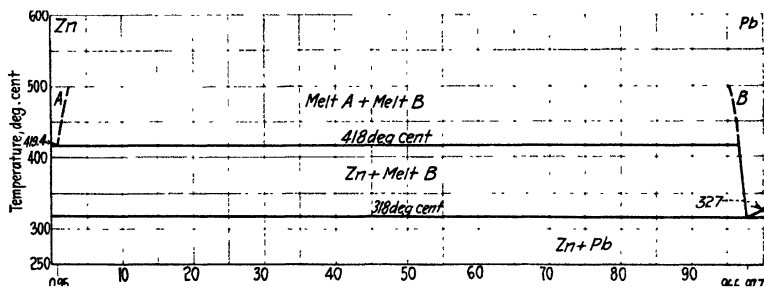
FIG. 1.—LEAD-ZINC EQUILIBRIUM DIAGRAM; BASED ON DATA OF HEYCOCK AND NEVILLE, ARNEMANN, PEIRCE.

Arnemann,[4] and Heycock and Neville.[5] The eutectic lies at 318° C. and 2.3 per cent zinc. The *monotectic* point (*i.e.*, the concentration and tem-

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[1] Some Properties and Applications of Rolled Zinc Strip. *Trans.* (1920) **64**, 305.

[2] Matthiessen and M. von Bose: *Proc.* Royal Soc. London (1861) **11**, 430.

[3] Spring and Romanoff: *Zeits. Anorg. Chem.* (1896) **13**, 29.

[4] Arnemann: *Metallurgie* (1911) **7**, 201.

[5] Heycock and Neville: *Jnl.* Chem. Soc. (1892) **61**, 888; (1897) **71**, 383.

perature at which the single crystal phase, zinc, freezes out at constant temperature with no further change in concentration of the rich liquid phase but with the formation of a second lead-rich liquid phase) lies at 418° C. and 0.95 per cent. lead. (These figures are by Heycock and Neville whose work deserves most weight.) The best value available for the solubility of zinc in lead at the monotectic temperature is 3.4 per cent., as given by Arnemann. In the present investigation, conductivity and hardness measurements gave no evidence of any solid solubility, Fig. 11. Microscopically, a secondary constituent can be detected when lead is present in a concentration as low as 0.03 per cent. Fig. 19 shows zinc containing 0.05 per cent. lead.

The distribution of the lead is of interest. According to the equilibrium diagram, an alloy containing 0.5 per cent. lead, when the course of freezing has reached the end of the monotectic horizontal, should consist of 98.8 per cent. primary zinc and 1.2 per cent. of the lead-rich liquid phase; the latter, when completely frozen, consists of one-fifth primary zinc and four-fifths eutectic containing 97.7 per cent. lead.

The lead-rich liquid phase formed during the course of freezing along the monotectic line must necessarily remain in the interstices of the primary mass of zinc crystals. Microscopic examination shows that it is entrapped as small globules between the growing branches of the zinc crystals. Fig. 20 shows a linear group of these small globules at high magnification and Fig. 19 shows the general distribution of these droplets, which in a microsection appear as a polygonal network composed of these linear groups. Fig. 21 shows the effect of rolling in elongating these "lead lines."

The fact that the lead-rich liquid phase forms globular droplets instead of attenuated intercrystalline films, together with the difference in hardness between zinc and lead and its resultant effect in polishing and etching, makes detection of very small percentages of lead possible. The soft minute globules of lead appear in many cases to be torn out during the polishing and the ensuing pits are enlarged by the etching treatment.

CADMIUM-ZINC[6]

In Fig. 2 is shown the equilibrium diagram of the cadmium-zinc system. The liquidus is as given by Heycock and Neville.[7] The eutectic horizontal at 264° is shown as ending about 2 per cent. from the cadmium side, as several investigators report solid solubility of a few per cent. of zinc in cadmium. On the zinc side, the discrepancy in the results of

[7] *Op. cit.*

Hindricks,[8] Curry,[9] Arnemann,[10] Lorentz and Plumbridge,[11] Kurnakow and Zemczuzny,[12] Glasnow and Matweew,[13] and Ludwig,[14] in regard to solid solubility makes an entire reconsideration of the problem necessary.[15]

Microscopic examination of chill-cast alloys showed that secondary cadmium is present in alloys containing above 1.0 per cent. cadmium. Fig. 23 shows white areas of cadmium (divorced and coalesced eutectic) in an alloy containing 1.25 per cent. cadmium after an anneal of 10 hr. at 250° C. Fig. 22 shows a specimen from the same chill-cast ingot unannealed. Here the cadmium is present in the thin films of eutectic which appear as dark streaks between the grains. (The hydrochloric-acid etch employed does not differentiate the constituents as well as might be desired.)
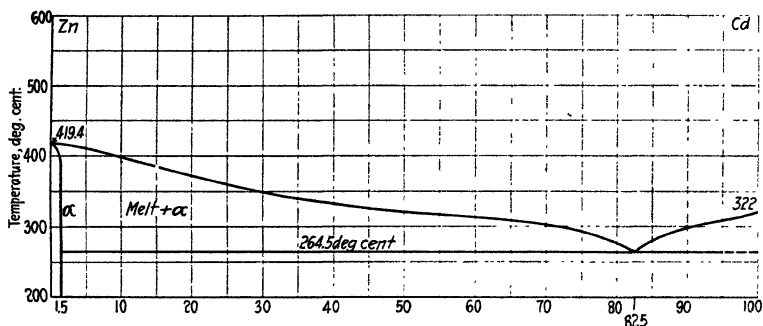
FIG. 2.—CADMIUM-ZINC EQUILIBRIUM DIAGRAM; BASED ON DATA OF HEYCOCK AND NEVILLE, BRUNI, SANDONNINI AND QUERCIGH, PEIRCE.

The specimens shown in Figs. 22 and 23 were etched with alcoholic hydrochloric acid to develop the zonal structure that appears in Fig. 22. Ten hours annealing at 250° C. was necessary to remove this zonal structure in a specimen of the same alloy (which contains 1.25 per cent. cadmium) shown in Fig. 23. It is evident from this that equilibrium is attained quite slowly in these alloys at moderate temperatures. Five weeks' anneal just below the eutectic temperature caused absorption of all visible amounts of secondary cadmium in an alloy containing 1.5 per

[8] Hindricks: *Zeits. Anorg. Chem.* (1907) **55**, 415.
[9] Curry: *Jnl. Phys. Chem.* (1910) **13**, 589.
[10] *Op. cit.*
[11] Lorentz and Plumbridge: *Zeits. Anorg. Chem.* (1913) **83**, 228.
[12] Kurnakow and Zemczuzny: *Ibid.* (1908) **60**, 32.
[13] Glasnow and Matweew: *Intnatl. Zeits. Metallographie* (1914), **5**, 1.
[14] Ludwig: *Zeits. Anorg. Chem.* (1916) **94**, 161.
[15] Since completing this investigation, recent work by Miss K. E. Bingham [*Jnl. Inst. Metals* (1920) **24**, 333] has come to the author's attention. Her results are reasonably consistent with those reported here. However, it is believed that the long anneals employed by the author are necessary for the attainment of equilibrium and hence it seems doubtful whether the limit of solid solubility which she reports really represents the equilibrium condition at 150° C. Probably it conforms more closely to the chill-cast state.

cent. cadmium, Fig. 24, while in an alloy containing 2.0 per cent. cadmium, secondary cadmium was still clearly visible.

Conductivity measurements were made on strips containing 0.03, 0.05, 0.10, 0.21, 0.42, 0.63, 0.95, 1.25, 2.44, 4.39 per cent. cadmium, in both the cold-rolled condition and after annealing ½ hr. at 150° C. to recrystallize. Both curves (see Fig. 11) show a minimum followed by a sharp rise and then a slow uniform drop. The lowest point of the inflection lies near 1.0 per cent., the concentration at which eutectic was first detected in microspecimens from unhomogenized chill castings.

The evidence is, in the writer's opinion, consistent in showing that in chill castings about 1.0 per cent. of cadmium is held in solid solution by zinc and that under conditions of approximate equilibrium at 250° C. the concentration is higher, being at this temperature at least 1.5 per cent. cadmium. It was felt that an unwarranted length of time would be required to determine the course of the solid solubility curve at lower temperatures.

IRON-ZINC

Among the more recent investigations of this system are those of Vegesack[16] and Raydt and Tammann.[17] The equilibrium diagram, as described by these investigators, is shown in Fig. 3, except that the solid solubility of iron, or $FeZn_7$, in zinc has been reinvestigated and the diagram modified in accordance with the results obtained in this laboratory. The frequently quoted statement of Vegesack that zinc holds 0.7 per cent. iron in solid solution is greatly in error and is apt to be misleading since it might be assumed that any amount of iron ordinarily found in zinc as an impurity exists in solid solution. As a matter of fact, the limit of the solid solubility of iron in zinc lies in the neighborhood of 0.02 per cent. iron.

Microscopically, iron is easily identified in zinc, as the compound $FeZn_7$ is a hard white crystalline constituent left in high relief by polishing and less rapidly attacked than the zinc matrix in etching (marked in Fig. 25). In this laboratory, small widely scattered crystals of this compound have frequently been observed in specimens containing as low as 0.025 per cent. iron and these remain unabsorbed after 110 hr. annealing at 400° C.

The conductivity curves of specimens both in the cold-rolled condition and after annealing ½ hr. at 150° C. to bring about recrystallization show inflections in the region of 0.04 per cent. iron, Fig. 14.

COPPER-ZINC

In the numerous general investigations of this system, only a small number of zinc-rich alloys have been studied. Fig. 4 shows the consti-

[16] Vegesack: *Zeits. Anorg. Chem.* (1906) **52**, 30–40.
[17] Raydt and Tammann: *Ibid.* (1913) **83**, 257–266.
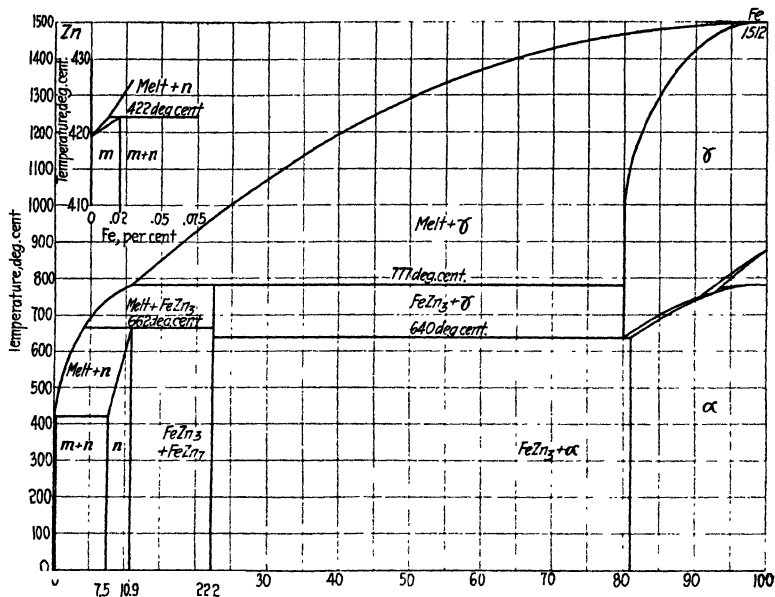
FIG. 3.—IRON-ZINC EQUILIBRIUM DIAGRAM; BASED ON DATA OF RAYDT AND TAMMANN, VEGESACK, PEIRCE.
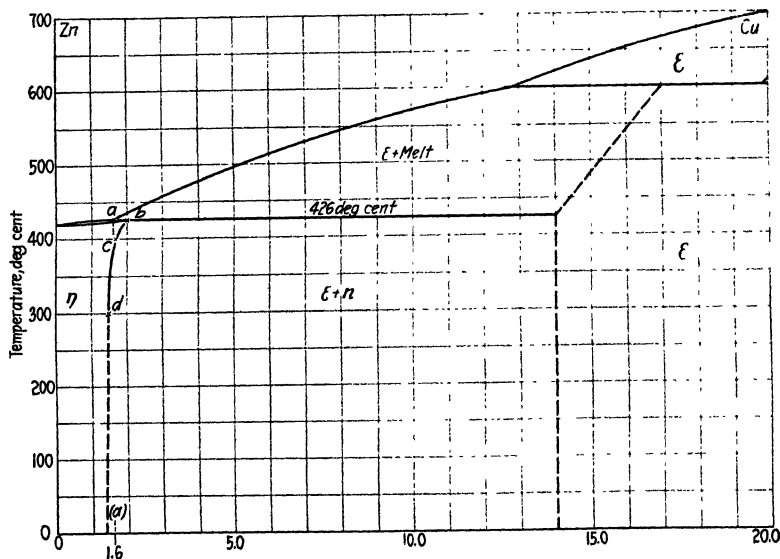
FIG. 4.—COPPER-ZINC EQUILIBRIUM DIAGRAM; BASED ON DATA OF TAFEL, HAUGHTON AND BINGHAM, PEIRCE.

tution of these alloys from 90 to 100 per cent. zinc, based on Tafel's revision of the copper-zinc system[18] supplemented by the results of the present investigation.[19]

Two solid solutions, known as ϵ and η, have been identified in the range of concentration under present consideration by earlier investigators. The limit of solid solubility of the ϵ constituent in the η has not been determined, however. In this case, the microscopic method furnished an effective means of determining the solid solubility curve. Figs. 26, 27, and 28 show the effect of annealing for 36 hr. at 400° C. and quenching from the same temperature upon alloys containing 1.81 and 1.95 per cent. copper, respectively. In the former case, the alloy has become homogeneous; while in the latter case, primary ϵ crystals remain unabsorbed. This locates the point c on the equilibrium diagram. Figs. 29, 30, and 31 show the effect of annealing and quenching a specimen containing 1.5 per cent. copper first at 400° C., then at 300° C., and then at 350° C. By a series of such experiments, the point d was located. Point a was determined from the fact that in chill-cast specimens, the ϵ constituent first appears in the specimen containing 1.81 per cent. copper and is totally absent in the 1.5 per cent. copper alloy. The zonal structure shown in Fig. 26 is completely eliminated by 2-hr. anneal at 350° C.

The conductivity curves of both cold-rolled strips and similar material softened by annealing, Fig. 12, show inflections at about 1.25 per cent. copper. It may be observed that the conductivity of the strips recrystallized by annealing is consistently higher than that of the cold-rolled material. This is true of all observations reported in this paper except in the case of the zinc-aluminum alloys. No conductivity determinations were made on the heat-treated alloys used for microstructural observations.

An interesting fact to be noted in connection with the conductivity of these alloys is that the first small addition (0.05 per cent.) of copper to zinc raises the conductivity slightly above that of pure zinc.

ALUMINUM-ZINC

Two recent comprehensive investigations of this alloy system have been undertaken. Rosenhain and Archbutt,[20] in 1912, brought the equilibrium diagram to the form shown in Fig. 5, except for the location of the points H and L, the ends of the 443° and 256° C. horizontals. Bauer and

[18] Tafel: *Metallurgie:* (1908) 12, 349–352; 13, 375–383.

[19] Lately, the work of Haughton and Bingham [*Proc.* Roy. Soc. London (1921) A99] has come to the writer's attention and the present diagram shows the liquidus and solidus referring to the η constituent as given by them. Their result for the temperature of transformation $\epsilon \rightarrow \eta$ has also been used. Other portions of their diagram are consistent with the diagram here given.

· [20] Rosenhain and Archbutt: National Physical Laboratory *Collected Researches* VIII (1912) 41–77.

Vogel[21] determined these points and also checked Rosenhain and Archbutt's results on the rest of the diagram. The fact that these two investigations are in substantial agreement makes it possible to accept the results without further investigation, except in regard to minor details.

One point in which these previous investigations are incomplete, however, concerns the exact location of the solid-solubility curve of aluminum in zinc. Both investigators agree that limited solid solubility occurs and Bauer and Vogel have determined microscopically that at the single temperature 380° C., the saturation limit is between 0.5 and 1.0 per cent. aluminum.
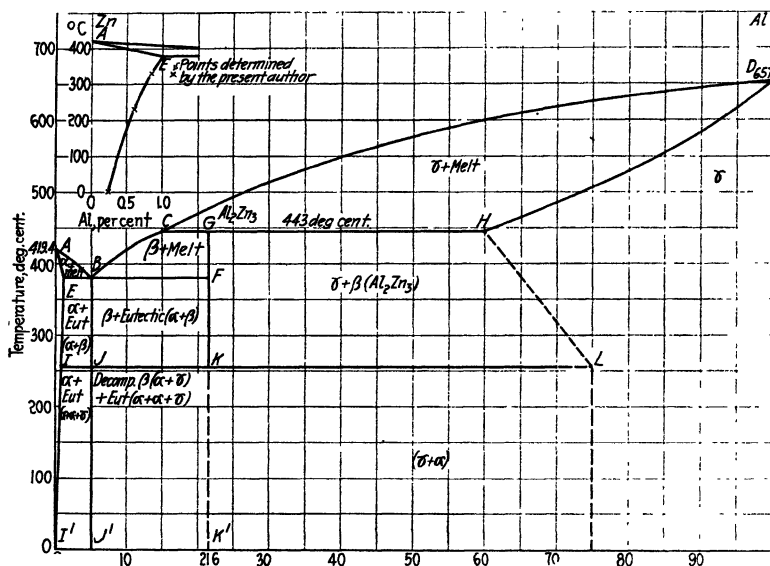
FIG. 5.—ALUMINUM-ZINC EQUILIBRIUM DIAGRAM; BASED ON DATA OF ROSENHAIN AND
ARCHBUTT, BAUER AND VOGEL, PEIRCE.

To determine this curve more precisely, microspecimens containing 0.1, 0.25, 0.50, 0.75 and 1.00 per cent. aluminum were prepared and suitably annealed. Conductivity strips of these concentrations, including also 2.0 per cent. aluminum, were tested in the cold-rolled state and also after suitable anneals. The castings from which these strips were rolled were homogenized before rolling by a 24-hr. anneal at 220° C. Quenching of both microspecimens and conductivity strips was accomplished in an iron-tube furnace with water connections, which permitted almost instantaneous flooding of the tube with a large stream of water in a manner similar to that used by Rosenhain.[22]

[21] Bauer and Vogel: *Mit. Konig. Materialprufungsant* (1915) **33**, 146–198.
[22] Rosenhain: *Jnl.* Iron and Steel Inst. (No. I, 1908).

It was expected from the information already available that the alloys containing about 1.0 per cent. aluminum and above would show two constituents, the primary one α being a solid solution of aluminum in zinc and the secondary one the eutectic between the compound $AlZn_2$ and the solid solution α. It was also expected that at some concentration below 1.0 per cent., depending on the temperature, the secondary constituent would disappear.

In the unannealed material, eutectic is unmistakably present in those specimens containing 0.50, 0.75 and 1.00 per cent. aluminum and unquestionably absent in the 0.10 per cent. aluminum alloy (Figs. 32 and 33). In a specimen of the alloy containing 0.25 per cent. aluminum annealed 40 hr. at 240° C. followed by quenching, there is no trace of eutectic (Fig. 34), while a slight decrease in the amount present is noticeable in the case of the alloy containing 0.5 per cent. aluminum after similar treatment. Quenching from 300° C. after 36 hr. annealing causes a complete disappearance of any areas that can be positively identified as eutectic in the 0.5 per cent. aluminum specimen and a marked diminution in the amount present in the 0.75 per cent. aluminum alloy (Figs. 35, 36, and 37).

Fig. 15 shows the conductivity curves of this series of alloys after the following treatments:

CURVE MARKED	TREATMENT
1...................	cold rolled without subsequent annealing
2..................	cold rolled and annealed ½ hr. at 150° C.
3..................	cold rolled annealed 36 hr. at 335° C., and quenched
4...................	cold rolled, annealed 44 hr. at 230° C. and quenched

These curves check with extraordinary consistency an earlier set of curves made on a similar series of alloys. The latter were replaced by the present set in order to take advantage of a more effective method of quenching.

Curves 3 and 4 give clear indications of the probable solid solubility limits at the respective temperatures of quenching; viz., 0.85 per cent. aluminum at 335° C. and 0.60 per cent. aluminum at 230° C. These are the two principal concentrations used in locating the solid solubility curve, aluminium in zinc, in Fig. 5. Curves 1 and 2, obtained on cold-rolled specimens and on specimens recrystallized by annealing at 150° C. for ½ hr. are difficult to correlate. However, it seems safe to conclude that the solid solubility at room temperature is not more than 0.25 per cent. aluminum.

The microscopic results are confirmatory of the indications of the conductivity observations that the solid solubility of aluminum in zinc varies with the temperature from about 0.85 per cent. aluminum at 335° to not more than 0.25 per cent. at room temperature.

NICKEL-ZINC

The work heretofore done on the nickel-zinc system is thoroughly reviewed in a paper by Victor Tafel.[23] Fig. 6 shows the equilibrium relations existing between the compound $NiZn_3$ and pure zinc. A eutectic between $NiZn_3$ and zinc is indicated by the thermal results of Heycock and Neville, who find a slight lowering of the freezing point of zinc by the addition of nickel.

The present writer has found through microscopic examination that primary crystallites of $NiZn_3$ are present in an alloy containing 0.25 per cent. nickel, Fig. 39. None could be detected, however, in an alloy containing 0.12 per cent. nickel. The eutectic point must therefore lie at a concentration between 0.12 and 0.25 per cent. nickel.
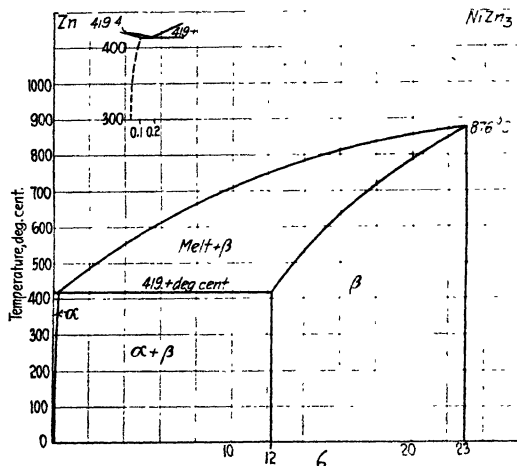
FIG. 6.—NICKEL-ZINC EQUILIBRIUM DIAGRAM; BASED ON DATA OF TAFEL AND PEIRCE.

The eutectic between zinc and $NiZn_3$ appears to be structurally associated with the lead-zinc eutectic, perhaps as a ternary eutectic. This conclusion is based on the fact that the $NiZn_3$-Zn eutectic is distributed in exactly the same manner as the lead eutectic (Figs. 19 and 38) and that the latter loses its identity when the former is present.

That there is an appreciable range of solid solubility is indicated by both microscopic and conductivity methods. As little as 0.034 per cent. nickel gives rise to the appearance of eutectic, and annealing this alloy 100 hr. at 300° C. followed by drastic quenching does not result in an homogeneous structure but does result in a redistribution of the $NiZn_3$. Fig. 40 shows an alloy containing 0.12 per cent. nickel after the same treatment. This indicates greater solid solubility of nickel in zinc at

[23] Tafel: *Metallurgie* (1908) **14**, 413–430.

300° C. than at room temperature. At the same time it would appear that the limit of solid solubility at room temperature is less than 0.034 per cent.

The conductivity curves of both cold-rolled strips and strips recrystallized by annealing ½ hr. at 150° C. have well-defined inflections at about 0.1 per cent. nickel. These curves are based on measurements of strips containing 0.034, 0.058, 0.12, 0.25, 0.48 and 0.84 per cent. nickel, Fig. 17. In view of this conflicting evidence, the writer does not feel warranted in any more definite conclusion than that the solid solubility of nickel in zinc varies, increasing with the temperature, and probably never exceeding 0.1 per cent. nickel.

MANGANESE-ZINC

Parravano[24] was the first to offer a manganese-zinc equilibrium diagram. Two compounds, $MnZn_3$ and $MnZn_7$, were described, the latter entering into an eutectic relationship with zinc at a temperature of 418° C. and a concentration of about 1 per cent. manganese. The question of solid solubility of manganese in zinc was not seriously raised.

Siebe[25] observed eutectic crystallization at 416° C., but made no accurate determination of the eutectic concentration or of the concentration of the zinc-rich constituent. The other eutectic constituent is described as the end member of a series of solid solutions extending up to at least 50 per cent. manganese (the highest manganese concentration investigated) and not a zinc-rich compound, $MnZn_7$, as specified by Parravano.

Gieren[26] locates the eutectic concentration at approximately 2 per cent. manganese by the method of thermal analysis. At 0.5 per cent. manganese, he finds no microscopic evidence of the eutectic constituent, but at 1 per cent. this constituent appears. The thermal treatment of these alloys is not specified. Gieren confirms Parravano in locating a zinc-rich compound, the composition of which, however, is not specified. There is also a lack of agreement concerning equilibrium relations at higher manganese concentrations.

The present investigation indicates that the eutectic concentration lies close to 0.9 per cent., as an alloy containing 0.8 per cent. manganese showed no primary $MnZn_7$ while one containing 0.9 per cent. manganese did. Fig. 41 shows the eutectic in a sample containing 0.8 per cent. manganese; Fig. 42 shows the primary crystallites of $MnZn_7$ in an alloy containing 0.9 per cent. manganese. Annealing makes small amounts of eutectic easily visible through a reabsorption and reprecipitation de-

[24] Parravano: *Gazz. Chim. Ital.* (1914) **45**, [1] 1–6.
[25] Siebe: *Zeits. Anorg. Chem.* (1919) **108**, 171.
[26] Gieren: *Zeits. Metallkunde* (1919) **11**, 14.

scribed later. The eutectic is readily visible in a specimen containing 0.2 per cent. manganese after annealing 24 hr. at 250° C., Fig. 43, but it does not appear in the alloy containing 0.1 per cent. manganese treated similarly. This interesting effect of heat treatment is further shown in Figs. 44 and 45. The large areas of eutectic $MnZn_7$ seen in Fig. 44 have been partly reabsorbed on annealing 5 hr. at 40° C. and reprecipitation of $MnZn_7$ in smaller, more widely scattered, particles has commenced in spite of quenching in water.

It is worthy of note that in alloys of zinc containing iron and in those containing manganese crystals occur whose sections have the same shape. This shape, a hollow parallelogram, is illustrated in Fig. 42. In view of the close relation iron and manganese bear to each other in the periodic system, this would seem entirely consistent with the evidence tending
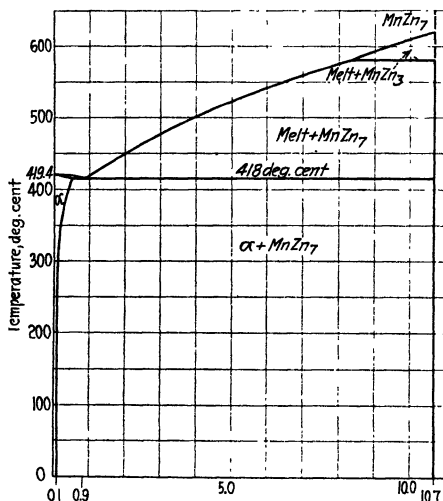
FIG. 7.—MANGANESE-ZINC EQUILIBRIUM DIAGRAM; BASED ON DATA OF PARRAVANO AND PEIRCE.

to show that these crystals are the homologous compounds $FeZn_7$ and $MnZn_7$.

The conductivity curves of the cold-rolled and the recrystallized strips, Fig. 13, show sharp inflections at about 0.1 per cent. manganese. Annealing ½ hr. at 400° C. and cooling in air, which in the case of thin strips is rapid cooling, shifts this point of inflection to about 0.35 per cent. manganese, indicating an increasing solubility of $MnZn_7$ in zinc with rising temperature. This is consistent with the observation noted above, that on annealing redistribution of the $MnZn_7$ takes place.

That actual reabsorption of the $MnZn_7$ does occur on annealing and that by quenching a certain amount above the normal equilibrium amount can be retained in solid solution was shown as follows. A strip

containing 0.33 per cent. manganese was tested and found to have a conductivity of 14.92 (mhos $\times$ 10⁴). After annealing at 325° C. and cooling in air, the conductivity was 13.26. Reannealing and cooling in the furnace from the same temperature raised the conductivity to 14.84. Annealing at 400° and cooling in air lowered the conductivity to 11.20, and reannealing at 300° and cooling in air raised it again to 13.28.

Fig. 7 represents closely the equilibrium relations of this system though it was not found possible to quench specimens effectively enough to preserve entirely at room temperature the equilibrium conditions obtaining at higher temperatures. The slope[27] of the solid solubility curve is, therefore, more or less speculative. From a practical point of view, however, the structure of the alloys as it is affected by thermal treatments has been fairly well determined.

COBALT-ZINC

Lewkonja[28] and Ducilliez[29] have found evidence of a compound $CoZn_4$ which is said to form a eutectic with zinc at 413° C. and at a concentration of 0.5 per cent. cobalt. Lewkonja's statement that the eutectic concentration is only 0.5 per cent. cobalt while the lowering of the freezing point is over 6° C., and the writer's conclusion that the eutectic concentration is even lower than this led to a redetermination of the eutectic temperature.

Freezing-point curves of alloys containing 0.5 and 0.9 per cent. cobalt were taken, using a platinum platinum-rhodium thermocouple and potentiometer checked before and after the experiments with C. P. zinc. One arrest only could be observed; this occurred at a fraction of a degree below the freezing point of pure zinc. No arrest at 413° C., as recorded by Lewkonja, was found and the consistency of his results with several alloys is difficult to explain unless some impurity was present in considerable amounts in his alloys.

In the microscopic examination of these alloys, primary crystallites of this compound were identified in specimens containing as low as 0.03 per cent. cobalt, which indicates that both the eutectic concentration and the solid solubility are very low. Annealing a specimen containing 0.03 per cent. cobalt 48 hr. at 300° C. does not cause a complete absorption of the $CoZn_4$ but causes a slight redistribution, new areas of extremely fine particles becoming visible. From this, it is evident that cobalt dissolves in zinc to a very slight extent, the limit of solubility being higher at elevated temperatures, for the redistribution of the $CoZn_4$ must have resulted from absorption on heating and reprecipitation on cooling. Fig. 46 shows crystallites of $CoZn_4$ in an alloy containing 0.35 per cent. cobalt,

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[27] This system was investigated before the development of the quenching furnace employed in the aluminum-zinc system.

[28] Lewkonja: *Zeits. Anorg. Chem.* (1908) **59**, 293.

[29] Ducilliez: *Bull.* Soc. Chem. **9**, 1017.
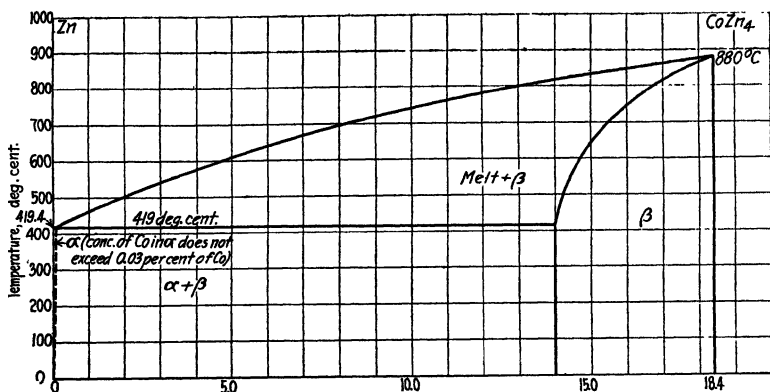
FIG. 8.—COBALT-ZINC EQUILIBRIUM DIAGRAM; BASED ON DATA OF LEWKONJA AND PEIRCE.
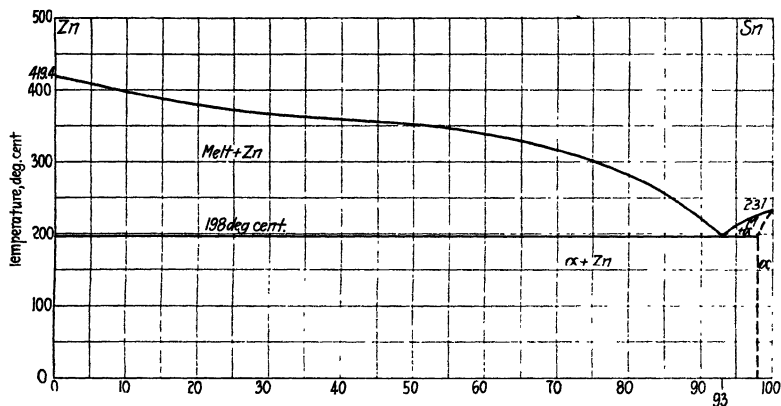
FIG. 9.—TIN-ZINC EQUILIBRIUM DIAGRAM; BASED ON DATA OF HEYCOCK AND NEVILLE MAZZOTTO, ARNEMANN, PEIRCE.
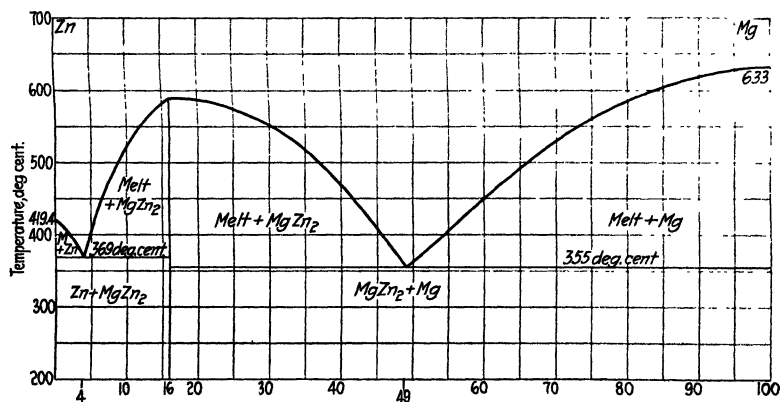
FIG. 10.—MAGNESIUM-ZINC EQUILIBRIUM DIAGRAM; BASED ON DATA OF EGER AND PEIRCE.

and Fig. 47 shows the reprecipitated $CoZn_4$ in an alloy containing 0.04 per cent. cobalt.

The conductivity curve, Fig. 18, shows a slight irregularity below 0.05 per cent. cobalt, which is doubtless connected with the slight solid solubility that exists. The equilibrium relations are shown in Fig. 8.

TIN-ZINC

A number of investigators, Heycock and Neville, Mazzotto,[30] Arnemann,[31] and Lorenz and Plumbridge[32] have dealt with this system; Fig. 9 shows the results of these various investigations. No indication of solid solubility of tin in zinc has been noted and the present investigation has shown that there is no appreciable solid solubility on the zinc side.

Microscopic examination of a cast specimen containing 0.1 per cent. tin shows clearly an intercrystalline film of tin-zinc eutectic, Fig. 48. Fig. 49 shows a specimen containing 0.5 per cent. tin.

Alloys containing as little as 0.05 per cent. tin are extremely brittle, fracturing between the crystals through the eutectic films. This is evidence of the existence of such films at a concentration as low as 0.05 per cent. tin. The primary tin visible in the alloy containing 0.1 per cent. tin cannot be removed by annealing 60 hr. at 190° C. Owing to the brittleness of the alloys, no rolled strips for conductivity tests were prepared.

MAGNESIUM-ZINC

The work of George Eger,[33] which reviews the earlier work of Grube,[34] Boudouard,[35] and Bruni and Sandonnini[36] is the most recent on this system. Fig. 10 embodies the results of this previous work. The absence of solid solubility has been confirmed by original work.

Magnesium forms the compound $MgZn_2$ with zinc, and this compound forms a simple eutectic system with zinc. The compound contains 84 per cent. zinc and the eutectic is 75 per cent. zinc, 25 per cent. $MgZn_2$. It follows that 0.01 per cent. magnesium gives rise to 0.25 per cent. eutectic and that very small amounts of eutectic can, therefore, be recognized microscopically.

Figs. 50 and 51 show the appearance of the eutectic in specimens containing respectively 0.06 and 0.12 per cent. of magnesium, while Fig. 52 shows the structure of the eutectic at high magnification in an alloy

[30] Mazzotto: *Intnatl. Zeits. Metallographie,* **4**, 273–96.

[31] *Op. cit.*

[32] Lorentz and Plumbridge: *Zeits. Anorg. Chem.* (1913) **83**, 228–242.

[33] Eger: *Intnatl. Zeits. Metallographie,* **4**, 29–127.

[34] Grube: *Zeits. Anorg. Chem.* (1906) **49**, 77.

[35] Boudouard: *Bull. Soc. Enc.* (1904).

[36] Bruni, Sandonnini and Quercigh: *Zeits. Anorg. Chem.* (1910) **68**, 78–9.

of 1.11 per cent. magnesium. Annealing 60 hr. at 300° C. caused no change in the amount or distribution of the eutectic in any of these specimens. The conductivity curve of the alloys, as shown in Fig. 16, indicates an absence of solid solubility.
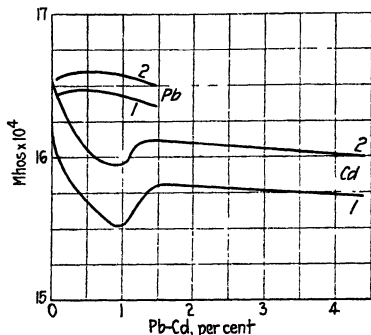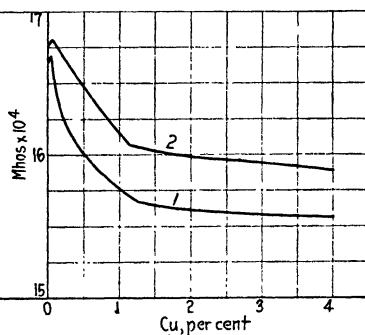
FIG. 11. FIG. 12.

FIG. 11.—CONDUCTIVITY OF LEAD-ZINC AND CADMIUM-ZINC ALLOYS; 1, COLD-ROLLED; 2, RECRYSTALLIZED.

FIG. 12.—CONDUCTIVITY OF COPPER-ZINC ALLOYS; 1, COLD-ROLLED; 2, RECRYSTALLIZED.
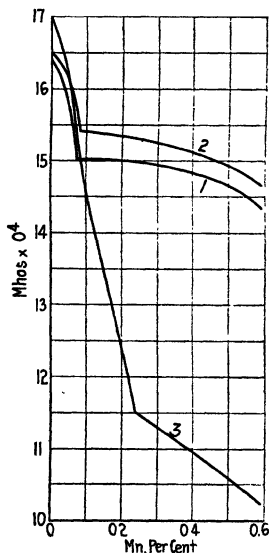
FIG. 13. FIG. 14.

FIG. 13.—CONDUCTIVITY OF MANGANESE-ZINC ALLOYS; 1, COLD-ROLLED; 2, RECRYSTALLIZED; 3, ANNEALED ½ HR. AT 400° C. AND AIR COOLED.
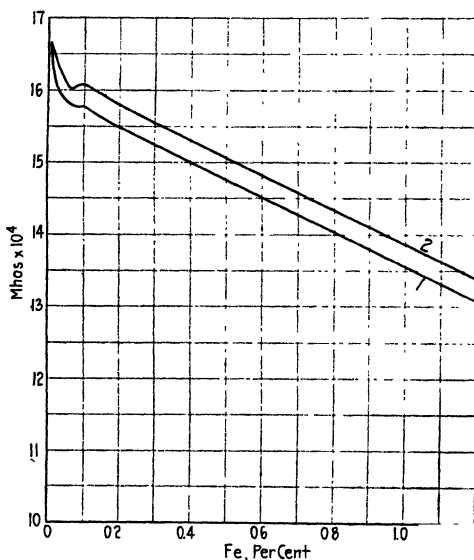
FIG. 14.—CONDUCTIVITY OF IRON-ZINC ALLOYS; 1, COLD-ROLLED; 2, RECRYSTALLIZED.

ACKNOWLEDGMENT

The writer wishes to acknowledge his indebtedness to Dr. C. H. Mathewson for his helpful advice and criticism, throughout the course

of this work, to Mr. T. A. Hatch, formerly of this laboratory, who did a large amount of preliminary work in preparing for this series of investi-
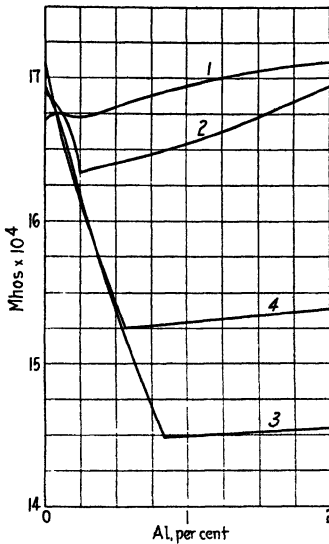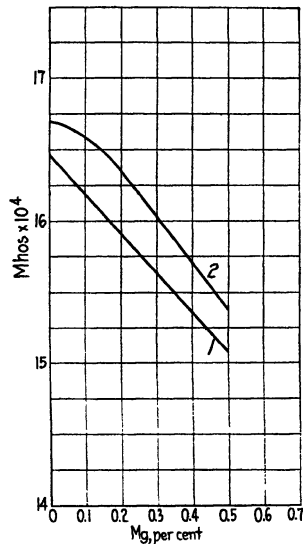
FIG. 15.   FIG. 16.

FIG. 15.—CONDUCTIVITY OF ALUMINUM-ZINC ALLOYS; 1, COLD-ROLLED; 2, COLD-ROLLED AND ANNEALED ½ HR. AT 150° C.; 3, COLD-ROLLED AND ANNEALED 36 HR. AT 335° C. AND QUENCHED; 4, COLD-ROLLED AND ANNEALED 44 HR. AT 230° C. AND QUENCHED.

FIG. 16.—CONDUCTIVITY OF MAGNESIUM-ZINC ALLOYS; 1, COLD-ROLLED; 2, RECRYSTALLIZED.
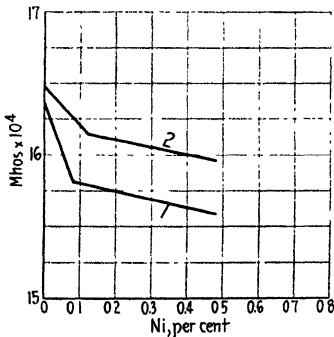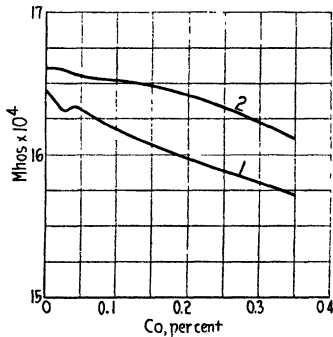
FIG. 17.   FIG. 18.

FIG. 17.—CONDUCTIVITY OF NICKEL-ZINC ALLOYS; 1, COLD-ROLLED; 2, RECRYSTALLIZED.

FIG. 18.—CONDUCTIVITY OF COBALT-ZINC ALLOYS; 1, COLD-ROLLED; 2, RECRYSTALLIZED.

gations and who carried out a part of the work reported here on the iron-zinc system, and to Miss Marianne Goettsch for her assistance in carrying out portions of the work.

FIG. 19.—DISTRIBUTION OF LEAD IN CAST HORSE–HEAD ZINC CONTAINING 0.05 PER CENT. PB. × 20.
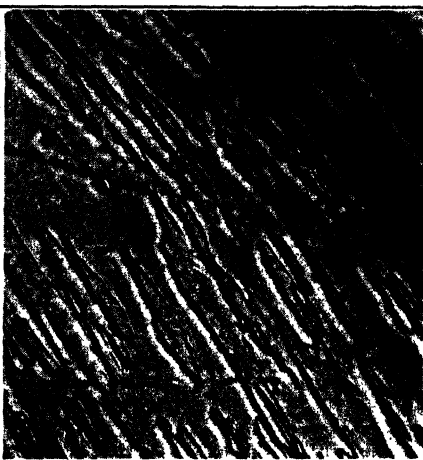
FIG. 20.—GLOBULES OF LEAD AT HIGHER MAGNIFICATION. × 500.
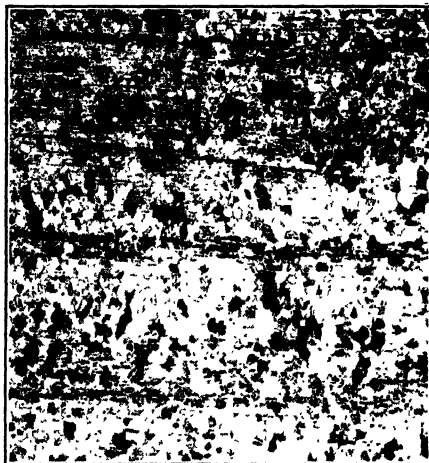
FIG. 21.—DISTRIBUTION OF LEAD IN STREAKS IN ROLLED ZINC CONTAINING 0 25 PER CENT. PB. × 50.

FIG. 22.—ALLOY CONTAINING 1.25 PER CENT. CD, SHOWING ZONAL STRUCTURE AND EUTECTIC FILMS. × 150. ETCHED WITH ALCOHOLIC HCL.

SPECIMENS, NOT OTHERWISE NOTED, WERE ETCHED WITH CHROMIC ACID
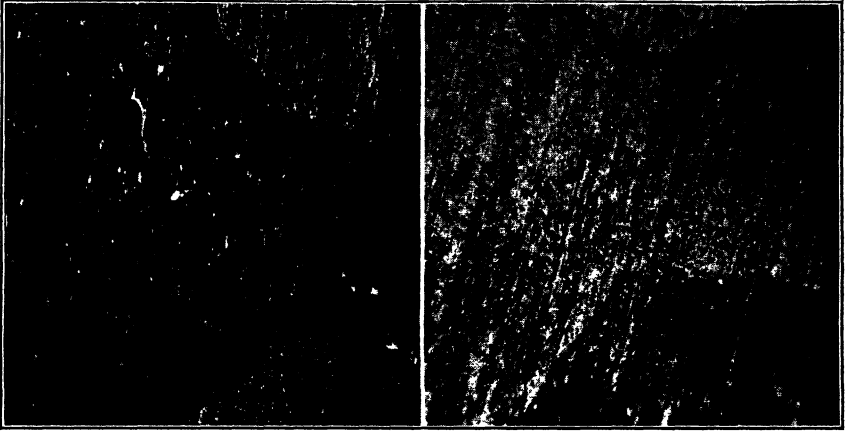
FIG. 23.—ALLOY CONTAINING 1.25 PER CENT. CD, SHOWING REMOVAL OF ZONAL STRUCTURE BY ANNEALING 10 HR. AT 250° C., ALSO COALESCED CD. × 150. ETCHED WITH ALCOHOLIC HCL.

FIG. 24.—HOMOGENIZED ALLOY CONTAINING 1.5 PER CENT. CD, ANNEALED 5 WK. AT 250° C. ×!500. ETCHED WITH ALCOHOLIC HCL.
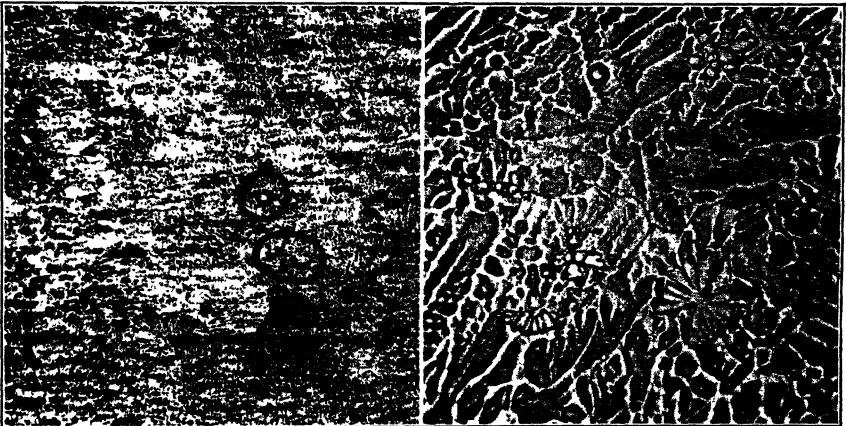
FIG. 25.—CRYSTALS OF FEZN7 IN ALLOY CONTAINING 0.03 PER CENT. FE. × 250.

FIG. 26.—ZONAL STRUCTURE AND PRIMARY IN ALLOY CONTAINING 1.81 PER CENT. CU. × 75.

SPECIMENS, NOT OTHERWISE NOTED, WERE ETCHED WITH CHROMIC ACID.
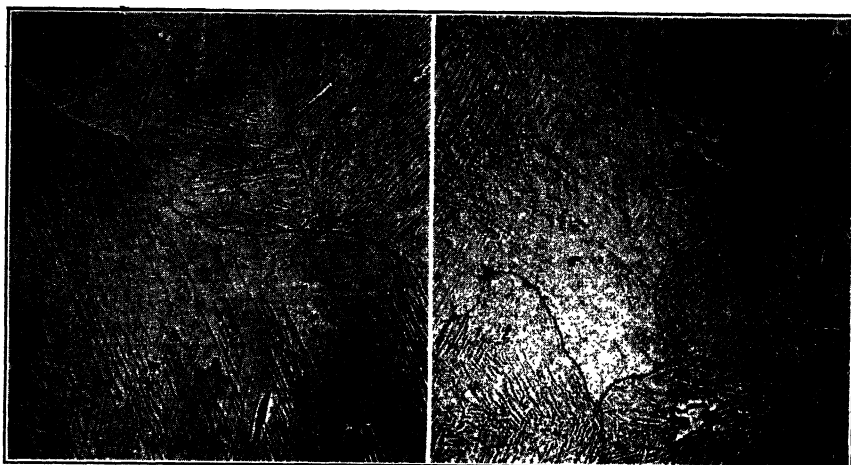
FIG. 27.—QUENCHED ALLOY CONTAINING 1.81 PER CENT. CU PREVIOUSLY HOMOGENIZED BY ANNEALING 36 HR. AT 400° C.   × 75.

FIG. 28.—PRIMARY CRYSTALS REMAINING IN QUENCHED ALLOY CONTAINING 1.95 PER CENT. CU PREVIOUSLY ANNEALED AS IN FIG. 27.   × 75.
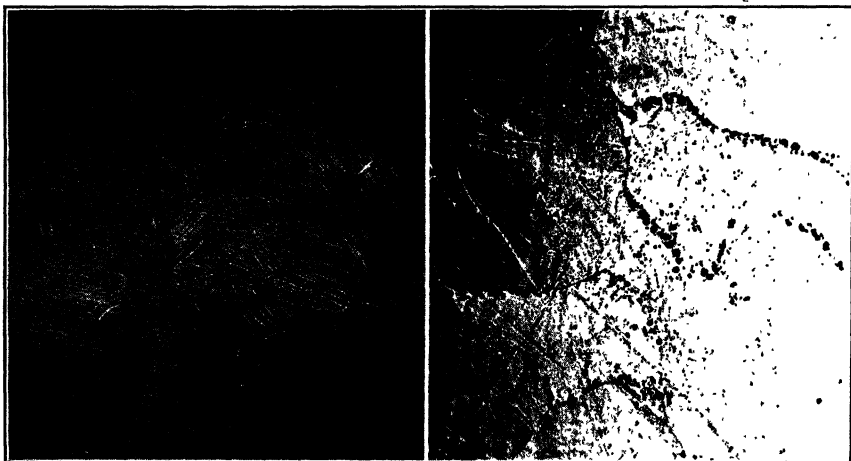
FIG. 29.—ALLOY CONTAINING 1.5 PER CENT. CU HOMOGENIZED AND QUENCHED AT 400° C.   × 50.

FIG. 30.—SAME SPECIMENS AS IN FIG. 29, QUENCHED AFTER FURTHER ANNEALING 5 HR. AT 300° C.   × 200.

ALL SPECIMENS WERE ETCHED WITH CHROMIC ACID.
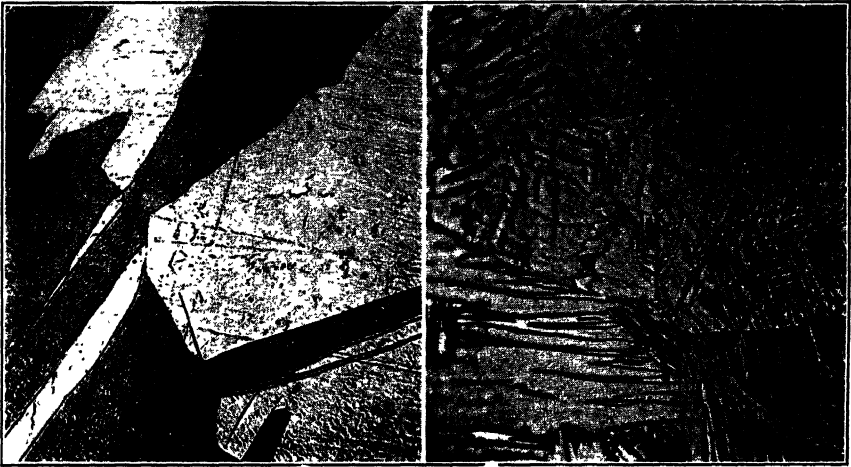
FIG. 31.—SAME SPECIMENS AS IN FIG. 30, QUENCHED AFTER FURTHER ANNEALING 18 HR. AT 350° C.   × 200.

FIG. 32.—ALLOY CONTAINING 0.10 PER CENT. AL UNANNEALED.   × 200.
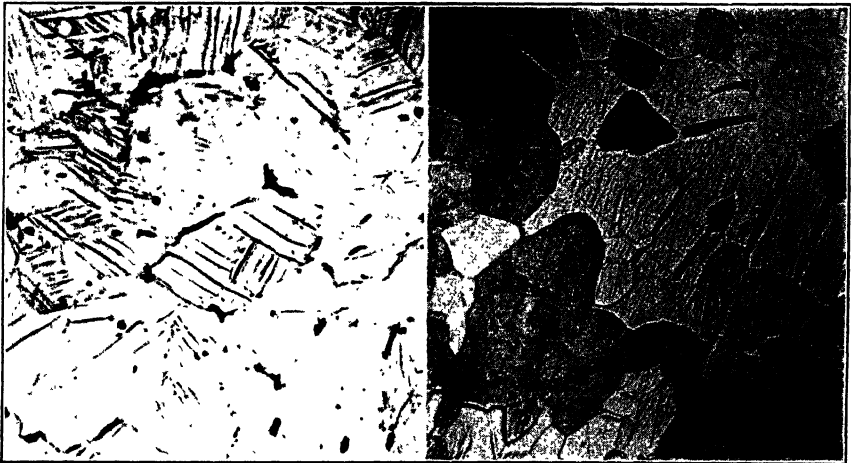
FIG. 33.—ALLOY CONTAINING 0.50 PER CENT. AL UNANNEALED, SHOWING EUTECTIC.   × 200.

FIG. 34.—ALLOY CONTAINING 0.25 PER CENT. AL ANNEALED 40 HR. AND QUENCHED AT 240° C., SHOWING ABSENCE OF EUTECTIC.   × 200.

ALL SPECIMENS WERE ETCHED WITH CHROMIC ACID.
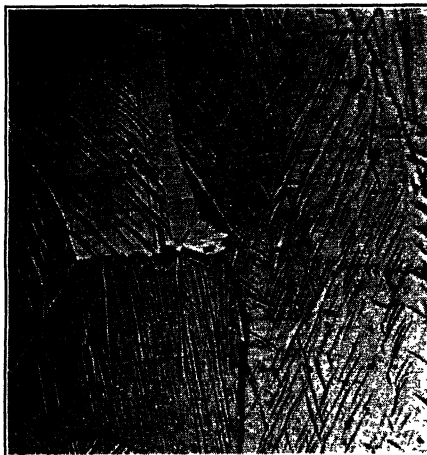
FIG. 35.—ALLOY CONTAINING 0.50 PER CENT. AL ANNEALED 36 HR. AT 300° C.; EUTECTIC HAS DISAPPEARED ALMOST ENTIRELY. × 200.
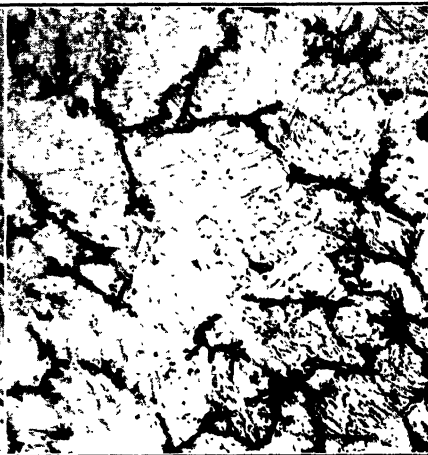
FIG. 36.—ALLOY CONTAINING 0.75 PER CENT. AL UNANNEALED. × 200.

FIG. 37.—ALLOY CONTAINING 0.75 PER CENT. AL ANNEALED 36 HR. AND QUENCHED AT 300° C.; EUTECTIC DIMINISHED IN QUANTITY. × 200.

FIG. 38.—ALLOY CONTAINING 0.12 PER CENT. NI, SHOWING EUTECTIC. ×150.

ALL SPECIMENS WERE ETCHED WITH CHROMIC ACID.

FIG. 39.—ALLOY CONTAINING 0.25 PER CENT. NI, SHOWING PRIMARY NiZn₃ AND EUTECTIC. × 150.

FIG. 40.—ALLOY CONTAINING 0.12 PER CENT. NI, ANNEALED 100 HR. AT 300° C. AND QUENCHED, SHOWING REPRECIPITATED NiZn₃. × 350.

FIG. 41.—ALLOY CONTAINING 8.0 PER CENT. MN, SHOWING EUTECTIC. × 150.
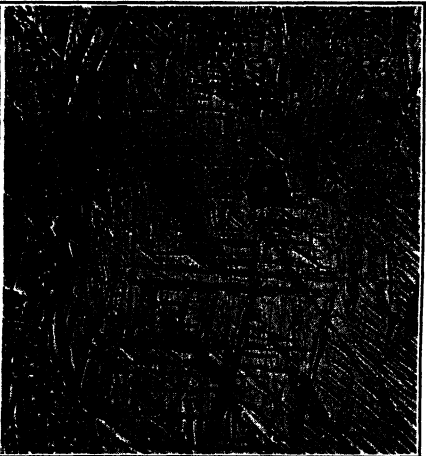
FIG. 42.—ALLOY CONTAINING 0.9 PER CENT. MN, SHOWING PRIMARY MnZn₇ AND EUTECTIC. × 250.

ALL SPECIMENS WERE ETCHED WITH CHROMIC ACID.
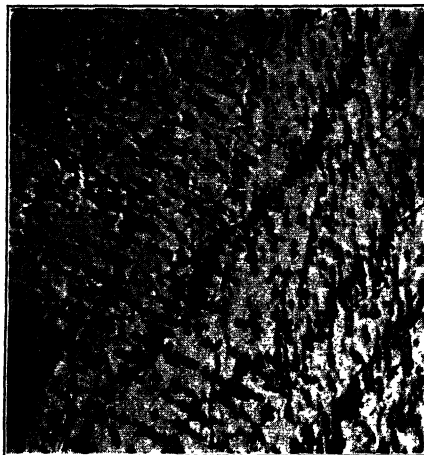
FIG. 43.—ALLOY CONTAINING 0.2 PER CENT. MN ANNEALED 24 HR. AT 250° C. × 500.

FIG. 44.—ALLOY CONTAINING 0.5 PER CENT. MN, SHOWING LARGE AREAS OF EUTECTIC AT LOW MAGNIFICATION. × 150.

FIG. 45.—SAME SPECIMEN AS IS SHOWN IN FIG. 44, AFTER ANNEALING 5 HR. AT 400° C., SHOWING REDISTRIBUTION OF EUTECTIC MNZN7. × 500.
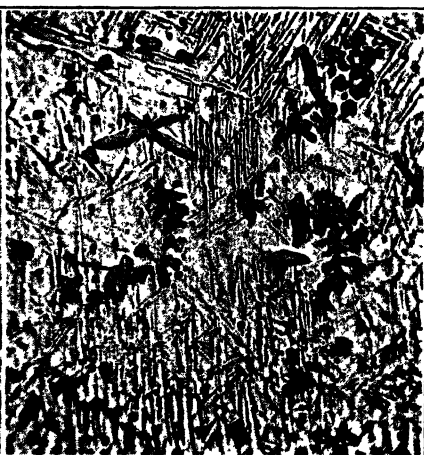
FIG. 46.—ALLOY CONTAINING 0.35 PER CENT. CO, SHOWING PRIMARY COZN4. × 150.

ALL SPECIMENS WERE ETCHED WITH CHROMIC ACID.
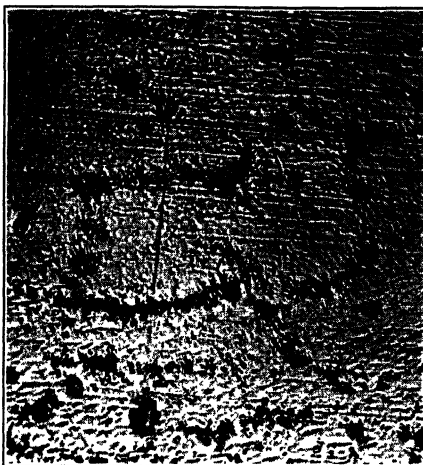
FIG. 47.—ALLOY CONTAINING 0.04 PER CENT. CO ANNEALED 48 HR. AT 300° C., SHOWING REPRECIPITATED CoZn₄. × 350.

FIG. 48.—ALLOY CONTAINING 0.10 PER CENT. SN, SHOWING INTERCRYSTALLINE EUTECTIC FILM. × 50.

FIG. 49 —ALLOY CONTAINING 0.5 PER CENT. SN, SHOWING EUTECTIC. × 50.
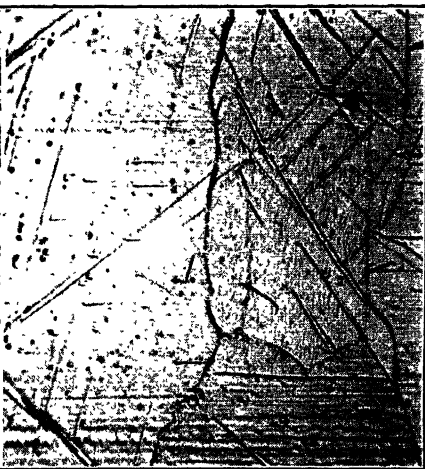
FIG. 50.—ALLOY CONTAINING 0.06 PER CENT. MG, SHOWING EUTECTIC. × 150.

ALL SPECIMENS ETCHED WITH CHROMIC ACID.

FIG. 51.—ALLOY CONTAINING 0.12 PER CENT. MG, SHOWING EUTECTIC. × 150.

FIG. 52.—ALLOY CONTAINING 1.11 PER CENT. MG, SHOWING EUTECTIC AT HIGH MAGNIFICATION. × 750.

ALL SPECIMENS ETCHED WITH CHROMIC ACID.

DISCUSSION

GEORGE K. BURGESS,* Washington, D. C.—What is the reason for the increase in the conductivity of zinc when a small amount of copper is added.

WILLIS M. PEIRCE.—I cannot say. When we noticed that there was a slight increase in conductivity with the first slight additions of copper, we checked the results and found that, at least in the alloys with which we worked, that there is no doubt that a slight increase does occur. It is entirely possible that there is a slight deoxidizing effect. Although we have not been able to prove whether or not zinc dissolves its oxide; a slight deoxidizing effect or simply a lowering of the amount of mechanically included oxides may account for the increased conductivity.

D. H. INGALL, Wednesbury, Staffs., Eng. (written discussion).— With regard to the lead-zinc system, I cannot agree with Mr. Peirce's microscopic deductions concerning the distribution of lead as small globules between the growing branches of the zinc crystals. In Fig. 19, the large polygonal white grain shown was formed during solidification; and if the lead was thrown out to the interstices of the growing branches of zinc crystals, it would appear in the grain arranged in dendritic form. The photomicrograph shows that it appears at the boundaries of small polygonal areas and the small polygonal areas are the result of an allo-

* Chief, Division of Metallurgy, Bureau of Standards.

tropic change in the zinc after solidification and that possibly the solubility of lead in zinc is less in the lower temperature allotrope so that it is thrown out at the small grain boundaries formed when the allotropic change takes place.

J. L. HAUGHTON,* Woolwich, Eng. (written discussion).—In general, the study of limited solid solubility in metallic systems has been much neglected, doubtless because of the length of time and the trouble involved in securing complete equilibrium and the difficulty of obtaining an etching reagent that will reveal minute traces of a second constituent in a nearly homogeneous material. The softer metals, tin, lead, zinc, etc., are particularly difficult from this point of view, and the frequent cross hatching found in zinc does not render the matter any easier. In some cases, Fig. 24, for example, it would be difficult to determine traces of a second constituent, so that the check afforded by electrical conductivity measurements is especially valuable.

The author in every case gives values for the conductivity of the material in the cold-rolled condition, as well as in the annealed state. It is dangerous to draw any conclusions as to the equilibrium condition from the former, as the conductivity will depend, to a considerable extent, on the amount of rolling, etc. Apart from this, however, some interesting results appear on comparing the conductivity of rolled and annealed samples. Mr. Peirce points out that in every case, except that of the aluminum-zinc alloys, the conductivity of the annealed material is higher than that of the rolled material. This is not the only anomaly in the aluminum-zinc results; in the rolled material, the addition to zinc of aluminum (at least up to 2 per cent.) raises the conductivity practically continuously. This, surely, is an isolated exception to the rule that the addition of a second metal lowers the conductivity when solution takes place. The same is partly true in the case of the partly annealed metal (Fig. 15, curve 2) where the conductivity, after the first drop, rises rapidly and at 2 per cent. of aluminum is as high as that of pure zinc. Rolled aluminum-zinc alloys appear to be anomalous in many ways. The writer, in collaboration with Doctor Rosenhain and Miss Bingham, showed recently that, in the case of certain aluminum-zinc alloys containing a little copper, the heat of solution (in acid) of rolled material was less than that of annealed material, which is contrary to what is known for all other rolled material. In conclusion, the writer would like to express his appreciation of the paper, and his gratification that the author's very accurate work confirms the results obtained, on the copper-zinc system, by Miss Bingham and himself.

* Research Dept., Royal Arsenal.

K. E. Bingham,[*] Teddington, Middlesex, Eng. (written discussion).— The author is to be commended for his use of electrical conductivity measurements in conjunction with microscopic analysis in investigating the equilibrium conditions of the binary zinc-base alloys which he has studied.

With reference to the solubility of cadmium in zinc, while long annealings are probably necessary for the attainment of complete equilibrium, unless a high-power microscope were used it would be quite possible to miss the first appearance of the cadmium. It may be useful to refer to the investigation by the writer of the solubility limits of cadmium in zinc to which the author has referred.[37] Fig. 4 in that paper represents an alloy containing 1 per cent. cadmium at a magnification of 1000 diameters. This specimen was annealed for 173 hr. at 150° C., but previous to that treatment it had been annealed at 250° C. for 89 hr. and found to be homogeneous. After an anneal of 89 hr. at 250° C. the writer found only a trace of cadmium in an alloy containing 1.5 per cent. of that metal, which agrees with the author's results. The writer cannot confirm the author's statement that "in chill castings about 1.0 per cent. of cadmium is held in solid solution by zinc." Her experience has been that a minute trace of cadmium can be distinguished in the chill-cast state in an alloy containing as little as 0.25 per cent. of that metal, while there is a marked increase of the same constituent in the 0.5 per cent. cadmium alloy. This agrees better with the author's opinion as to the necessity of long annealings than his own observations on the chill-cast alloys of zinc with cadmium.

It is gratifying to notice that the author's determination of the solubility of copper in zinc is in reasonably good agreement with that of the writer in collaboration with Doctor Haughton.[38] It may be pointed out, however, that the solubility line given by Haughton and Bingham (see Fig. 10 in the paper quoted) slopes backwards appreciably more than that in the author's diagram, Fig. 4. In this case also the writer found rather less solubility in the chill-cast state than that indicated by the author, and the presence of the ε constituent is shown in Fig. 10 in the paper on Zinc Alloys with Aluminium and Copper, by Rosenhain, Haughton and Bingham,[39] which represents, at a magnification of 1000 diameters, an alloy containing 1.3 per cent. of copper, annealed at 255° C.

Willis M. Peirce (author's reply to discussion).—As stated in the paper, the lead lines occur very frequently as a polygonal network; in nearly as many cases, however, they occur as parallel lines. The arrange-

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[*] The National Physical Laboratory.
[37] See foot-note 15.
[38] Proc. Roy. Soc. London (1921).
[39] Jnl. Inst. Metals (1920) 23, 261.

ment of these lead lines is not inconsistent with the original explanation. While Mr. Ingall's explanation may be correct, it is difficult to accept, without further evidence, the three premises on which the theory must rest, *viz.*: (1) That the existence of an allotropic change in zinc has been definitely established. (2) That the solid solubility of lead in one allotrope is nearly as great as the liquid solubility at the melting point. (This must be assumed as lead up to about 1 per cent. is present as these lead lines.) (3) That either the polygonal structure or the parallel arrangement of the lines is more indicative of separation coincident with an allotropic change than of separation during freezing. Until at least the first two of these assumptions are conclusively demonstrated, the author prefers the simpler explanation.

Doctor Haughton has called attention to several anomalies in the aluminum-zinc conductivity curves, which the author hopes eventually to explain. When the paper was prepared, time for further study was lacking.

The author fears that he was misled by the statement in which Miss Bingham originally reported her experiments on the solid solubility of cadmium in zinc. The fact that the specimens "annealed at 150° C. for several hours" had in fact been annealed a week after being previously homogenized, satisfies the writer that Miss Bingham's results represent equilibrium conditions.

It is quite probable that with sufficiently drastic chilling, small amounts of secondary cadmium will occur in alloys containing less than 1 per cent. cadmium. It is evident, from the conductivity curves, and microscopic examination, however, that even in chill-cast alloys containing 1 per cent. or less of cadmium, the latter is principally in solid solution. The effects on mechanical properties that might be anticipated from this fact are found in practice.

With regard to the presence of ϵ in the 1.5 per cent. copper alloy annealed at 255° C., (Fig. 10 of Miss Bingham's paper to which she refers) attention is called to Fig. 30 of the present paper, in which it is shown that secondary ϵ separates from this alloy upon annealing below 315° C. The appearance of the ϵ in Miss Bingham's photomicrograph is typical of secondary reprecipitated ϵ at high magnification as observed by the writer.

The structure of annealed specimens has no bearing on the determination of the point a (Fig. 4), however. This point, which marks the first appearance of ϵ as a primary constituent, must be determined by examination of unannealed chill castings in which there has been no opportunity for reabsorption of primary ϵ or reprecipitation of secondary ϵ. The point a was determined in this manner as stated in the paper.

The Effect of Impurities on the Oxidation and Swelling of Zinc Aluminum Alloys

H. E. BRAUER* AND W. M. PEIRCE*

PART I

INTRODUCTION

Among the zinc base alloys used for casting in metal moulds, particularly die casting, those alloys containing aluminum usually together with copper, are probably the most widely used. The reason lies in their desirable mechanical properties and the fluidity and absence of hot shortness which facilitate casting.

Objection to the use of these alloys may be raised on account of the rather frequent instances of failure of a type variously described as disintegration, exfoliation and intercrystalline corrosion. In all cases which have come to our knowledge where the conditions of service were known, failure has occurred upon exposure to moisture and heat varying from a hot moist atmosphere to high pressure steam. Moreover, the alloys described in all such cases contain impurities or alloying metals which the present authors have found to have enormous accelerating effects on intercrystalline oxidation in zinc-aluminum alloys.

Thus Williams[1] experimented with an alloy containing a large percentage of tin exposed in Cuba. Bauer[2] reports failure of a boiler plug containing over 1 per cent. of impurities, doubtless lead for the most part, and Rawdon's[3] examinations were made on an alloy containing 0.4 per cent. lead also exposed to the Tropics.

No thorough investigation of the nature and cause of this type of failure has been made heretofore, so far as the authors are aware, although Rawdon correctly classified it as internal oxidation.

We have, therefore, carried out this investigation in order to determine the exact nature of this phenomenon and the effect of other metals present as impurities or intentional additions in accelerating or retarding it. In the appendix we include a considerable amount of data which may be valuable in selecting an alloy combining good physical properties with high resistance to oxidation.

At the time this work was undertaken the best available equilibrium

diagram of the zinc-aluminum system was that published by one of the present authors[4] based on the work of Rosenhain and Archbutt and Bauer and Vogel supplemented by his own investigations. Recently the work of Hanson and Gaylor[5] has come to our attention and their findings in regard to the β constituent have been useful in explaining certain of our results.

PART II

SUMMARY OF RESULTS

(a) *Investigation of pure zinc-aluminum alloys.*

We have found that the type of failure under discussion is due to intercrystalline oxidation which attacks a but not γ.

When the β constituent is decomposed into a and γ, the resulting structure offers an enormous number of secondary grain boundaries in which oxidation can occur.

Expansion measurements show that quenching from above the decomposition temperature of β increases the expansion. This may be due to the fact that some undecomposed β remains and is especially susceptible to oxidation or it may be due to a finer particle size in the decomposed β or to both.

This oxidation occurs at a rapid rate in wet steam. Six months' exposure in dry air at 100° C., however, has produced no attack. Moisture appears, therefore, to be essential. The grain size of zinc-aluminum alloys decreases with increasing aluminum content and in coarse grained alloys (alloys below two per cent. aluminum) the depth of penetration is great though the expansion may be slight.

The severity of intercrystalline oxidation as affecting the physical characteristics of a specimen depends upon the grain size and the presence or absence of impurities, or other metals added in the alloy.

(b) *Investigation of zinc aluminum alloys containing other metals.*

Certain metals when present even in hundredths of a per cent. accelerate the oxidation noticeably. Lead is the most powerful accelerator. Cadmium and tin are nearly as. effective. Iron is somewhat less active. Nickel and manganese when present alone in zinc aluminum alloys are moderately detrimental.

Copper to the extent of 0.5 per cent. is a powerful retarding agent and in any amount up to 5 per cent. is more or less beneficial.

In dense castings of fine grained alloys free from detrimental impurities the rate of penetration is very small and uniform resulting in a slight

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[4] W. M. Peirce—See page 767.
[5] Hanson and Gaylor—Inst. of Metals (1922).

expansion without distortion. The density of the castings, however, is of prime importance, and in die castings the effect due to differences in soundness of the castings of various alloys may equal the effects due directly to the composition.

In zinc-aluminum alloys containing detrimental impurities, the depth of penetration and the expansion are very pronounced and result in great distortion. The percentage of lead and cadmium in any but the highest grade zinc is sufficient to cause this effect and many commercial alloys of this type contain considerable tin which is an accelerator.

While in chill cast alloys and alloys quenched from above 256° C., the amount of expansion increases with increasing aluminum up to 21 per cent., the effect of impurities may entirely overshadow this.

Thus an alloy containing only 0.5 per cent. aluminum together with 7 per cent. tin and 3.75 per cent. copper has expanded as much after 15 days in steam as a 4 per cent. zinc aluminum alloy containing no added metals and corrosion has penetrated deeper. On the other hand, an alloy containing 0.5 per cent. copper and 10 per cent. aluminum expands no more in 15 days than a 0.10 per cent. zinc-aluminum alloy containing no added metal.

The rate of penetration and expansion decreases as the depth of penetration increases in the case of fine grained alloys free from harmful impurities.

It is the belief of the writers that zinc-aluminum-copper alloys, free from appreciable amounts of detrimental impurities, will withstand long exposures to any natural atmosphere with no appreciable loss of strength and with only slight expansion, and that, except in tropical climates, no appreciable expansion will occur.

We feel that the alloy containing 5 per cent. aluminum and 2.5-3 per cent. copper made with a high grade distilled zinc constitutes a very great improvement over many of the zinc base alloys made with impure zinc now in use, and consequently should open up a wider field for zinc base die castings.

While the question of protection of these alloys from oxidation by oil, paint, enamel, etc., has not yet been investigated, it is, of course, clear that a coating which helps to exclude oxygen must offer some protection from the intercrystalline oxidation which causes failure of these alloys.

PART III

METHODS

Exposure to wet steam as a means of testing the stability of zinc-aluminum alloys has been used by others and is without doubt the best means of obtaining rapid corrosion in these alloys. The results of the test led to the conclusion arrived at by the present authors, that the combined

effect of heat and moisture is responsible for the failure of zinc-aluminum alloys.

The test was made in a carefully insulated metal lined tank 2¼ ft. wide, 5½ ft. long and 4 ft. high. Water was retained in the tank to the height of about 1½ ft., and was held at 100° C. by three electrical resistance coils. Exposure samples were suspended in the steam in brackets. A firmly clamped cover prevented steam from escaping to any appreciable extent.

Although the steam test produces fundamentally the same type of failure as exposure in a warm, moist, natural atmosphere, that is, intercrystalline oxidation, nevertheless it is a very drastic test. Thus, while we believe it to be a useful method of determining whether one alloy is superior to another, nevertheless in the light of the present investigation we have come to feel that the temperature of steam is too high to yield results thoroughly indicative of what may be expected to occur upon ordinary atmospheric exposure. This is in harmony with the statement made by Williams (loc. cit.), "We consider that the effect of moist air at 176° F. (80° C.) is the most severe condition which will cause rapid swelling of the alloy and yet not destroy the surface of the wheels."

One practical instance which came under our observation was the case of a die casting made of high grade zinc, containing 12 per cent. aluminum and 3 per cent. copper. One of the writers examined several of these castings which had been exposed out of doors in Eastern Pennsylvania for about two years. The fracture showed only a very thin surface film of oxide and the casting had not swelled appreciably. Fortunately, a number of the same castings were available for a steam test and one day in wet steam produced appreciable swelling and penetration.

We may mention here a matter more fully discussed later. Our tests were made on sound open mould castings, whereas die castings invariably contain entrapped air and offer more free paths for penetration.

On account of these two factors, viz., the accelerated nature of the test and the nature of the castings from which our specimens were taken, we wish to emphasize that at least for the present, the data in this paper must be viewed as comparative and not quantitative in any sense of direct application to the behavior of commercial die castings.

To ascertain whether expansion is due solely to oxidation and to determine whether the presence of moisture is necessary for oxidation, sets of specimens from the first series of alloys were tested both at room temperature and at 95° C. in air, in oil, and in CO_2.

A few specimens were weighed before and after exposure to steam, to determine the approximate magnitude of the oxidation.

The specimens used were all of one type, namely, sections half an inch thick and 1¼ inches in diameter, cut from a cylinder of the same diameter and 15 inches long.

Three duplicate specimens were used for each set of conditions. One of these was fractured and sectioned for examination at an intermediate stage in most of the tests. At least two check specimens, however, are represented in the expansion results throughout.

Expansion in each specimen was measured between the parallel milled ends. Micrometer readings to .0001 inch were taken at two marked points.

Micro-specimens representing nearly every composition and heat treatment, before and after exposure and at one or more stages during exposure, were prepared. Tearing and rounding the edges of the cross-sections was prevented by clamping each specimen between iron strips with soft zinc liners during polishing.

For etching alloys up to 5 per cent. aluminum, a solution of 20 grams of C. P. Cr_2O_3, 1.5 grams of Na_2SO_4 (anhyd.) in 100 cc. water was used. The time was 7 to 12 seconds. Alloys containing from 5 to 50 per cent. aluminum were etched 2 or 3 seconds in the above solution followed by 5 to 8 seconds in weak hydrofluoric acid. These alloys could also be etched in a chromic acid solution similar to the above but containing less Na_2SO_4. High aluminum alloys were etched in hydrofluoric acid.

· The depth of penetration of corrosion and the crystal structure were also observed macroscopically on fractured specimens. Corroded areas were easily distinguishable by their dark appearance.

The 15 in. x $1\frac{1}{4}$ in. cylinders or bars from which the specimens were cut were chill cast in a heavy walled iron mould. Pyrometric control of metal and mould temperature was used to insure uniform casting conditions.

It is evident from the equilibrium diagram that alloys up to about 75 per cent. aluminum undergo a phase change at 256° C. ($\beta \rightarrow a + \gamma$). Thus, in our first series of alloys three bars were cast, one of which was left in the chill cast condition. The other two were homogenized by a forty-hour anneal at a temperature of 350° C. for alloys up to 21 per cent. aluminum and 410° for alloys above the concentration in which there should be no eutectic. One of these bars was then slowly cooled in the furnace to room temperature and the other quenched from 300° C.

In this way we have sought to bring the constitutional factor to bear on these investigations.

Later a check run showed that re-annealing a set of slowly cooled specimens 12 hours at 230° C. caused no variation in the subsequent exposure tests.

In making up the alloys, metals of the analyses given in Table 1 were used.

Analyzed intermediate alloys were used where necessary to obtain easy alloying and accurate control of composition.

For the first experiments, planned from a purely theoretical point of view, a series of alloys were made from C. P. zinc and the purest commercial

TABLE 1

Metals Used in Alloys and Their Analyses

Metal	Designation	% Pb	% Cd	% Fe
C. P. Zinc......................	"A"	Less than 0.005	....	0.007
High grade Distilled Zinc..........	"B"	0.039	....	0.010
Intermediate Zinc (exceptionally pure)......................	"C"	0.070	0.060	0.003
Prime Western Zinc....	"D"	0.610	0.620	0.007
Aluminum............	0.42% Fe	0.44% Si	0.06%	Cu
Copper...................	Best grade	electrolytic		
Nickel....................... .	Best grade	electrolytic		
Manganese.....................	97% Mn.			
Lead ⎫				
Cadmium ⎬ Best grades obtainable				
Tin ⎭				

aluminum available at that time. In this series, the compositions of which are shown by a series of dots at the top of Fig. 5, we endeavored to represent each structural field, as indicated by the diagram, and to cover with especial thoroughness the zinc rich region. The compositions are: C. P. zinc, 0.05, 0.10, 0.25, 0.50, 2.00, 5.00, 10.00, 40.00 and 90.00 per cent. aluminum.

A similar series of alloys including somewhat different concentrations were prepared to obtain the data for curves in Fig. 5, which shows the variation in expansion with aluminum content. The compositions were: 6, 10, 15, 21, 24, 30, 40, 50, 65 per cent. aluminum. These alloys were made and tested in duplicate, as shown by dots in Fig. 5.

TABLE 2

Alloys Tested to Determine the Effect of Added Elements on the Rate of Corrosion of Zinc Aluminum Alloys

No.	% Zn (C. P.)	% Al	Added Metal % of Total
1	90	10	
2	90	10	0.5% Pb
3	90	10	0.2% Cd
4	90	10	0.5% Pb, 0.2% Cd
5	90	10	0.025% Fe
6	90	10	0.5% Su
7	89.5	10	0.5% Cu
8	85.0	10	5.0% Cu
9	85.0	12	3.0% Cu
10	89.5	10	0.5% Mn
11	89.9	10	0.1% Ni
12	87 5	10	2.5% Ni

For the second series of experiments it was planned to determine the effect of impurities occurring in zinc and certain addition metals on the rate of corrosion of alloys. C. P. zinc was used in these experiments also. The following table gives the compositions of the alloys used.

These alloys were tested in the manner previously described but only in the chill cast condition.

In order to determine more directly the effect of impurities in zincs on the corrosion of die casting alloys, each one of five commercial die casting alloys was cast up with the various spelters, "A to D," given in Table 1. The composition of the alloys is given in Table 3 below.

TABLE 3

Alloys Cast up With Various Commercial Zincs

No.	% Zn	% Al	% Cu	% Sn
13	85	12	3.0	
14	85	10	5.0	
15	90.75	0.5	2.75	6.00
16	90	5.0	5.00	
17	89	10.0	1.00	

TABLE 4*

Probable Commercial Alloys

No.	% Zn	% Al	% Cu	% Ni
18	94.5	5.00	0.50	
19	93.5	5.00	1.50	
20	93.4	5.00	1.50	0.10
21	92.0	5.00	3.00	
22	90.0	5.00	5.00	
23	89.0	7.00	4.00	(Ternary Eutectic)
24	92.5	7.00	0.50	
25	91.5	7.00	1.50	
26	91.4	7.00	1.50	0.10
27	90.0	7.00	3.00	
28	88.0	7.00	5.00	
29	89.5	10.00	0.50	
30	88.5	10.00	1.50	
31	88.4	10.00	1.50	0.10
32	87.0	10.00	3.00	
33	85.0	10.00	5.00	
34	85.0	12 00	3.00	

*In addition to the corrosion studies, tensile tests were made on all alloys (cf. Table 5 of Appendix).

Having determined the effect of impurities and various added metals on the rate of corrosion of zinc-aluminum alloys, a new set of compositions designed to include a large number of alloys of probable commercial significance was selected, as represented in Table 4.

PART IV

RESULTS

Discussion of Tests on Binary Alloys Made With C. P. Zinc

(a) Microexamination of sections through exposed specimens has revealed most clearly the nature of the deterioration of these alloys. We have found:

First—That all evidences of corrosion begin at crystal boundaries whenever crystal boundaries can be resolved.

Second—That corrosion proceeds from the surface inwards.

Third—That the corrosion is most severe in chill cast specimens and least severe in slowly cooled specimens.

Fourth—That areas of the β structure whether in quenched or slowly cooled specimens appear to be completely oxidized. This will be more fully discussed shortly in connection with the expansion measurements.

Fifth—Specimens exposed to dry air at 100° C. for 6 months show no attack.

(b) The expansion curves (plotted expansion in inches against time of exposure or in some cases against composition) furnished an index to the amount of metal oxidized. Curves in Figs. 1, 2 and 3 show the effect of 500 hours' exposure to wet steam on three series of alloys, 1 chill cast, 2 quenched and 3 slowly cooled.

Fig. 4 shows the same results plotted to show the effect of composition.

Fig. 5 shows expansion curves superimposed on the equilibrium diagram of Hanson and Gaylor.

(c) A few samples were weighed before and after exposure to steam. The increase in weight due to oxidation indicated that in the case of an impure alloy very badly attacked at least 15 per cent. of the metal has been oxidized.

(d) Our investigation has shown that the α constituent is attacked by intercrystalline corrosion and that the γ constituent is not.

The microscopic evidence would indicate that the β structure, whether quenched or slowly cooled, is completely penetrated. We have not, however, resolved the structure of the unattacked slowly cooled β, which should be a duplex structure of $\alpha+\gamma$, and hence it is impossible from the microscopic evidence to determine the nature of the attack on this constituent

Fig 1- Time Expansion Curves of Cast Specimens in Chill Cast Condition Using C P Zinc

Fig 2- Expansion Curves of Specimens Annealed at 350°C and Quenched from 300°C Using C P Zinc
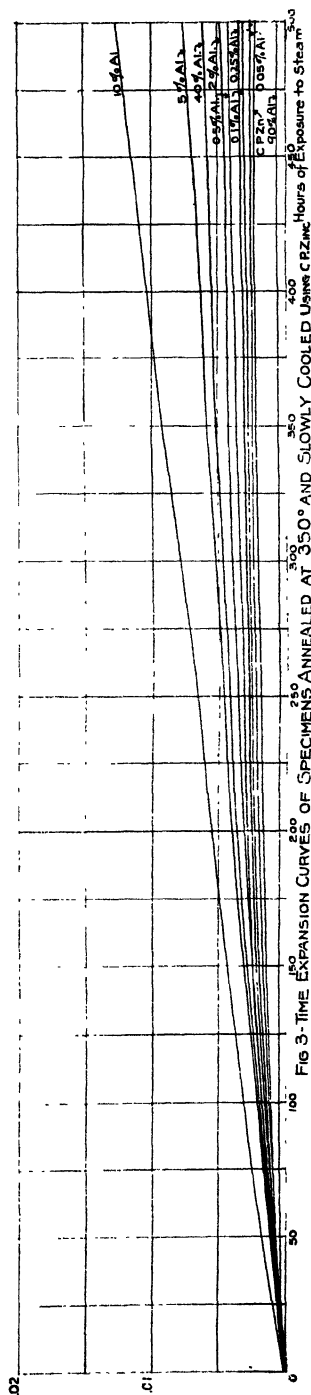
Fig 3—Time Expansion Curves of Specimens Annealed at 350° and Slowly Cooled Using C P Zinc. Hours of Exposure to Steam

Fig 4 - Composition-Expansion Curves Using C.P Zinc

Fig.5 Expansion Curves of Zn-Al Alloys Superimposed on Equilibrium Diagram According to Hanson and Gaylor
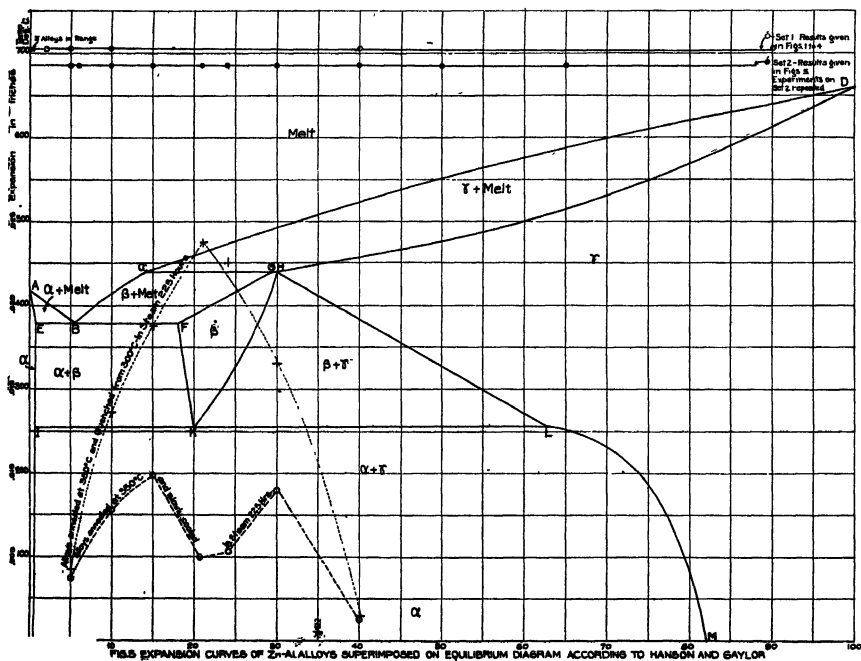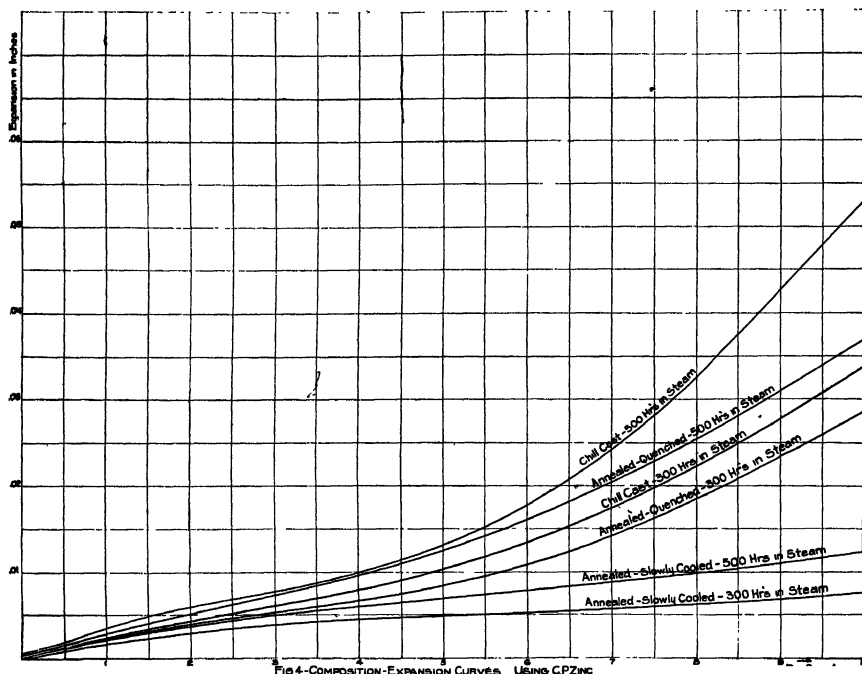
The expansion curves, however, reveal a pronounced difference between the slowly cooled and chill cast or quenched alloys containing β.

According to Hanson & Gaylor, the reaction $\beta \rightarrow a + \gamma$ takes place rapidly at room temperature even in quenched alloys. If this is so, the β constituent never exists below 256° C., but the structure observed and called β is in reality a mixture of finely divided $a + \gamma$. The hardness of alloys in the β range has been shown by the above authors to vary with ageing and the explanation would appear to rest upon a progressive increase in the size of the a and γ particles. A similar variation in particle size must also result from variation in the rate of cooling from 256° C., to room temperature.

The writers believe that the most logical explanation of all differences in the amount of expansion encountered in these alloys depends on two factors, of which the first is variation in grain or particle size, and hence variation in the number of grain boundaries where oxidation can occur.

Another point factor controlling the oxidizing attack on these alloys is the distribution of the two constituents a and γ, the former of which we have shown is subject (when constituting a single phase alloy) to intercrystalline attack and the latter of which is not so attacked. If these two phases maintain their characteristics when in contact with each other, the effect of heat treatment becomes complex in the range between about 15 and 35 per cent. aluminum, as shown in Fig. 5.

The general feature of the curves in Fig. 5 can be correlated and at least tentatively explained on the premises set forth in the two preceding paragraphs. Thus the alloys quenched from 300° C. should contain decomposed β, but the a and γ particles in this should be of the smallest possible size, and oxidation should be severe. This is the case.

The alloys slowly cooled from 350° C. differ from the quenched alloys in two respects. First, the a and γ particles of the decomposed β have had an opportunity to grow below 256° C. Between 350° and 256° C. separation of a and γ along the lines FK and HK has had an opportunity to occur and the presence of envelopes of γ or a around the β grains may explain the drop in the expansion curve.

The fact that chill cast alloys are somewhat more rapidly attacked than alloys quenched at 300° C., may be due to rapid cooling through the entire range from the point of freezing to the decomposition temperature, instead of from 300° only.

The slight amount of oxidation, as measured by expansion, above 40 per cent. aluminum, indicates that the presence of a matrix of γ (which must be present from about this point up to pure aluminum) protects the enclosed decomposed β and secondary a from attack.

Micrographs, Figs. 15 to 22, show unexposed specimens. Fig. 15 is C. P. Zinc and Fig. 16 is 0.25 per cent. aluminum chill cast. Fig. 17 shows

eutectic in a chill cast 2 per cent. aluminum alloy. Figs. 18 and 19 show the effect of a forty-hour anneal, in causing collescence of β in an alloy of approximately eutectic composition.

Figs. 20, 21 and 22 show the variation in the appearance of β, caused by different rates of cooling.

Micrographs 23 to 31 show the effect of 197 hours' exposure to wet steam. Fig. 23, C. P. Zinc, shows a slight edge (surface or exterior) attack due, we believe, to the tearing of the surface in machining. This invariably occurs to a slight extent in cutting coarse grained zinc. Fig. 24 shows a decidedly intercrystalline and very deep penetration in an alloy containing 0.25 per cent. aluminum. Fig. 25 shows penetration in a chill cast 2 per cent. aluminum alloy. The grain is finer and the depth of penetration less. Figs. 26, 27 and 28, a and b, show penetration of the 5 per cent. alloy as chill cast, slowly cooled and quenched, respectively. At this stage of penetration the attack is still largely intercrystalline. Figs. 29, 30 and 31 show a similar series in the case of the 10 per cent. alloy. Here the β is being severely attacked.

Micrographs 32, 33 and 34 show the 10 per cent. alloy after 600 hours in steam at a magnification of 250 diameters. The areas of β are completely altered in appearance.

Fig. 6 shows the great variation in depth of penetration caused by large variation in grain size. The photograph shows the fractures of three different alloys under three different conditions of heat treatment after 600 hours in steam. Corrosion has gone entirely through specimens 4263 and 4363 (Fig. 6) because of their enormous grain size. Specimens 5163, 5263 and 5363, having a small grain size, show least penetration.

Alloys composed wholly of γ show no penetration. Microscopic examination of a 90 per cent. aluminum alloy has verified this fact although no micrograph is shown.

PART V

Discussion of Tests Made on Alloys Containing One or More Added Metals

Micro-examination of sections through exposed specimens of alloys containing various metals occurring naturally as impurities or metals intentionally added, has served admirably to determine the depth to which corrosion has penetrated.

The effect of impurities and other metals on the rate of corrosion of zinc-aluminum alloys in the series made up with C. P. Zinc (Table 2) is shown in Figs. 35 to 46. Since corrosion also varies with the aluminum content, these micrographs with the exception of Fig. 46 were made from alloys containing the same percentage of aluminum, viz., 10 per cent.

After 144 hours' exposure to steam the first signs of corrosion are visible in the alloy of C. P. Zinc and aluminum without any other additions

3163

0.25% Al—chill cast. Penetration very deep as shown by dark border.

.3363

0.25% Al—annealed at 350° C. and slowly cooled. Grain size approximately same as 3163 but less marked penetration.

3263

0.25% Al—annealed at 350° C. and quenched from 300° C. Grain size same as 3263. Penetration greater.

4163

0.50% Al—chill cast. Grain size less than 3163. Penetration also less.

4263

0.50% Al—annealed at 350° and slowly cooled. Enormous grain size. Corrosion entirely thru specimen.

4363

0.50% Al—annealed at 350° C. and quenched at 300° C. Large grain size. Corrosion entirely thru specimen.

5163

2.00% Al—chill. Small grain size.

5263

2% Al—annealed at 350° C. and slowly cooled.

5363

2% Al—annealed at 350 °C. and quenched from 300 °C.

ALLOYS OF C. P. ZINC AND AL—EXPOSED TO STEAM 600 HOURS.

Penetration less marked at edges than in above specimens due to small grain size. Corrosion has penetrated through center due to porosity of casting.

FIG. 6.

FIG 7—EXPANSION CURVES OF ALLOYS SHOWING THE EFFECT OF ADDITIONS ON THE CORROSION OF ZINC-ALUMINUM ALLOYS ON EXPOSURE TO STEAM   USING C P ZINC

FIG.8—EXPANSION CURVES OF ALLOYS SHOWING THE EFFECT OF ADDITIONAL METALS ON THE CORROSION OF ZINC-ALUMINUM ALLOYS ON EXPOSURE TO BOILING WATER   USING C.P. ZINC

FIG 9-EXPANSION CURVES OF 85% Zn. 10% Al, 5% Cu ALLOY MADE WITH VARIOUS GRADES OF ZINC ANALYSES OF ZINCS GIVEN IN TABLE 1

FIG 10- EXPANSION CURVES OF 85% Zn, 12% Al, 3% Cu, ALLOY MADE WITH VARIOUS GRADES OF ZINC. ANALYSES OF ZINCS GIVEN IN TABLE 1.

FIG 11-EXPANSION CURVES OF 85% Zn, 10% Al, 10% Cu, ALLOYS MADE WITH VARIOUS GRADES OF ZINC
ANALYSES OF ZINCS GIVEN IN TABLE 1
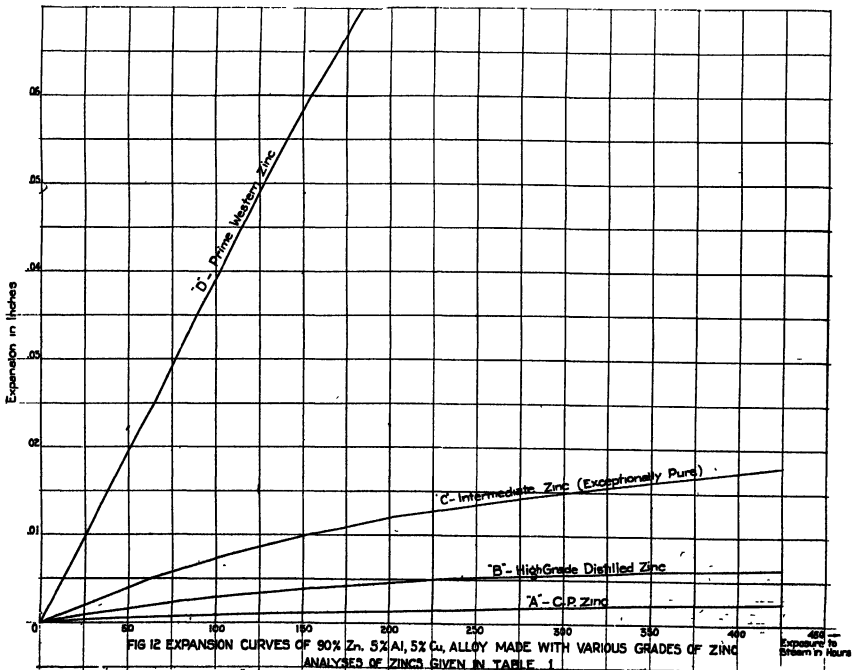
FIG 12 EXPANSION CURVES OF 90% Zn, 5% Al, 5% Cu, ALLOY MADE WITH VARIOUS GRADES OF ZINC
ANALYSES OF ZINCS GIVEN IN TABLE 1

FIG 15- EXPANSION CURVES OF 50 75% Zn, 0.5% Al, 2.75% Cu, 6% Sn ALLOY MADE WITH VARIOUS GRADES OF ZINC
ANALYSES OF ZINCS GIVEN IN TABLE 1
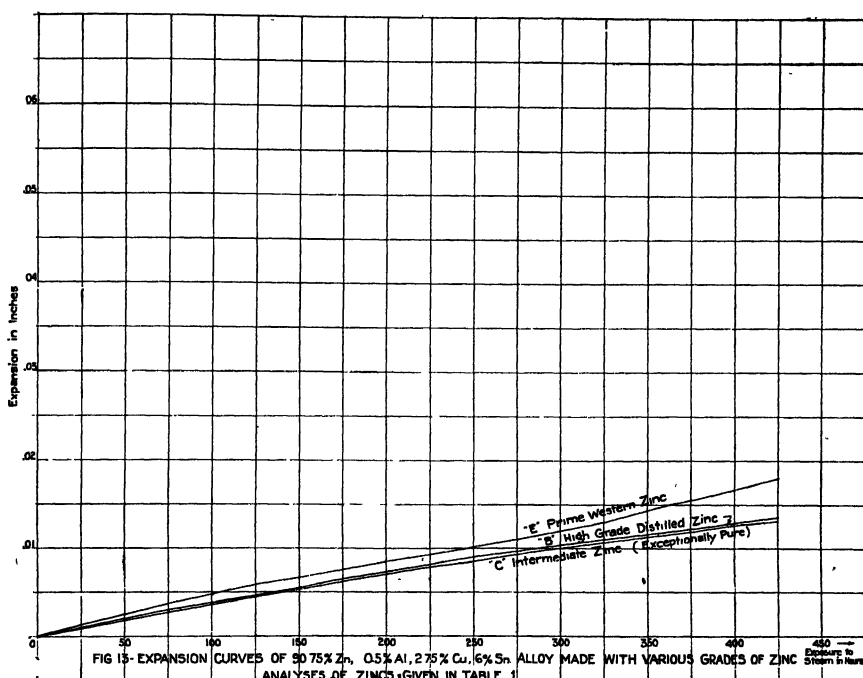
(Fig. 35), while the alloys containing 0.5 per cent. tin, 0.2 per cent. cadmium and 0.5 per cent. lead (Figs. 37, 38 and 39) are penetrated to a marked extent, especially the last mentioned. The alloy containing 0.5 per cent. copper is not visibly attacked.

After 288 hours, or 12 days, exposure to steam, the alloys containing tin, lead and cadmium had corroded so severely that oxidized layers could be peeled off and the specimens had been distorted considerably (micrographs were, therefore, not prepared). The pure zinc-aluminum alloy (Fig. 40) showed considerable corrosion. In alloys containing small quantities of copper (Figs. 41 and 42) corrosion had just become visible.

Finally, microsections were prepared after 572 hours, or approximately 24 days in steam. Micrographs of these are shown in Figs. 43 to 46. They show conclusively the retarding effect of the addition of copper on the corrosion of zinc-aluminum alloys. Fig. 46 shows furthermore that a 5 per cent. aluminum alloy with the requisite amount of copper and made with C. P. Zinc appears almost unattacked after 24 days in wet steam.

The facts illustrated in the micrographs lead to some speculation as to what causes acceleration or retardation of corrosion in these alloys. Copper probably causes the formation of a protective coating which slows down oxidation. Lead, cadmium, tin and iron, on the other hand, probably render the coating more permeable to oxygen or moisture.
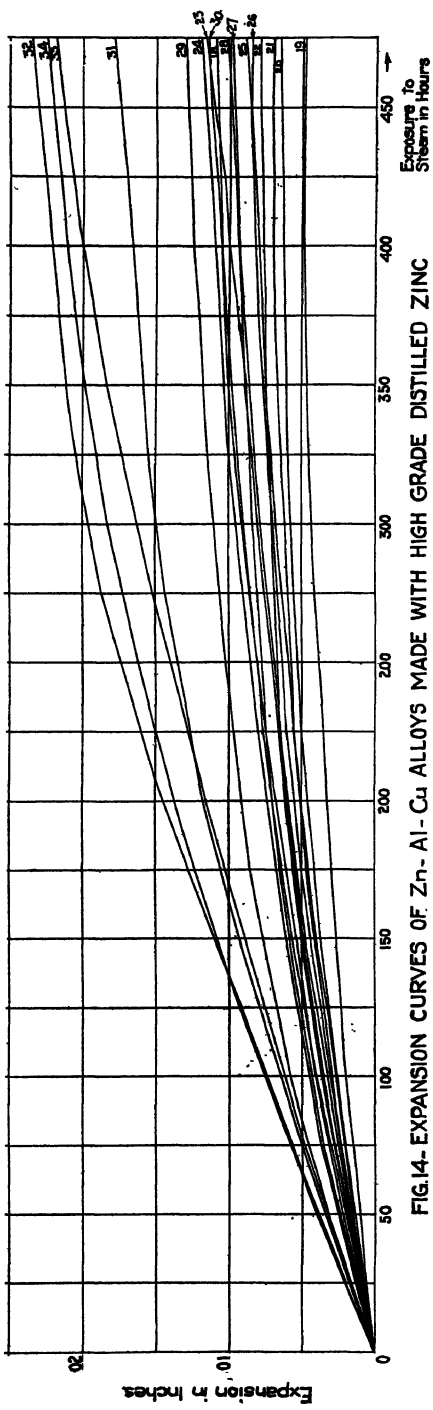
FIG. 14- EXPANSION CURVES OF Zn- Al- Cu ALLOYS MADE WITH HIGH GRADE DISTILLED ZINC

No.	% Zn	% Al	% Cu	% Ni
18	94.5	5.00	0.50	
19	93.5	5.00	1.50	
20	93.4	5.00	1.50	0.10
21	92.0	5.00	3.00	
22	90.0	5.00	5.00	
23	89.0	7.00	4.00 (Ternary Eutectic)	
24	92.5	7.00	0.50	
25	91.5	7.00	1.50	
26	91.4	7.00	1.50	0.10
27	90.0	7.00	3.00	
28	88.0	7.00	5.00	
29	89.5	10.00	0.50	
30	88.5	10.00	1.50	
31	88.4	10.00	1.50	0.10
32	87.0	10.00	3.00	
33	85.0	10.00	5.00	
34	85.0	12.00	3.00	

On the other hand, the marked effect of other elements on the rate of corrosion might be attributed to constitutional changes, as for example: a change in the rate of decomposition of β, or a change in the distribution of a and γ.

The alloys thus far considered were made of C. P. Zinc and represent ideal, but not practical, conditions. Figs. 47 to 54 are micrographs of alloys made with a high grade distilled zinc analyzing 0.039 per cent. lead, 0.010 per cent. iron and less than 0.005 per cent. cadmium (Table 4). A comparison between Figs. 46 and 54 shows that the effect of very slight impurities is noticeably harmful when alloys are subjected to the intense corrosive action of steam for 22 days.

A type of zinc base alloy, containing considerable tin (5-7 per cent.) and a small amount of aluminum (about 0.5 per cent.) is in common use and an example has been included (No. 15, Table 3). Fig. 52 shows such an alloy after long exposure to steam. Corrosion is considerably more severe than in alloys containing 5 per cent. aluminum and small percentages of copper (Figs. 53 and 54).

In alloys of small grain size, made with a high grade distilled zinc, the rate of corrosion is probably more clearly indicated by expansion curves. However, in view of the two factors, depth of penetration and amount of expansion, affecting corrosion, microstructure and expansion should be considered together. The effect of ordinary impurities and various other metals intentionally added, on the rate of expansion of alloys, of C. P. Zinc and aluminum is shown by the curves, Figs. 7 and 8, Fig. 7 showing the effect of steam and 8 of boiling water. The effect of various commercial zincs on the rate of corrosion of certain die casting alloys is shown in Figs. 9 to 13. Fig. 14 shows the effect and corrosion of varying the aluminum and the copper in zinc-aluminum-copper alloys made up with a high grade distilled zinc.

The harmful effects of lead, cadmium, tin and iron, respectively, as well as the beneficial action of copper are clearly shown in Figs. 7 and 8. Nickel and manganese (curves, Figs. 7 and 8) do not effect the rate of corrosion seriously and were, therefore, not dealt with, in detail after the first experiment with C. P. Zinc and aluminum (Table 2).

The importance of the use of pure zinc in zinc-aluminum-copper alloys is clearly seen on examination of Figs. 9 to 12.

The expansion of the commercially important high tin, low aluminum alloy, when made up with various commercial zincs, is shown in Fig. 13.

It may be noted in this connection that while this type of alloy shows fairly satisfactory permanence when made from any grade of spelter, it is inferior in this respect to the alloys containing moderate percentage of aluminum and copper but no tin, with the distinct reservation that they be made from high grade distilled metal. These facts may be best appreciated by comparing Figs. 13 and 14.
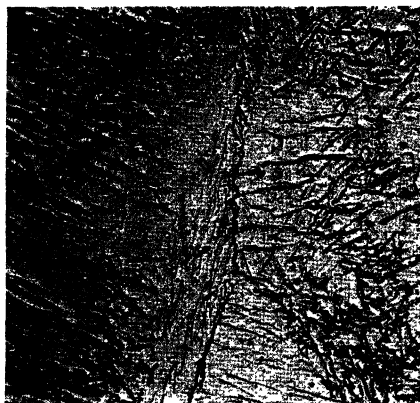
Fig. 15   X200.   Fig. 16.   X200.
          C. P. Zinc—chill cast.   0.25% Al—chill cast.

Fig. 17.   X200.   Fig. 18.   X200.
          2.00% Al—chill cast.   5.00% Al—chill cast.
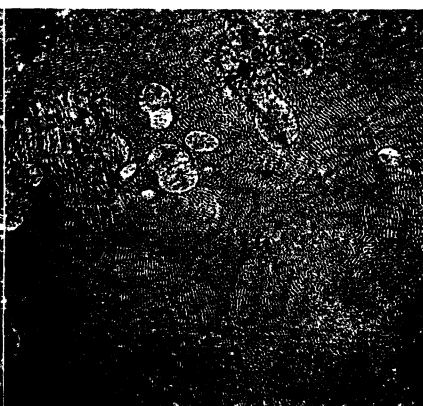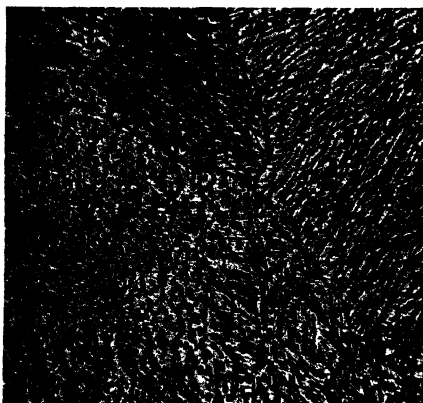
Fig. 19.   X200.   Fig. 20.   X200.
5.00% Al—annealed at 350° and slowly cooled.   10% Al—chill cast.
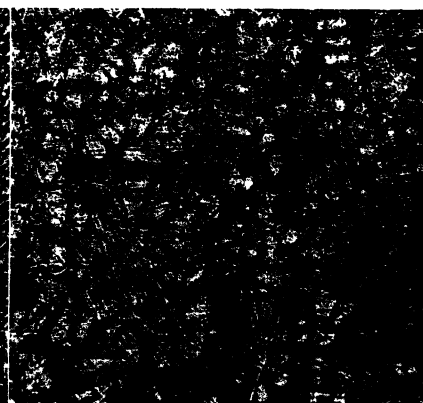
ALLOYS MADE WITH C. P. ZINC.
In the case of micrographs of specimens exposed to steam, the edge of specimen' may
be found at the left.
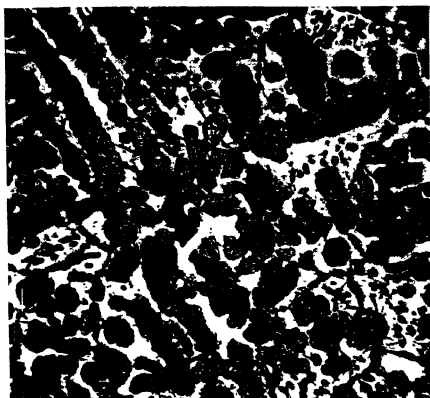
Fig. 21. X200.
10% Al—annealed at 350° C. and quenched from 300° C.
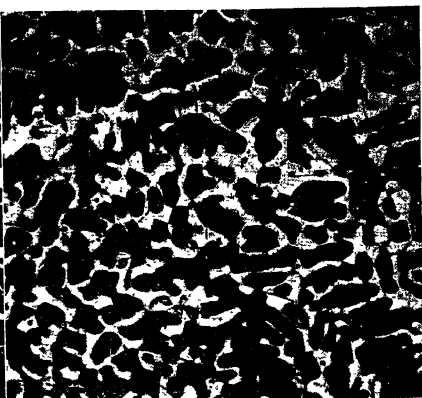
Fig. 22. X200.
10% Al—annealed at 350° C. and slowly cooled.

Fig. 24-a (edge). X150.
0.25% Al—chill cast. Exposed to steam 197 hours.
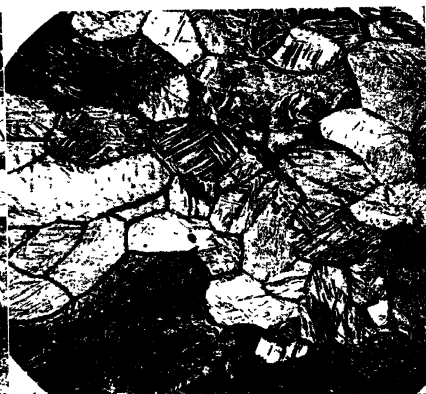
Fig. 24-b (center 1). X150.
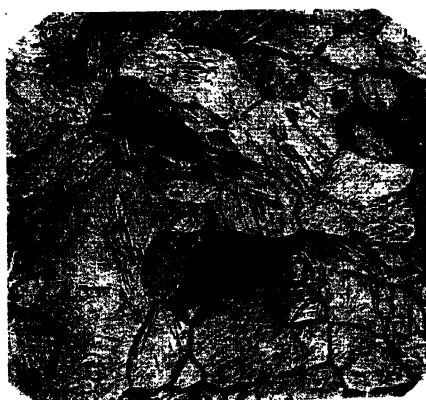
Fig. 24-c (center 2). X150.

Fig. 24-d (center 3). X150.

ALLOYS MADE WITH C. P. ZINC.
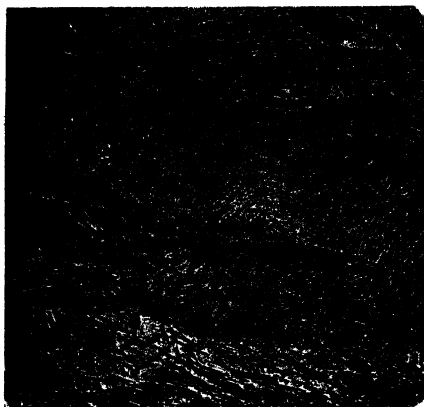
Fig. 23 (edge).   X150.
C. P. Zinc—chill cast. Exposed to steam 197
                hours.
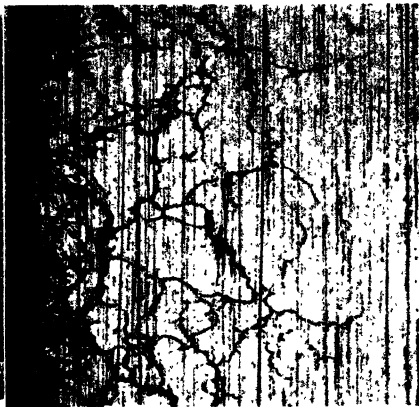
Fig. 25 (edge).   X150.
2.00% Al—chill cast. Exposed to steam 197
                hours.

Fig. 26 (edge).   X150.
5.00% Al—chill cast. Exposed to steam 197
                hours.
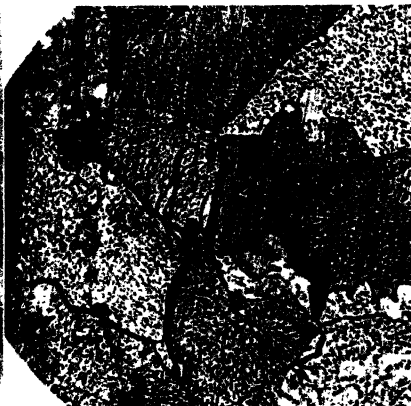
Fig. 27 (edge).   X150.
5.00% Al—annealed at 350° C. and slowly
     cooled.   Exposed to steam 197 hours.

Fig. 28-a (edge).   X150.
5.00% Al—annealed at 350° C. and quenched
from 300° C.   Exposed to steam 197 hours.
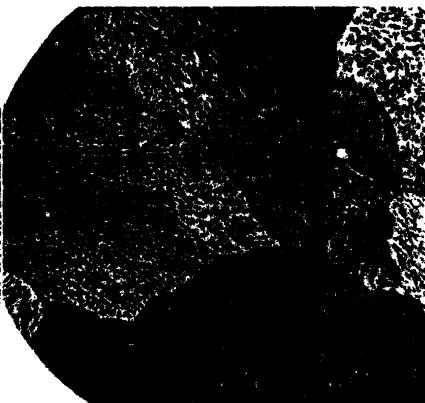
Fig. 28-b   X150.
5.00% Al—annealed at 350° C. and quenched
from 300° C.   Exposed to steam 197 hours.

ALLOYS MADE WITH C. P. ZINC.

Fig. 29 (edge). X150.
10% Al—chill cast. Exposed to steam 197 hours.

Fig. 30 (edge). X150.
10% Al—annealed at 350° C. and slowly cooled. Exposed to steam 197 hours.
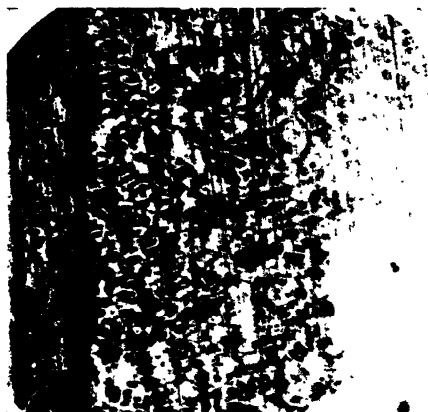
Fig. 31 (edge). X150.
10% Al—annealed at 350° C. and quenched from 300° C. Exposed to steam 197 hours.
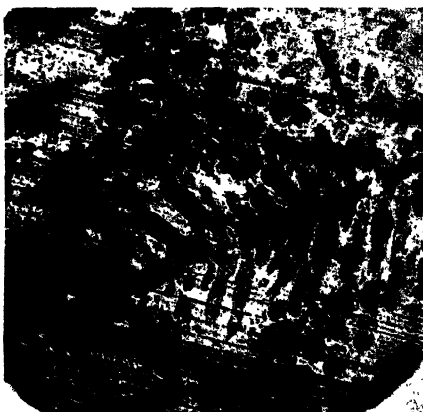
Fig. 32 (near edge). X250.
10% Al — chill cast. Exposed to steam 600 hours.
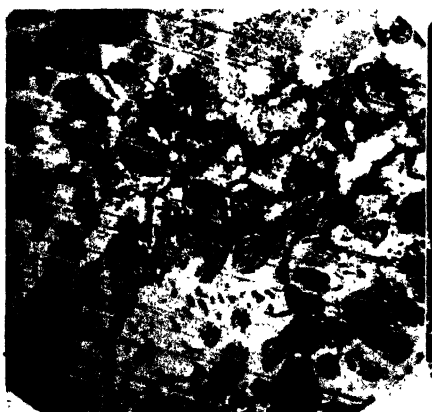
Fig. 33 (near edge). X250.
10% Al—annealed at 350° C. and slowly cooled. Exposed to steam 600 hours.
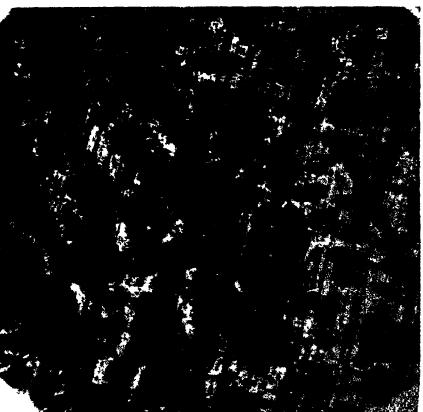
Fig. 34 (near edge). X250.
10% Al—annealed at 350° C. and quenched from 300° C. Exposed to steam 600 hours.
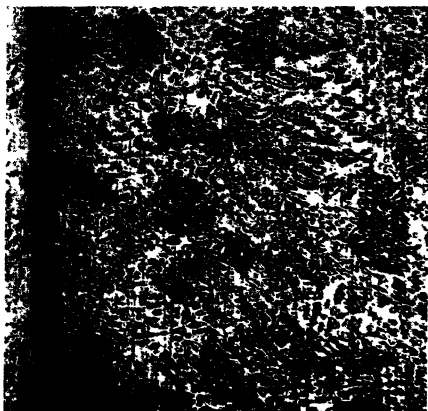
Fig. 35 (edge).   X100.
90% Zn, 10% Al. Very slight penetration of
                corrosion.
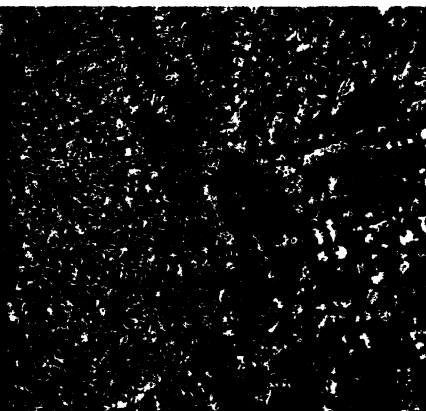
Fig 36 (edge).   X100.
Ratio Zn : Al—90 : 10, 0.5% Cu. No visible
                ·trace of corrosion.

Fig. 37 (edge).   X100.
Ratio Zn : Al—90 : 10, 0.5% Sn. Severe depth
   of penetration. Vertical lines show progress
   of corrosion.

Fig 38 (edge).   X100
Ratio Zn : Al—90 : 10, 0.2% Cd. Black por-
   tion shows average depth of penetration.

Fig. 39 (edge).   X100.
Ratio Zn : Al—90 : 10, 0.5% Pb. Very severe
        corrosion to considerable depth.

ALLOYS MADE WITH C. P. ZINC.
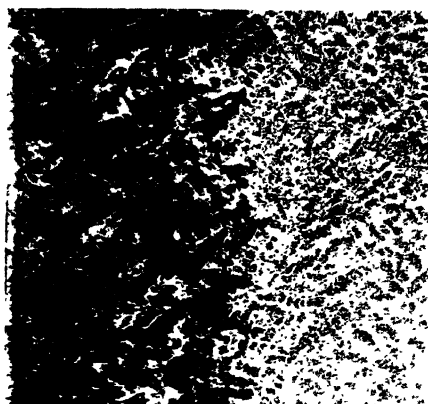
Fig. 40 (edge).   X100.
                90% Zn, 10% Al.

Fig. 41 (edge).   X100.
Ratio Zn : Al—90 : 10, 0.5% Cu. Very slight
            corrosion at edge of specimen.

Fig. 42 (edge).   X100.
Ratio Zn : Al—90 : 10, 5.0% Cu. Very slight
                corrosion.

ALLOYS MADE WITH C. P. ZINC.
Specimens exposed to steam 288 hours. All specimens in chill cast condition.

Fig. 53.   X100.
93.5% Zn, 5% Al, 1.5% Cu. Vertical lines of
        corrosion are visible.

Fig. 54.   X100.
90% Zn, 5% Al, 5% Cu. Vertical lines show
                corrosion.

ALLOYS MADE WITH HIGH GRADE DISTILLED ZINC.
steam 520 hours. Specimens in chill cast condition.
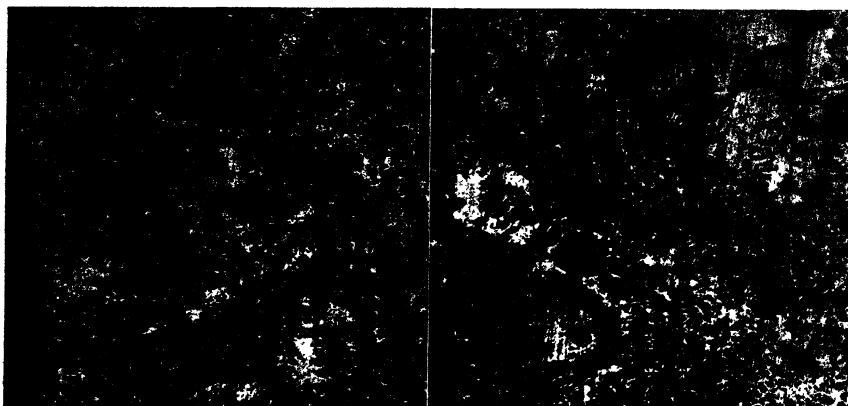
Fig. 43 (edge).   X100.
90% Zn, 10% Al. Very severe corrosion (deep
penetration).
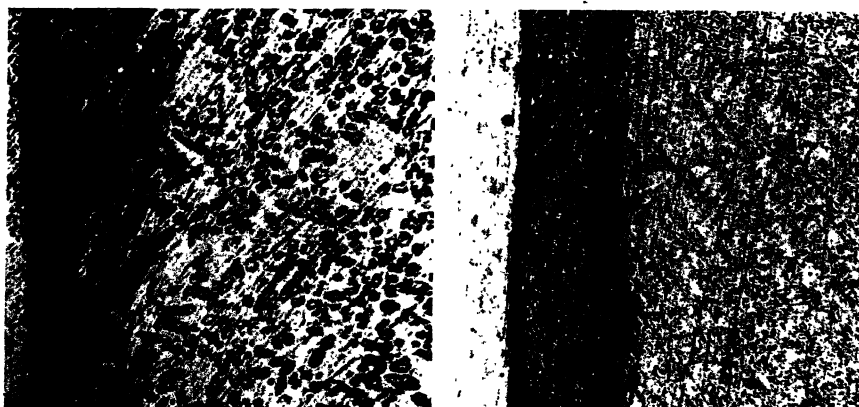
Fig. 44.   X100.   Fig. 45.   X100.
   Ratio Zn : Al—90 : 10, 0.5% Cu.   Ratio Zn : Al—90 : 10, 5.0% Cu.
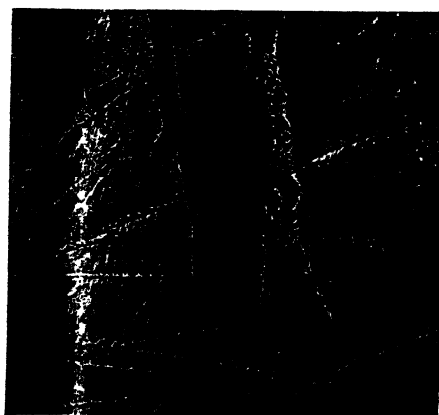
Fig. 46.   X100.
90% Zn, 5% Al. 5% Cu. Corrosion very slight.

ALLOYS MADE WITH C. P. ZINC.
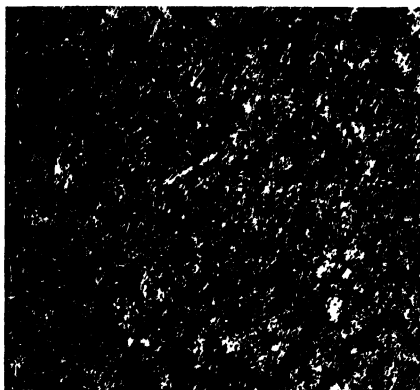
Fig. 47 (edge).   X100.
85% Zn, 12% Al, 3% Cu. Corrosion has pene-
    trated thru entire section shown.
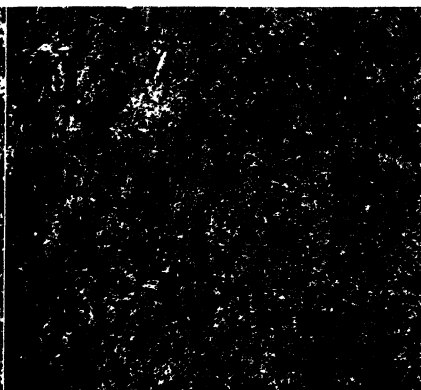
Fig. 48 (edge).   X50.
Same as 47 at lower magnification. Sound
    metal shows at extreme right.
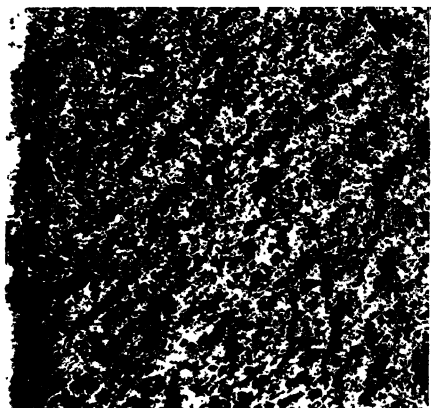
Fig. 49 (edge).   X100.
88.5% Zn, 10% Al, 1.5% Cu. Corrosion has
    penetrated thru entire section shown.

Fig. 50 (edge).   X50.
    Same as 49 at lower magnification.
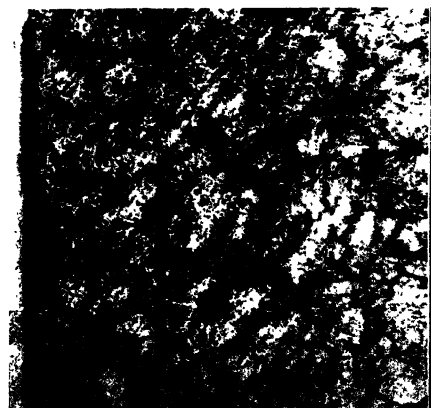
Fig. 51.   X100.
    91.5% Zn, 7% Al, 1.5% Cu.

Fig. 52.   X100.
90.75% Zn, 0.5% Al, 2.75% Cu, 6% Sn.
Severe corrosion to within ½ inch of right
    margin.

castings as possible. The section of the die casting used as a tensile specimen was approximately round with a diameter of ⅜ in.

Rough tests of the ductility made by bending die castings showed that the 5% Al, 1.5 to 3.0% Cu alloys, have considerably greater ductility than the other alloys tested.

While, as we have stated early in the paper, the expansion figures given in the table for sound chill castings probably are less than for die castings (which are more or less porous), nevertheless we have enough comparative data available to support the natural assumption that the various alloys will show the same relative merits when die cast, so as to produce castings of equal soundness.

Tests made on actual die castings indicate that the soundness of the die casting is a factor of prime importance, and that under certain conditions an alloy inferior to another in a sound chill casting, may, because of its tendency to produce denser die castings, equal the second alloy when die cast.

To a certain extent this may be true of high tin alloys made with high grade distilled zinc, which, in the limited number of cases we have observed appear to give somewhat sounder castings.

TABLE 5

Comparative Tensile Strength and Corrosion Data on Zinc-Aluminum Die Casting Alloys Using High Grade Distilled Zinc

Composition			Tensile Strength		Expansion in inches after 10 days in steam on sound chill castings
Al	Cu		Chill Cast	Die Cast	
5	0.5		21,600	......	.....
5	1.5		31,400	14,300	.0033
5	3.0		42,600	20,150	.0046
5	4.0		......	27,200	.....
5	5.0		47,600	23,200	.0055
7	.5		27,100	......	.0080
7	1.5		30,100	......	.0060
7	3.0		28,000	......	.0077
7	5.0		43,300	......	.0077
10	.5		36,900	......	.0097
10	1.5		33,800	......	.0075
10	3.0		46,100	......	.0150
10	5.0		49,100	......	.0135
12	3.0		50,600	17,450	.0152
		Sn			
0.5	2.75	6.0	......	13,500	.0085

DISCUSSION

C. H. MATHEWSON,* New Haven, Conn.—The authors set out to determine the causes of failure of die castings containing zinc and aluminum. The worst of these failures reveal actual swelling and exfoliation of the metal, indicating a positive change of dimensions caused by oxidation of a penetrating type. This effect can be produced artificially by exposure to moisture at an elevated temperature. The authors, therefore, sought, first, to determine the effect of composition, in terms of zinc, aluminum and various impurities, on the resistance of the alloys to corrosion of this sort.

Confirming various circumstantial evidence, they have shown that zinc-aluminum alloys made from pure metals are far more resistant than cruder products, and that even lead, the commonest of all impurities in commercial zinc, greatly lowers the resistance of zinc or of a zinc-aluminum alloy to corrosion in moist air. The tests were made on small chill-cast samples, but the authors have also experimented on actual die castings with similar results and are now making thorough experiments with die-cast test pieces, extending somewhat the scope of the original tests.

No change of dimensions could be measured on the small samples exposed under various conditions in the Palmerton tests, except in the case of the wet-steam exposures. The authors plan to discover, if possible, some relation between the time required for the casting to swell measurably at an ordinary low temperature in a water-saturated atmosphere and the time required for it to swell the same amount at the usual wet-steam temperature; so that some kind of an approximate permanence factor may be associated with alloys of various compositions.

Aside from the effect of corrosion, there is the possibility that small changes in length and mechanical properties may be caused by constitutional changes in these zinc-aluminum alloys with or without copper. For example, Rosenhain[6] and his associates measure a small amount of shrinkage in cast aluminum-zinc alloys after 12 months' exposure but the conditions of exposure are not stated. These effects are not unlike changes known to occur in rapidly cooled steels and other alloys that pass through constitutional changes, and can doubtless be controlled by simple heat treatments where such a need appears. They can hardly contribute in any great degree to the striking disintegration that may occur in alloys of incorrect composition, and which, I believe, is always the result of oxidation proceeding inwards.

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* Professor of Metallury, Sheffield Scientific School, Yale University.
[6] *Jnl.* Institute of Metals (1920).

W. M. PEIRCE.—There has been some criticism of Table 5, the critics claiming that the values for the tensile strength of the die castings are much too low.   To prevent any erroneous impressions, I wish to emphasize that we offer these results as relative values only.   Table 5 is based on tensile tests made on a die-cast part, and not on a standard tensile specimen.   This made it difficult to secure perfect alignment in the grips which, together with the irregular shape of the specimen, makes it impossible to compare these results with results from standard test bars.   We feel convinced, however, that the results give a true indication of the relative strength of the alloys.

C. H. BIERBAUM,* Buffalo, N. Y.—This problem has given our company trouble for over 20 years.   Being the manufacturers of a zinc-copper-aluminum alloy, we found many years ago that probably the greatest detrimental impurity in a casting is tin.   I have a small die casting made in 1896 that is in perfect condition while a casting made of the same composition 8 years ago has deteriorated and is warped and cracked on the surface.   The latter contains a small fraction of 1 per cent. of tin.   Tin in a zinc-aluminum alloy is detrimental; it seems to have somewhat the same effect as when alloyed with aluminum alone.

The problem is rather complex.   I do not believe that the trouble is entirely caused by oxidation from the outside; microscopic examinations made both recently and years ago caused me to believe that under some conditions disintegration of the material occurs completely within closed areas of bright surfaces.   Moisture and heat will cause the disintegration of zinc-copper-aluminum alloys that would last for years if heat and moisture were not present.

We have to look for three phases of this problem: (1) A constitutional change within an alloy of zinc-copper-aluminum and an impurity of tin without either heat or moisture; a case in which neither heat nor moisture are required for complete disintegration.   (2) The disintegration caused by oxidation and which seems to occur from an outside penetration—this is the one that is particularly aggravated or increased by moisture and heat.   (3) The suggestion made in the article in the *Metall-Kunde*, on the chemical compound $Al_2 Zn_3$, which is stable at the higher temperatures but not at the lower.

HENRY S. RAWDON,† Washington, D. C. (written discussion).—The results of this study of the deterioration of zinc-base die castings will go far to counteract the disfavor with which such alloys are regarded at present.   It is not to be denied that this disfavor, in many cases, has been the result of unsatisfactory experience with such alloys, and the

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* Vice-President, Lumen Bearing Co.
† Physicist, Bureau of Standards.

thorough manner in which the authors have attacked the problem shows that they fully appreciate its seriousness. The conclusions reached and the recommendations offered appear to be fully justified by the experimental data given and the writer believes that there should be no reason for hesitancy upon the part of manufacturers and users of die castings in following them. It is gratifying to the writer to learn that his analysis of the problem, referred to by the authors, based on a relatively small number of specimens was borne out by the investigation just made.

The phenomenon of internal oxidation in alloys, particularly those of the softer metals, is of considerably wider occurrence than might be inferred from the title of the present article. An important case that was investigated by the Bureau of Standards,[7] is the deterioration of tin when exposed to heat and moisture. Safety plugs in boilers are usually made of a cylindrical bronze casing filled with a metal of low melting point that will fuse and relieve the pressure within the boiler if for any reason the conditions of temperature and pressure exceed the safety limit. Tin is used almost exclusively as a filling for such plugs. It has been found necessary by the Steamboat Inspection Service of the Department of Commerce, to specify closely the purity of the tin that may be used for this purpose. The examination by the Bureau of Standards of a great number of plugs showed that the tin often deteriorates under the conditions to which it is exposed in service. Oxidation occurs at the water or steam end and progressively travels inward into the metal in the form of a tree-like network, presumably intercrystalline in nature. A pronounced swelling of the tin filling usually results from the internal oxidation. In some plugs examined, the oxidation of the tin filling was practically complete so that a hard, almost infusible mass of oxide filled the plug and by its failure to melt and relieve the pressure within the boiler when the necessity for it arose, caused serious accidents by boiler explosion. The presence of small amounts of zinc and lead as contaminations of the tin was found to exert a pronounced accelerative effect on the rate of deterioration of the tin. At present, these are two of the metals that are carefully specified as to the permissible amount in tin which is intended for use as a filling for safety-boiler plugs.

H. E. BRAUER and W. M. PEIRCE.—Since the presentation of this paper, the authors have received several criticisms, first, that the tensile strengths given for the die castings were too low, and, second, that it might be inferred from the paper that certain die-casting alloys, which the critics claim to be satisfactory in resistance to intercrystalline oxidation, are not resistant to this type of corrosion.

[7] G. K. Burgess and P. D. Merica; An Investigation of Fusible Tin Boiler Plugs, Bureau of Standards *Tech. Paper* No. 53.

In reply to the second criticism, we emphasize the fact that our results are based on exposure to wet steam, which is a very drastic test, but that we consider the results obtained of great value in judging the
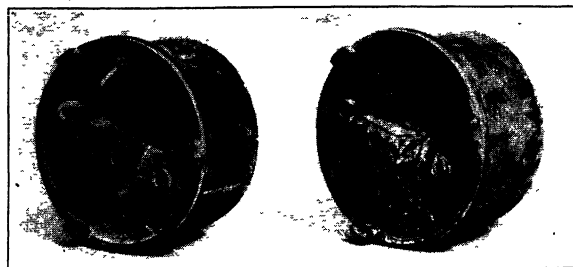
(a)   (b)

FIG. 56.—CASTINGS AFTER 4 DAYS EXPOSURE TO STEAM.   (a) SHOWS ONLY SURFACE OXIDATION; (b) SHOWS SEVERAL CRACKS; (a) 0.5 PER CENT. AL, 2.75 PER CENT. CU, 6 PER CENT. SN WITH HIGH-GRADE ZINC; (b) 0.5 PER CENT. AL, 2.75 PER CENT., 6 PER CENT. SN WITH "PRIME WESTERN."
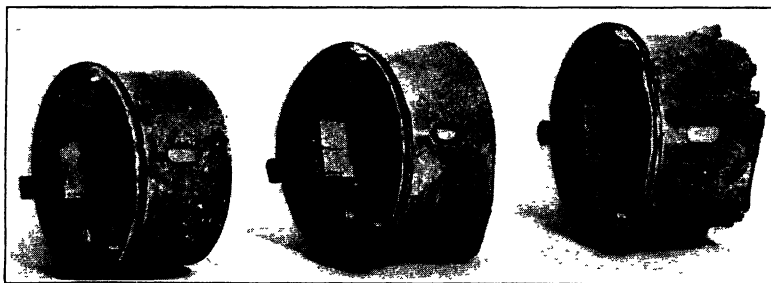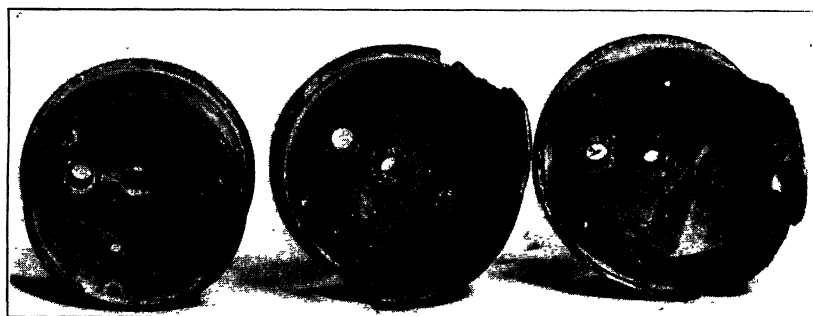
(a)   (b)   (c)

FIG. 57.—CASTINGS AFTER 4 DAYS EXPOSURE TO STEAM.   (a) IS AS GOOD AS CAST, EXCEPT FOR SURFACE OXIDATION; IT HAS EXPANDED AND CHANGED DIMENSIONS LESS THAN CASTING SHOWN IN FIG. 56; (b) AND (c) SHOW EXTREME CRACKING AND DISTORTION.   (a) 5 PER CENT. AL, 3 PER CENT. CU WITH HIGH-GRADE ZINC; (b) 5 PER CENT. AL, 3 PER CENT. CU WITH "PRIME WESTERN;" (c) 12 PER CENT. AL, 3 PER CENT. CU WITH "PRIME WESTERN."

TABLE 6.—*Physical and Casting Properties of Die-casting Alloy of Various Compositions*

Num-ber	Composition				Grade of Zinc Used	Tensile Strength, Lb. per sq. in.	Per Cent. Elongation	Angle of Bend, Degrees	Impact Energy Absorbed by Test Specimens with 0.634 sq. in. Cross Section, Ft.-lb.	Hardness	Per Cent. of Defective Castings (30 Castings) Inspected	Castings properties
	Per Cent. Zn	Per Cent. Al	Per Cent. Cu	Per Cent. Sn								
1	91.75	0.5	2.75	6.0	High-grade distilled	19,700	0.0	42	3.74	43.0	0.0	Poor, due to hot short.
2	91.75	0.5	2.75	6.0	Prime western	20,500	0.0	26	2.54	43.0	33.0	Poor, due to hot short.
3	92.0	5.0	3.0	...	High-grade distilled	40,700	1.7	102[c]	17.40	54.7	0.0	O.K.; no hot shortness.
4	92.0	5.0	3.0	...	Prime western	34,000	0.0	28	6.315	63.5	33.0[b]	O.K.; no hot shortness.
5	85.0	12.0	3.0	...	Prime western	38,000	0.0	15	1.31	71.3	0.0	O.K.; no hot shortness.

[a] Test made by bending test bar around a 1-in. mandrel by hand.
[b] The defective castings showed cracks where gates were broken off.
[c] A bent specimen of this alloy is shown in Fig. 58.

relative merits of different alloys.   Thus, if the alloy containing 0.5 per-
cent. aluminum, 2.75 per cent. copper and 6 per cent. tin satisfactorily
resists corrosion, as stated by several familiar with the alloy, we are
convinced that the 5 per cent. aluminum-3 per cent. copper alloy made
with high-grade zinc will prove more resistant, as indicated by the wet-
steam test.   Figs. 56 and 57 show strikingly the difference in resistance
to oxidation exhibited by alloys of several compositions after 96 hr.
exposure to wet steam.   We stated that the tensile-strength tests were
not made on a standard tensile specimen.   We are able now, however,
to present tensile, Brinell, bending and impact values on die-cast tensile
specimens of standard form having a section $1\frac{1}{8}$ by $1\frac{1}{2}$ in. (Fig. 58).   These
results are given in Table 6.   It will be found by comparison that the
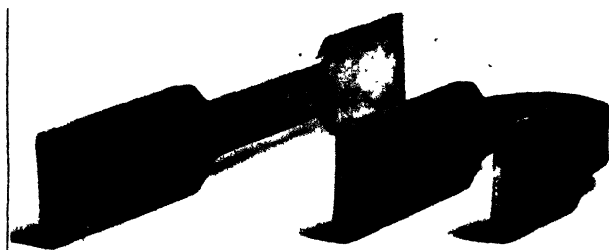
FIG. 58.—No. 3 ALLOY SPECIMEN USED FOR PHYSICAL-PROPERTIES TESTS.   IMPACT
TEST WAS MADE ON NARROW SECTION; HARDNESS TESTS WERE MADE ON ENDS OF
SPECIMENS.

results for standard specimens with a thin section are practically the
same as for ordinary chill castings.   The impact tests were made on
a machine practically identical with the Charpy machine but with a test
piece of different section, which renders these results only comparative.

A careful analysis of these and all previous data convince us that
the 5 per cent. aluminum-3 per cent. copper alloy with high-grade zinc
is highly desirable as a die-casting alloy because:

1. It has a high tensile strength and good ductility and is, there-
fore, tough.

2. It is not hot short, gives smooth castings, and is easily removed
from die.

3. It possesses high resistance to corrosion.

4. The resistance to corrosion is greater in the 5 per cent. aluminum-
3 per cent. copper alloy with high-grade zinc, than in an alloy contain-
ing 0.5 per cent. aluminum, 2.75 per cent. copper, 6 per cent. tin.

Cracks in Aluminum-alloy Castings*

By ROBERT J. ANDERSON, MET. E.,† PITTSBURGH, PA.

ROUGHLY, a crack in a casting may be considered, for the moment, to be due to fracture of the alloy resulting from the stress set up by the contraction in volume on passing from the liquid to the solid state. Cracks may be external or surface, internal or deep-seated, or incipient or so-called strain cracks.

Draws may appear as depressions, unevenness, and "pinching in" of the surface of a casting. Roughly, draws are due to contraction in volume and are governed largely by the same factors as cracks. A casting may be cracked, however, without being drawn.

Shrinkage is a foundry term and refers to the contraction in volume of a metal or alloy on passing from the liquid to the solid state. Pattern-maker's shrinkage is the total contraction in volume (including liquid, solidification, and solid shrinkage) that occurs when an alloy is poured into a mold. The contraction in volume of aluminum is one of the principal factors affecting the occurrence of cracks, as will be shown later, and the question of shrinkage and its control requires careful study in any treatment of cracking in castings.

In the case of sand and die castings, cracks have been attributed to a number of causes, but in the case of most light aluminum alloys, they are due to the large contraction in volume on freezing and the lack of its regulation or control.

The most important factors are: (1) Contraction in volume; (2) composition of alloy; (3) quality of melting charge; (4) design of casting; (5) method of molding; (6) hardness of ramming; (7) method of gating; (8) hardness and characteristics of cores; (9) chills; (10) risers; (11) melting temperatures; (12) furnace used for melting; (13) pouring temperatures; (14) inclusions in alloys; (15) hot shortness of alloys; and (16) physical properties of alloys at high temperatures. Of these, the first, second, third, and eleventh to sixteenth involve ascertainable metallurgical facts; the remainder are largely matters of opinion, judgment, and technical knowledge of men experienced in the production of castings.

Intercrystalline brittleness may be set up in aluminum alloys on heating for welding, owing to the absorption of gases.

EFFECTS OF VARIOUS FACTORS ON CRACKING OF ALUMINUM ALLOYS

Contraction in Volume

The liquid shrinkage, *i.e.*, the contraction in volume in the liquid state, is the amount of diminution of volume that occurs on cooling a metal or alloy at any temperature above its melting point to any other temperature above its melting point. The liquid shrinkage of aluminum, and some of its light alloys, is a nearly linear function of the temperature, as the density in the liquid state varies quite uniformly with the temperature.

The solidification shrinkage is the contraction in volume of a metal or alloy on passing from the liquid state at the freezing point to the solid state at the melting point.

The solid shrinkage is the contraction in volume that occurs on cooling a metal or alloy in the solid state; it may be determined over any temperature range from the formula for expansivity. Ordinarily, from the foundry standpoint, the solid shrinkage is regarded as the linear contraction in cooling from solid metal at the freezing point to room temperature. The coefficient of expansion of the light aluminum alloys varies from about 0.000022 to 0.000027 per degree centigrade, while for the usual alloys the patternmaker's shrinkage is taken as 0.156 in. per foot.

The solidification shrinkage of substantially pure aluminum, on passing from the liquid state (density, 2.382) at 658.7° C. to the solid state (density, 2.55) at the same temperature, is 6.6 per cent.[1] The liquid shrinkage of substantially pure aluminum on cooling from 1000° C. (density, 2.262) to 700° C. (density, 2.371) is about 4.6 per cent. The solid shrinkage of aluminum by calculation from the expansivity formula is 0.22 in. per ft., or 1.83 per cent. The total shrinkage (contraction in volume) of aluminum, or of any metal or alloy, on cooling from any temperature in the liquid state to any temperature in the solid state is the sum of the liquid shrinkage plus the solidification shrinkage plus the solid shrinkage.

Few data are available as to the contraction in volume of the various commercial aluminum alloys, but Edwards[2] gives the contraction in volume of a 92 : 8 Al-Cu alloy from the beginning to the end of freezing, *i.e.*, over the temperature interval 630° C. to 540° C., as about 7.0 per cent. The liquid shrinkage of this alloy in cooling from 1000° C. (density,

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[1] J. D. Edwards and T. A. Moorman: Density of Aluminum from 20° to 1000° C. *Chem. & Met. Eng.* (1921) **24**, 61–64.

[2] J. D. Edwards: Mechanism of Solidification of a Copper-aluminum alloy, *Chem. & Met. Eng.* (1921) **24**, 217–220.

2.434) to 700° C. (density, 2.524) is about 3.5 per cent. The solid shrinkage from the freezing point (540° C., where freezing is complete) to room temperature is taken as about 0.156 in. per ft., or 1.30 per cent.

TABLE 1.—*Linear Contraction of Some Aluminum Alloys in Sand Molds*

Approximate Composition, Elements, Per Cent.					tion, Per Cent.	Scale
Al*	Cu	Mn	Sn	Zn		
85.0				15.0	1.42	$\frac{1}{69}$
88.0	2.0			10.0	1.40	$\frac{1}{71}$
91.0	8.0	1.0			1.31	$\frac{1}{75}$
83.75	2.75			13.5	1.27	$\frac{1}{78}$
82.0	3.0			15.0	1.25	$\frac{1}{79}$
90.0	10.0				1.25	$\frac{1}{79}$
88.0	12.0				1.25	$\frac{1}{79}$
86.5	12.0		1.5		1.25	$\frac{1}{79}$
88.75	10.0	1.25			1.22	$\frac{1}{81}$
85.0	14.0	1.0			1.21	$\frac{1}{82}$
91.0	7.0		1.0	1.0	1.19	$\frac{1}{84}$

* Aluminum, difference.

Only a few data are available as to the linear contraction of aluminum alloys on casting in sand molds: Tables 1 and 2 indicate the tendencies for different alloys. Table 1 gives the results of measurements made at the National Physical Laboratory, Teddington, England, on bars 10.0 in. long by 1.0 in. by 0.50 in. cast in a sand mold between the faces of a steel templet that formed the ends of the mold, and then measuring the differences in length between the cast bar and the templet when cold. The least contraction in the series was 1.19 per cent. for the alloy containing about 91 per cent. aluminum, 7 per cent. copper, 1 per cent. tin, and 1 per cent. zinc, while the greatest contraction was 1.42 per cent. for the alloy containing about 85 per cent. aluminum and 15 per cent. zinc.

A number of measurements have been made at the Bureau of Mines on the linear contraction of aluminum alloys in graphite molds; a summary of a few of the measurements is given in Table 2. The method consisted briefly in pouring the alloys into specially prepared molds 1.000 in. square by 6.000 in. long and measuring the length of the cast bar at room temperature. Table 2 shows that the greatest linear contraction of the alloys was 1.52 per cent. for the alloy containing about 95 per cent. aluminum, 3 per cent. copper, and 2 per cent. manganese, while the least contraction was 1.17 per cent. for the alloy containing about 87 per cent. aluminum and 13 per cent. silicon.

Although the less the contraction in volume (or the less the linear contraction) the less likely is the alloy to crack, this is not an absolute

TABLE 2.—*Linear Contraction of Some Aluminum Alloys in Graphite Molds**

Approximate Composition, Elements Per Cent.								Linear Contraction, Per Cent.	Patternmaker's Scale
Al†	Cu	Fe	Mn	Ni	Si	Sn	Zn		
100								1.74	1/59
92	8							1.40	1/70
90	10							1.25	1/79
88	12							1.20	1/82
91	8	1						1.24	1/80
95	3		2					1.52	1/65
90	8			2				1.28	1/77
87					13			1.17	1/84
90	8					2		1.35	1/73
85	3						12	1.20	1/82

* These figures refer to bars poured at 800° C.; the actual linear contraction varies with the pouring temperature.
† Aluminum, difference.

criterion of the cracking tendency because certain alloys that may have a relatively small contraction in volume may be excessively weak at high temperatures. The 91:8:1 Al-Cu-Fe alloy has a materially lower linear contraction than the 92:8 Al-Cu alloy but it has greater strength at elevated temperatures; consequently it is less likely to crack on casting. As the 87:13 Al-Si alloy has a low linear contraction, certain complicated castings have been poured from it without the use of chills, when the same castings made in the 92:8 Al-Cu alloy required chills. To sum up: The contraction in volume of aluminum alloys is a most important factor in the cracking tendency and is variable in different alloys depending on the composition.

Composition of Alloys

The fact that certain light aluminum alloys on casting are more likely to crack than others is well known; that is, the cracking tendency varies with the chemical composition of the alloys. As a general rule, the aluminum-zinc alloys are known to crack more readily than the aluminum-copper alloys, while some aluminum-copper-zinc alloys (such as 85:3:12 Al-Cu-Zn) crack less readily than the former. In general, the contraction in volume of the aluminum-zinc alloys is considerably greater than that of the aluminum-copper alloys; these alloys also are much weaker at high temperatures than the latter. On the other hand, while the contraction in volume of some aluminum-copper-zinc alloys is less than that of some aluminum-copper alloys, the former are much weaker at high temperatures. The strength at high tempera-

tures may be a more important factor than the contraction in volume, at least in the case of some alloys. Heterogeneity, due to segregation, is much more marked in some aluminum alloys than in others; this leads to planes of weakness that may give rise to actual cracks or to strain cracks. While the cracking of steel ingots and castings is somewhat analogous to the cracking of aluminum-alloy castings, the contraction in volume of the aluminum alloys is so large that the effect of this factor overshadows the effect of other factors that are of importance in the case of steel. In the case of aluminum alloys, consideration must be given to any possible contractions or expansions on passing through the solidification range or in cooling in the solid state.

Quality of Melting Charge

Some foundrymen claim that the 92:8 Al-Cu and related aluminum-copper alloys are more likely to crack when made up from all-primary materials, than when some foundry scrap is used in the charges. In other instances, it is claimed that secondary aluminum and secondary aluminum-alloy pig are likely to give rise to cracks. These statements show that the chemical composition of the alloy and the effect of impurities must be considered. Much of the trouble experienced in this direction is caused by lack of chemical control of the charges. It is also claimed that so-called "brittle" aluminum pig, which may or may not have cracks in the webs, will yield cracked castings. This question requires research; the results of tests made to date are not wholly conclusive. The question of impurities in aluminum pig and in melting charges, more especially silicon and aluminum oxide, appears to merit investigation in its relation to cracks, but the available data in regard to this matter are very meager.

Design of Castings

One of the most important factors that affect the occurrence of cracks in aluminum-alloy castings is the design. As is well known, thick and thin sections in contiguity form a prolific source of cracks. If a rounded fillet is placed at the juncture of the thin and thick sections, the bad effects of the design will be overcome in part, and the section will be less likely to crack. It is generally necessary to chill heavy sections that are in contiguity with light ones so as to increase the rate of cooling of the former; an alternative is to feed with a riser.

The effects of right and re-entrant angles on the occurrence of cracks have been discussed by Rosenhain[3] and the author.[4] The plane of weakness

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[3] W. Rosenhain: "Introduction to Study of Physical Metallurgy," second ed. 291–293, New York, 1917, D. Van Nostrand and Co.

[4] R. J. Anderson: Blowholes, Porosity, and Unsoundness in Aluminum-alloy Castings, Bureau of Mines *Tech. Paper* 241 (December, 1919) 21–22.

that results from the freezing of metal in the form of columnar crystals at a right angle has been discussed by Desch.[5] Both strain cracks and actual cracks may be associated with the planes of weakness set up by columnar-crystal growth; the latter may be prevented by the use of proper fillets. The shape and dimensions of the casting are important factors in the occurrence of columnar crystals, which generally stand perpendicular to the outside surfaces, *i.e.*, at sections in the casting that were in contact with the walls of the mold, and they are found most often in thick sections, unless these have been chilled, as well as at right angles and at re-entrant angles. Columnar crystallization gives rise to zones of weakness as has been pointed out by Aitchison[6] in the case of steel ingots. These zones in a casting may include the following: (1) The juncture of each columnar crystal with its neighbors; (2) the juncture of sets of columnar crystals; and (3) the juncture of a set of columnar crystals with a set of equiaxed crystals. Microscopic examination has shown that cracks may exist at the juncture of two sets of columnar crystals in aluminum alloys, and that foreign inclusions may be forced into such junctures as well as between any two given columnar crystals. Fig. 1 shows two fissures at the juncture of a set of thin and thick columnar crystals in a section of the 92 : 8 Al-Cu alloy.

METHOD OF MOLDING

Under the method of molding, there may be included ramming, gating, cores, risers, and chills. Of course, the sand should be tempered properly and it should not be too wet; wet sand may cause cracks and warping. The bottom boards under the flask or mold box should be placed correctly and the box should be set evenly on the floor; otherwise strains will be set up in the mold. All molding should be done as cleanly as possible. Molding methods are not very variable after they have been worked out properly for a given pattern, but the various details of molding, *viz.*, ramming, gating, risers, cores, chills, etc. are important in relation to the occurrence of cracks.

Ramming of Molds

Sand molds for aluminum alloys should be rammed lightly and evenly. If the sand is rammed too hard, the casting will be prevented from moving while contracting on cooling, thereby setting up strains and, at times, causing fractures in the casting. On the other hand, if the

[5] C. H. Desch: "Metallography," 177–178, London, 1910, Longmans, Green, and Co.

[6] L. Aitchison: Zone of Weakness in Solidified Ingots, *Chem. & Met. Eng.* (1920) **23**, 280.

mold is rammed too lightly, there may be sudden collapsing of parts of the mold while the temperature of the alloy is immediately below that of final freezing, or in the range between the liquidus and the solidus. Strains set up by the collapse of the mold may be sufficient to cause cracking because the aluminum alloys in general are quite hot short, *i.e.*, weak at temperatures immediately below the solidus. The sand may be rammed loosely around the gates to prevent "hanging" of the casting in the mold, especially for some types of castings. Thin sections should be rammed more lightly than thick sections. In general the hardness of ramming is an item governed principally by experience, but the amount of ramming can be determined with fair accuracy if power molding machines are used.

Method of Gating

Theoretically, a casting should be so gated that when the mold cavity is full of metal, the length of the path over which the metal has traveled will have been such that the heavy sections will be filled with colder metal than the light sections. Mechanical difficulties[7] make such gating impracticable in most commercial castings. In the average crankcase, and in many much more simple castings, perfect feeding and gating are mechanically impossible, and the only thing that can be done is to approach ideal conditions as nearly as possible. The sprue and leader should be made heavy and of such size that they will not solidify prior to the casting, and fillets of ample size should be employed where the gate is attached to the casting. If the gate solidifies prior to the casting, it will draw from the casting and hence be likely to cause cracks. Where cracks can be traced to the method of gating,[8] an alteration of this may often effect their elimination. Usually, that part of the casting nearest to the gate is the last to solidify.

Hardness and Characteristics of Cores

According to Gillett,[9] one of the principal causes for cracks is the use of cores that are too hard or will not crush on contraction of the alloys. Cracks will always be less with green-sand cores than with dry-sand cores, although if higher pouring temperature must accompany the use of green-sand cores, this would tend to increase cracks. The dry-sand core is unyielding as a rule, unless the binder, like resin, will soften readily at the temperature set up on pouring. Suitable tests may be

[7] W. A. Gibson: Position of Tensile Tests in the Foundry, *Iron Age* (1920) **105**, 725–728.

[8] F. H. Hurren: Wasters, Presidential address before Coventry branch, Inst. of British Foundrymen, 1921. Abstract in *The Foundry* (1921) **49**, 262–264.

[9] Private communication, H. W. Gillett, Feb. 28, 1921.

applied to determine[10] the kind of core required for different castings and the effect of different cores on the occurrence of cracks.

Chills

Speaking generally, so far as chills are related to the occurrence of cracks, this defect has been attributed to: (1) Lack of chills where chills are necessary; (2) chills placed in the wrong position in the mold or cores; (3) too heavy chills; and (4) too light chills. Where a thin and a thick section are in contiguity, the thin section solidifies first and the thick one later; the function of the chill is to equalize the rates of solidification in the two sections by increasing the rate of freezing of the heavy section. It may be said in general that the failure to chill heavy sections is a usual cause for cracks.

Chills should vary in power, depending on their position in the mold and on the size of the section to be chilled; the effects of too light and too heavy chills are self evident and they need not be elaborated upon here.

Risers

The number of risers required for a casting depends on the area of the surface at the top of the casting, among other things, and on the relation of the area of the upper surface to that of the lower surface. Further, the number of risers required depends on the number of sections to be fed, and their size depends on the size of the sections. Too many risers should not be used because the casting may be anchored in the mold and it will not be free to move.

Melting Temperatures

Irrespective of the type of furnace, heating to too high a temperature should be avoided at all times, for metals and alloys dissolve more gases with increasing temperatures. On the basis that the greater part of the dissolved gas is carried in the amorphous cement at the grain boundaries in the solid state, it might be expected that the forcing of gas into the grain boundaries would give rise to strains and weakness at these places, and consequently aggravate the tendency to crack.

Furnace Used

Good castings have been made from alloys melted in all types of furnaces. In considering the effect of the furnace used, it is necessary to know the constitution of the furnace atmosphere. Thus, melting at

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[10] F. L. Wolf and A. A. Grubb: Laboratory Testing of Sands, Cores, and Core Binders. *Trans.* (1920) **64**, 630.

a relatively high temperature in an atmosphere containing much free oxygen (as in most reverberatory and open-flame furnaces) would give rise to more aluminum oxide in the alloy than melting at the same temperature in an atmosphere containing little oxygen but much carbon dioxide. Occluded dross particles (and perhaps also dissolved aluminum oxide) will give rise to weak metal, which would tend to crack more readily than metal free from occlusions. The melting temperature is governed quite largely by the type of furnace employed; charges are generally heated to higher temperatures in furnaces of the open-flame type than in iron-pot furnaces. Further, the type of furnace employed determines to some extent the amount of impurities present in the resultant alloy. When the 92:8 Al-Cu alloy is melted in iron-pot furnaces, the iron content may be from 1.0 to 2.0 per cent., but if melted in crucibles or against refractory brick linings, it will be about 0.40 to 0.75 per cent. In the same way, the silicon content of alloys melted in crucibles or against siliceous linings will be much higher than when melted in iron pots. Other things being equal, therefore, it would be expected that the 92:8 Al-Cu alloy would be less likely to crack when melted in iron pots than in other types of furnaces.[11]

Pouring Temperatures

Cracks in aluminum-alloy castings have been attributed to too high and too low pouring temperatures; but opinion is variable as to what should be regarded as the correct pouring temperature for the different commercial alloys. An alloy poured at a high temperature will have a considerably greater total liquid and solidification shrinkage than one poured at a low temperature, since the total shrinkage is a function of the pouring temperature. In general, a high pouring temperature (which presupposes a high melting temperature) may be expected to cause cracking, which would not occur with a lower pouring temperature under otherwise similar conditions. A too low pouring temperature has been claimed to be the cause of cracking, but it is difficult to see how this can be the case.

Inclusions in Alloys

The presence of foreign inclusions in cast aluminum alloys, such as dross particles, so-called "hard spots," and other included matter has been associated with cracking. As will be explained later, cracks in both eutectiferous and solid-solution alloys have a strong tendency to be intergranular, and where foreign included matter is located at grain

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[11] It has been claimed that dissolved iron from the pots does not have the effect that added iron (in the form of scrap sheet or of ferro-aluminum) has, although this does not seem true.

boundaries (or elsewhere), it tends to weaken the alloy and aggravate any condition which may lead to cracks.

While cracks have been found in aluminum-alloy castings quite free from foreign inclusions, they are found more often in unsound and dirty castings. Inclusions in the alloys may be avoided by cleanliness in charging and melting practice, and much of the trouble traceable to cracks arising from this cause can be eliminated.

Hot Shortness of Alloys

The relative tendency of different aluminum alloys to crack on casting may be determined by the hot-shortness tester,[12] devised by Gillett and Skillman.

Physical Properties at High Temperatures

So far as is now known, the greater the ultimate strength of any alloy at temperatures approaching the melting point, and the greater the elongation, the less is the tendency to crack. The ultimate strength and elongation in the cold cannot be taken as criteria of this tendency, nor can either of these properties be considered alone. Thus, certain aluminum-zinc alloys have enormous elongation at elevated temperatures but very low strength. These alloys crack more readily than some aluminum-copper alloys that have only moderate elongation at high temperatures, but still fair strength.

On the whole, the elongation may be regarded as a poor measure of the cracking tendency and, as determined by the ordinary tensile strength at high temperatures, it will not reveal necessarily the presence of brittleness,[13] even when present in high degree. Extreme ductility under gradually and steadily applied tension may be accompanied by brittleness to sudden strains or shocks.

Bengough[14] has examined the effect of increasing temperatures on the properties of rolled aluminum and finds that the ultimate strength drops from 17,200 lb. per sq. in. at 20° C. to 3400 lb. at 375° C., and to 380 lb. at 625° C. The ultimate strength of various aluminum alloys at high temperatures is variable; some tests have shown that this value may be affected markedly by the addition of small amounts of other elements

[12] A. B. Norton: A Hot Shortness Testing Machine for Aluminum Alloys, *Trans. Am. Inst. of Metals*, (1914) **8**, 124–126.

[13] W. Rosenhain and S. L. Archbutt: Alloys of Aluminum and Zinc: Tenth report to the Alloys Research Committee. *Proc. Inst. Mech. Engrs.* (1912) **76**, 414–416.

[14] G. D. Bengough: A Study of the Properties of Alloys at High Temperatures. *Jnl. Inst. of Metals* (1912) **7**, 123–174.

to certain alloys. Thus, the addition of iron, nickel, or manganese in relatively small percentages to certain light aluminum-copper alloys increases the strength markedly at elevated temperatures; Table 3 shows the effect of iron on the 92:8 Al-Cu alloy. The addition of 1 to 3 per cent. nickel to aluminum-copper alloys containing 8 to 12 per cent. copper markedly improves the strength both in the cold and at elevated temperatures. The alloy 86:12:2 Al-Cu-Ni has an ultimate strength of about 20,000 lb. per sq. in. at 250° C., which is about the same as that of the alloy 85:14:1 Al-Cu-Mn. This may be compared with 14,000 lb. for the alloy 88:12 Al-Cu and with 13,000 lb. for the alloy 91:7:1:1: Al-Cu-Sn-Zn, both at 250° C.

TABLE 3.—*Tensile Strength of Two Light Aluminum Alloys (Sand-cast Bars) at High Temperatures**

Temperature, Degrees Centigrade	Values for 92 : 8 Al-Cu Alloy			Values for 90.25 : 8 : 1.75 Al-Cu-Fe Alloy		
	Ultimate Strength, Pounds per Square Inch	Elongation in 2 In., Per Cent.	Reduction in Area, Per Cent.	Ultimate Strength, Pounds per Square Inch	Elongation in 2 In., Per Cent.	Reduction in Area, Per Cent.
100	21,800	1.25	0.8	25,000	0.5	1.6
200	18,500	3.00	2.0	22,000	1.0	2.0
300	17,500	6.00	4.0	19,900	2.0	3.3
400	9,900	15.5	17.7	10,200	6.5	12.3
500	3,000	24.2	38.8	3,900	22.5	30.3

* Values given are the mean of three determinations.

Rosenhain and Archbutt[15] have examined the strength of a series of aluminum-zinc alloys, containing from about 9 to 26 per cent. zinc, at high temperatures. All these alloys are exceptionally weak, and many of them suffer a marked loss of strength at temperatures as low as 50° to 100° C. Some of the alloys have enormous elongation at high temperatures, as shown in Table 4. Attention is called to the 130 per cent. elongation of the 80:20 Al-Zn alloy at 595°C. Of course, at this temperature, the alloy is in the range between the liquidus and the solidus, and some liquid is present.

To sum up: alloys that have fairly good strength at high temperatures are less likely to crack than those which are very weak, but this property must be correlated with the contraction in volume. Thus, although the strength of one alloy at, say, 500° C. is considerably greater than that of another alloy, but the contraction in volume is substantially greater, the tendency to crack in the first alloy might be much greater than that in the second alloy.

[15] W. Rosenhain and S. L. Archbutt: *op. cit.*, 408–416.

TABLE 4.—*Tensile Properties of Two Aluminum-zinc Alloys at Elevated Temperatures**

Temperature, Degrees Centigrade	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.
Values for 80 : 20 Al-Zn alloy			
100	22,800	28,040	18.0
250	10,380	10,380	28.5
300	7,480	7,480	28.0
400	1,880	1,880	92.0
500	780	780	92.5
595	700	700	130.0
Values for 74 : 26 Al-Zn alloy			
150	26,260	35,280	15.5
200	21,140	21,140	11.5
250	13,500	13,500	21.5
300	8,160	8,160	28.5
400	2,680	2,680	48.0

* Based on Rosenhain and Archbutt.

METALLOGRAPHY OF CRACKS

Many samples of cracked aluminum-alloy castings have been examined in the metallurgical laboratory of the Bureau of Mines with a view to determining the causes for the occurrence of cracks and to obtain light on the problem in general. Illustrative cases where cracks can be ascribed definitely to particular causes are not met with frequently in foundry practice, and in the samples here described, the history is quite faulty. The metallography of cracks may be discussed conveniently by reference to several typical cases. The cases here described are a few from a large number of samples submitted to the Bureau of Mines.

Cracked Vacuum-cleaner Housing

The first case is the occurrence of cracks in certain lots of vacuum-cleaner housings made in the 90:10 Al-Cu alloy. The charge consisted of all-primary aluminum pig plus 50:50 Al-Cu alloy. When the job was first put into production, the castings were found to be cracked badly, but at the end of the third or fourth day, after the foundry scrap, defectives, gates, etc., were in circulation, cracking ceased. The iron content of the alloy had increased materially after repeated remeltings (iron-pot practice employed), so the cracks were attributed to lack of iron in the alloy. Figs. 2 and 3 show the appearance of two cracks in a housing, while Fig. 4 shows the intergranular character of the crack in

Fig. 3—the crack follows the boundaries of the large grains. Fig. 5 shows the appearance of part of the crack in Fig. 4, while Fig. 6 indicates

FIG. 1.—FISSURES AT JUNCTURE OF SETS OF COLUMNAR CRYSTALS IN 92 : 8 AL-CU ALLOY; ETCHED WITH NaOH; OBLIQUE ILL. × 5.

FIG 2.—CRACK IN 90 : 10 AL-CU VACUUM-CLEANER HOUSING, UN-POLISHED. × 4.5.

FIG. 3.—CRACK IN 90 : 10 AL-CU VACUUM-CLEANER HOUSING, UN-POLISHED. × 4.5.

a tendency for the crack to follow along the eutectic boundaries of the small grains.

Cracked Crankcase

The second case is a sample taken in the production of an eight-cylinder crankcase for a standard motor car made with the 92:8 Al-Cu

FIG. 4.—SAME AS FIG. 3, BUT POLISHED AND ETCHED WITH NaOH, OBLIQUE ILL.   × 6.
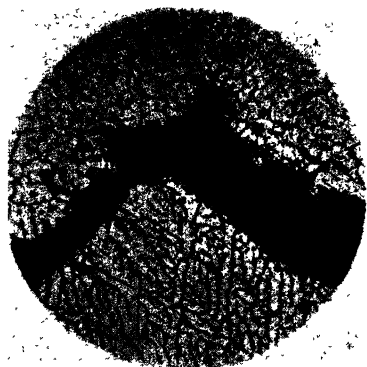
FIG. 5.—SAME AS FIG. 4; ETCHED WITH NaOH.   × 75.
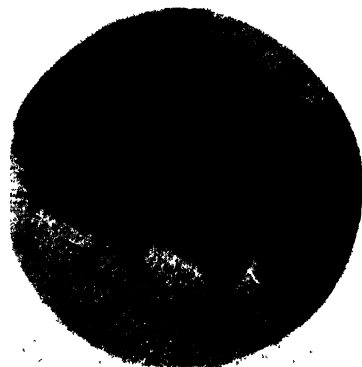
FIG. 6.—SAME AS FIG. 4; ETCHED WITH NaOH.   × 225.

alloy.   Fig. 7 shows a fine crack *a* and Fig. 8 shows a draw *b* on the opposite side at the juncture of a thin and a thick section.   Fig. 9 shows that the tendency of the crack is to be intergranular largely, although

parts of it appeared to be intergranular also. Fig. 10 shows the microscopic appearance of the crack at one point.

Specialty Casting

Fig. 11 shows the appearance of the crack a in a piece cut from the top of a specialty casting made of 92:8 Al-Cu alloy, while Fig. 12 shows a part of the crack in a polished and etched section. This crack did not
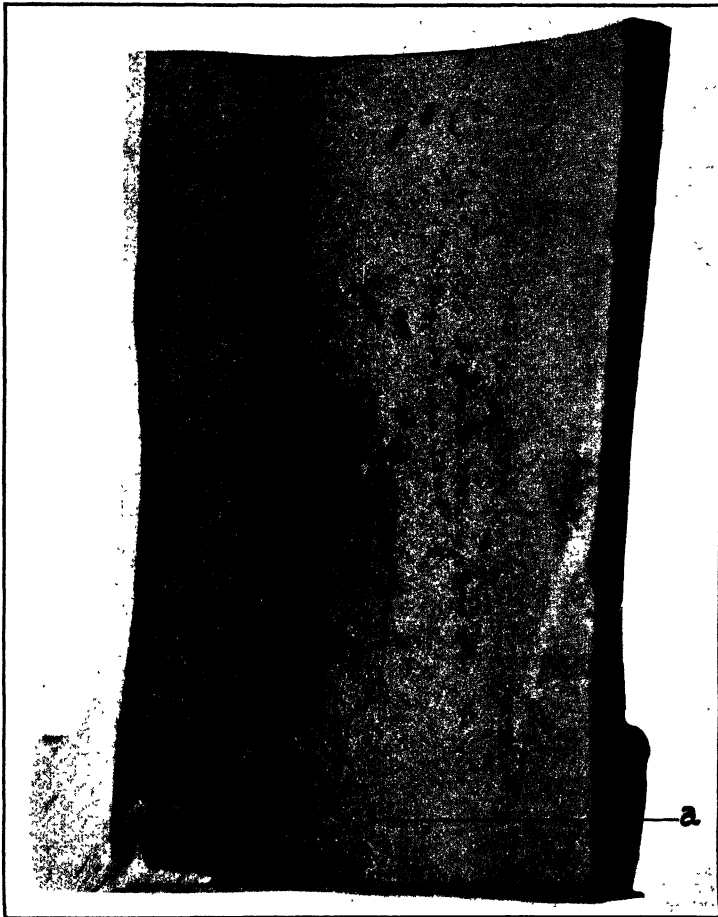
FIG. 7.—CRACK IN CRANKCASE. × ABOUT 1½.

follow the grain boundaries and the path of the crack was quite indiscriminate, being both intergranular and intragranular, with no particular tendency toward any particular path. This particular crack was attributed to hanging in the mold, because of too hard ramming of the sand around the pouring gate.
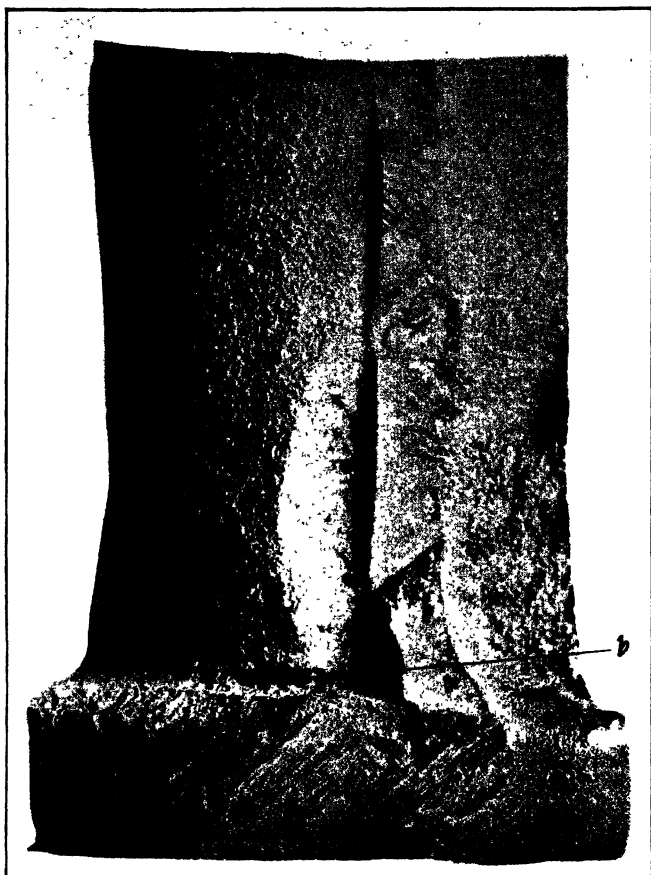
FIG. 8.—DRAW IN SAMPLE FROM CRANKCASE; OPPOSITE SIDE OF FIG. 7.   × ABOUT 1½.

FIG. 9.—CRACK SHOWN IN FIG. 7; MACROGRAPH; ETCHED WITH NaOH; OBLIQUE ILL.   × 3.
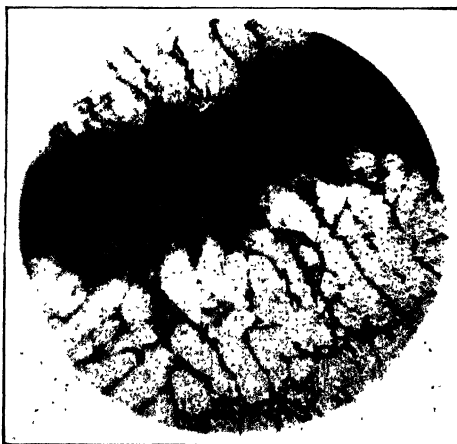
FIG. 10.—SAME AS FIG. 9; ETCHED WITH NaOH.   × 225.

FIG. 11.—CRACK IN SPECIAL CASTING; ½ ACTUAL SIZE.
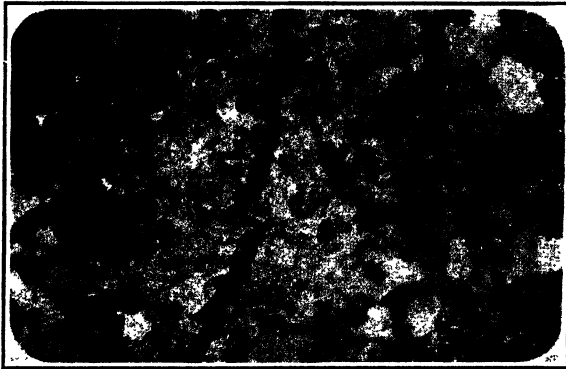
FIG. 12.—PART OF CRACK SHOWN IN FIG. 11; ETCHED WITH NaOH; OBLIQUE ILL. × 4.

FIG. 13.—CRACK IN CRANKCASE; ETCHED WITH NaOH; OBLIQUE ILL. × 8.

Cracked Crankcase

A sample taken from a crankcase for a twelve-cylinder motor, made of the 91:8:1 Al-Cu-Fe alloy, approximately is shown in Fig. 13. No definite cause could be ascribed for this particular crack. Fig. 14 shows a tenuous crack in the interior of the section, corresponding to the lines in Fig. 13. The gross grain size of the alloy was large, and the crack had a fairly pronounced tendency to follow an intergranular path.
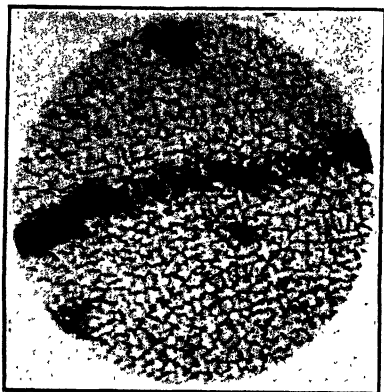
FIG. 14.—TENUOUS INTERNAL CRACK (CORRESPONDING TO SMALL BLACK LINES OF FIG. 13); ETCHED WITH NaOH. × 35.
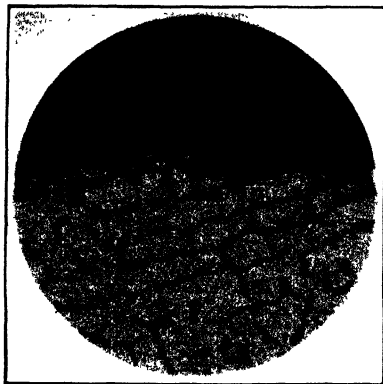
FIG. 15.—STRUCTURE AT EDGE OF CRACK IN FIG. 13; ETCHED WITH NaOH. × 150.
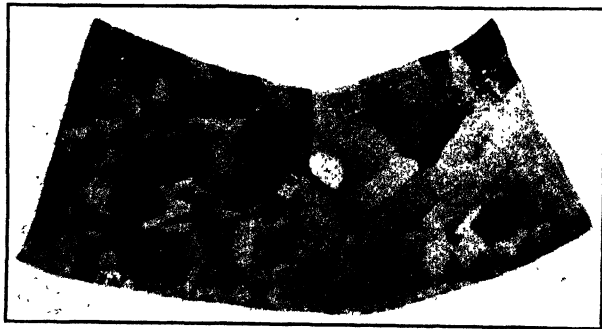
FIG. 16.—INTERGRANULAR CRACK IN CHILL-CAST 92:8 AL-CU ALLOY; ETCHED WITH NaOH; OBLIQUE ILL. × 3.

Fig. 15 shows the edge of the crack in Fig. 13 at higher magnification; at this point, the crack was fairly smooth and ran parallel to the eutectic stringers.

Miscellaneous Cases

A number of experiments have been made to produce cracks at will in various alloys under definite conditions, but principally by pouring

an annular-ring casting in a chill mold around a metal core. The complete results of these experiments are not yet in hand, but a few of the results show the tendency toward intergranular cracking. Although a crude method for determining the cracking tendency in different alloys, pouring a casting in the form of an annular ring is one method of determining the die-casting[16] qualities of aluminum alloys. It has been
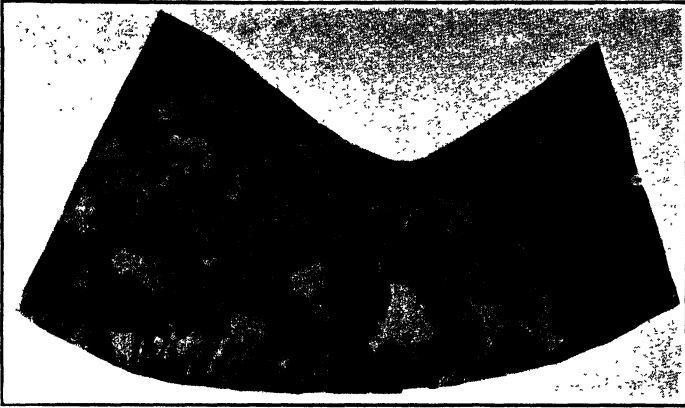
FIG. 17.—INTERGRANULAR CRACK IN CHILL-CAST 85 : 3 : 12 AL-CU-ZN ALLOY; WITH ETCHED NaOH; OBLIQUE ILL. × 3.
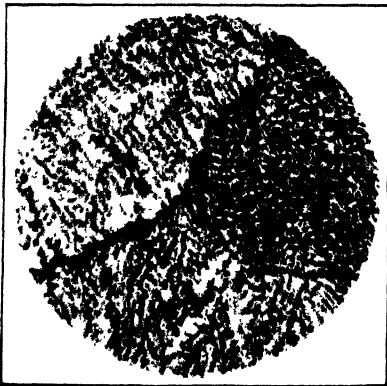
FIG. 18.—CRACK SHOWN IN FIG. 17; WITH ETCHED NaOH. × 75.
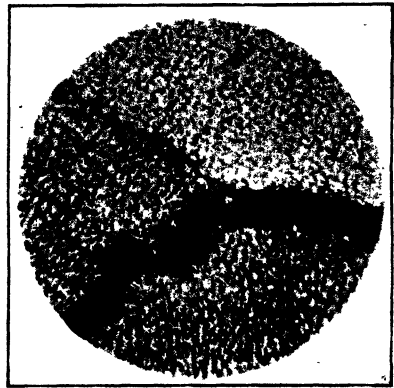
FIG. 19.—SAME AS FIG. 18, BUT AT ANOTHER PLACE; ETCHED WITH NaOH. × 75.

shown by preliminary experiments that the tendency to crack under this test varies with the alloy used and with the pouring temperature for a given annular-ring mold.

Fig. 16 shows an intergranular crack in the 92:8 Al-Cu alloy poured in a small annular-ring mold at 900° C. Fig. 17 is a macrograph of a

16 C. Pack: Die Castings and their Application to the War Program. *Trans.* (1919) 60, 577–586.

similar casting poured in the 85:3:12 Al-Cu-Zn alloy at 870° C.; the intergranular path of the crack is quite plainly shown. Figs. 18 and 19 are micrographs of two points in Fig. 17 showing that the path of fissure follows the boundaries of the large grains of the alloy, as indicated by the differential etching.

Discussion

In considering the occurrence of cracks in aluminum-alloy castings, the question naturally arises: Is the crack intergranular or intragranular? On theoretical grounds, it would be expected that a crack caused by stresses at high temperatures would follow the grain boundaries both in eutectiferous and in solid-solution alloys. While it is not necessary here to deal at length with the freezing of alloys of different types, in the case of an eutectiferous alloy a sufficient stress at high temperatures will cause fracture through the eutectic areas. At a sufficiently high temperature, the eutectic will be liquid when the matrix of the alloy is still solid, and under stress the fracture will occur through the weak liquid. Since eutectics are brittle, it would be expected that the fracture would take place through them rather than through the matrix owing to contraction or to stresses at lower temperatures.

In the case of pure metals and solid-solution alloys, cracks would be expected to follow the grain boundaries, i.e., go through the amorphous cement, in the case of stresses at high temperatures, viz., at temperatures exceeding the equi-cohesive temperature.[17] At lower temperatures, the fracture would be intergranular because the amorphous metal is stronger than the crystalline; but above the equi-cohesive temperature the amorphous phase is weaker than the crystalline phase, and consequently cracks due to solidification shrinkage will be intergranular. While at the ordinary temperature,[18] the amorphous cement may be regarded as hardening and strengthening the grain boundaries (and consequently the mass of metal), at a sufficiently high temperature the strength of the cement is less than that of the grains of metal. Where a crack is found assuming an indiscriminate path, it may be due to rupture at the equi-cohesive temperature, i.e., at that temperature where the resistance of the cement is the same as that of the grains.

It may be said that in the case of cracks originating from stresses at a low temperature, cracking will be less likely to occur in fine-grained metals than in coarse-grained ones. Fine-grained metals and alloys consist of a greater proportion of the amorphous phase and a lesser proportion of the crystalline phase than do coarse-grained ones. The

[17] Zay Jeffries: Amorphous Metal Hypothesis and Equi-cohesive Temperatures, *Jnl.* Am. Inst. of Metals (1917) 11, 300–323.

[18] W. Rosenhain and J. C. W. Humfrey: Tenacity, Deformation, and Fracture of Soft Steel at High Temperatures, *Jnl.* Iron and Steel Inst. (No. 1, 1913) 87, 219–268.

former are consequently stronger and, in general, less likely to crack. On the other hand, the cracking tendency will be greater in fine-grained metals than in coarse-grained in the case of stresses at high temperatures; *i.e.*, above the equi-cohesive temperature, because the amorphous phase is weaker than the crystalline phase at the higher temperatures.

Some of the illustrations show that the grain size was quite large and that the cracks passed along the boundaries of the large grains. In the eutectiferous 92 : 8 Al-Cu alloy, it is not to be supposed that the eutectic is segregated entirely at the boundaries of the large (macroscopic) grains, but each of the large grains will be found, on microscopic examination, to consist of a number of smaller grains surrounded by the eutectic. A crack may pass through a large grain and still be intergranular with respect to the small grains within the large grains (see Figs. 9 and 10). This etching phenomenon in the case of the light aluminum alloys has been described elsewhere.[19] The metallography of cracking has been treated only briefly in the present paper, but it is intended to discuss this more fully in a later communication.

METHODS FOR PREVENTION OF CRACKS

Of course, it is not to be expected that cracks, as a defect, will entirely disappear by the observance of the precautions here given, but they may be largely eliminated. The contraction of volume is a fixed constant for a given alloy, so alloys that have an excessively large contraction in volume should be avoided for the production of complicated castings or those of intricate design; for instance, large and complicated crank-cases. Of course, for some simple castings, alloys with high shrinkage may be employed satisfactorily. The contraction in volume may be controlled, within limits, by the correct use of risers and chills, so that a given alloy is not necessarily unsuitable for use in complicated castings simply because it has a high shrinkage.

Both the contraction in volume and the physical properties at high temperatures are dependent on the composition of the alloys, and alloys should be chosen, on the basis of adequate test data, that properly fulfill the requirements. The quality of the melting charge should be controlled by chemical analysis; much of the difficulty experienced with cracking comes from inadequate control of the melting stock.

Castings should be designed correctly, from both the foundry and the engineering standpoints. So long as complicated and intricately designed castings must be made, cracks cannot be wholly eliminated. From the foundry standpoint, the simpler the design, the less difficulty is experi-

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[19] R. J. Anderson: Metallography of Aluminum Ingot, *Chem. & Met. Eng.* (1919) **21**, 229–234. See also D. Hanson and S. L. Archbutt: Micrography of Aluminum and its Alloys, *Jnl.* Inst. of Metals (1919) **21**, 291–304; discussion, 313–315.

enced in production; it would seem as though many castings are made needlessly intricate. Methods of molding and the use of chills and risers are variable factors that depend on the casting to be made; it is not possible to formulate any definite rules for them. The expedient of experimental molding, as suggested by Traphagen,[20] is valuable in determining the most suitable method for molding a given casting, with particular reference to gating, risers, and chills. Too hard ramming should be avoided, and the required hardness of molds may be determined experimentally where experience is an insufficient guide. The gates should be of ample size and attached correctly to the casting; the use of horn gates, where possible, is recommended[21] for they permit the metal to enter the mold with the least disturbance.

Much of the difficulty experienced with cracking is traceable to the cores and to faulty core-room practice; too much emphasis cannot be placed on the necessity of using cores of the correct hardness.

Many cracked castings could have been saved if they had been chilled properly; also, if the risers had been correctly placed. While many simple castings do not require risers, they must be used on most large and complicated castings so that certain parts of the castings may be fed with liquid metal.

The melting temperatures should be low; overheating in the furnace should be avoided. Open-flame furnaces should be run with a non-oxidizing atmosphere in order to avoid the formation of much aluminum oxide, and consequently occluded dross in the alloy. Skimming should be adequately and carefully done and the melting practice should be conducted as cleanly as possible. Foundry-floor sweepings should not be charged into the furnaces unless sieved, and foreign materials should be kept out of the furnace charges. The pouring temperatures should be as low as is consistent with filling the mold readily with metal; high pouring temperatures should be avoided. This presupposes pyrometric control.

The observance of these several precautions, together with proper supervision and the study of each cracked casting followed by remediable steps taken to prevent the occurrence in subsequent production, should go a long way to eliminate cracks as a serious defect in the founding of light aluminum-alloy castings.

DISCUSSION

E. H. Dix., Jr.,* Dayton, Ohio (written discussion).—In his first paragraph, the author states that a crack is due "to fracture of the alloy

[20] H. Traphagen: Value of the Scrap Pile, paper before the Am. Foundrymen's Assn., Philadelphia meeting, September, 1919.

[21] J. W. Collins: Suggestions for Making Aluminum Castings. Paper before Detroit Foundrymen's Assn., 1914; abstract in *The Foundry* (1914) 42, 67–68.

* Chief, Metals Branch, Engineering Division, Air Service.
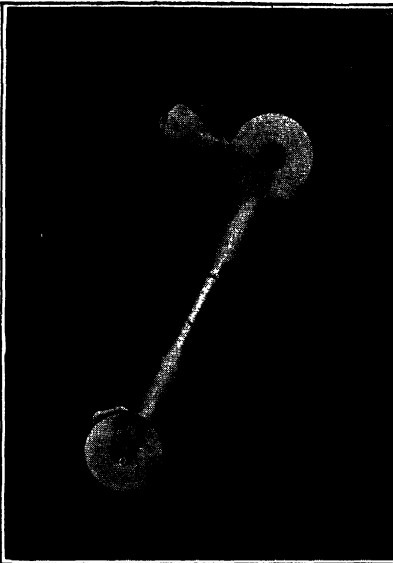
8 PER CENT. COPPER, BALANCE
ALUMINUM.
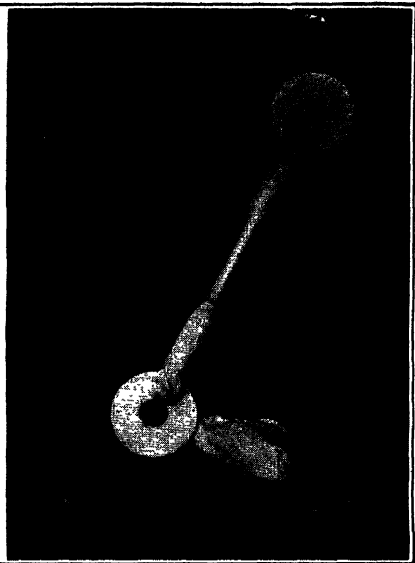
10 PER CENT. COPPER, BALANCE
ALUMINUM.

7 PER CENT. COPPER, 1 PER CENT.
TIN, BALANCE ALUMINUM.
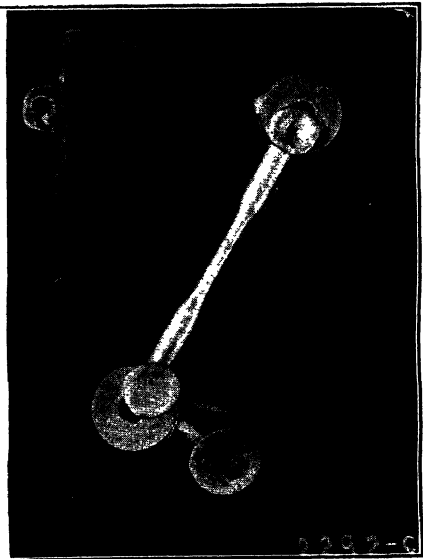
SILICON-ALUMINUM ALLOY.

FIG. 20.—HOT-SHORTNESS TESTS.

resulting from the stress set up by the contraction in volume on passing from the liquid to the solid state." In his third paragraph, he states that "patternmaker's shrinkage is the total contraction in volume (including liquid, solidification, and solid shrinkage) that occurs when an alloy is poured into a mold."

The writer obtained considerable information in regard to the first point by a crude hot-shortness test made in this laboratory on all experimental alloys. In this test, bars are cast with enlarged ends to fit around two steel pins rigidly held 12 in. apart; Fig. 20 shows the method in detail. As soon as the metal has set the cope is lifted off and the sand
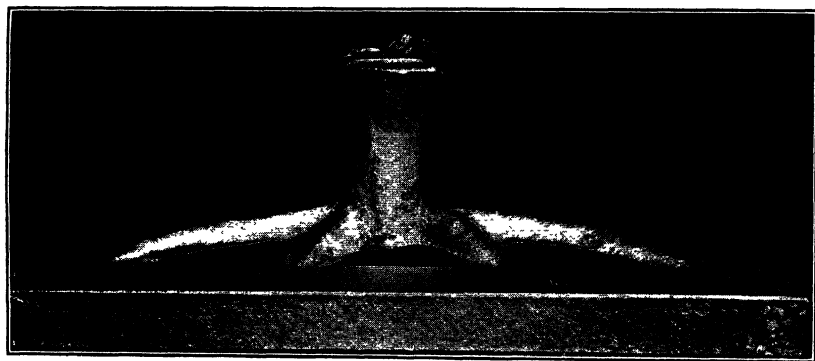
FIG. 21.—METHOD OF GATING SHRINKAGE BAR.

scraped away. The test bar is always solid and gives no indication of a crack until several minutes later; the time varies with the alloys under test. It is obvious that in this case the liquid shrinkage can have no effect on the tendency to crack because liquid metal is fed into the mold by the two risers. The solidification shrinkage does not have much effect in this particular case, so that the solid contraction is practically the sole cause of the cracking. In actual castings, it is not always possible to feed the casting with liquid metal as well as in this case; therefore the

TABLE 5.—*Results of Tests to Determine Patternmaker's Shrinkage*

	SHRINKAGE, INCHES PER FOOT
Commercial Al ingot......................	0.210
8 per cent. Cu-Al alloy....................	0.187
10 per cent. Cu-Al alloy..................	0.182
12 per cent. Cu-Al alloy..................	0.177
10 per cent. Zn-Al alloy..................	0.195
15 per cent. Zn-Al alloy..................	0.187
13 per cent. Si-Al alloy..................	0.170
4 per cent. Cu, 3 per cent. Si-Al alloy......	0.163
7 per cent. Cu, 1 per cent. Sn-Al alloy....	0.189
2 per cent. Cu, 1.3 per cent. Mn-Al alloy...	0.204

solidification shrinkage may have more effect, but it is hard to believe that the liquid shrinkage enters the problem at all.

To determine the patternmaker's shrinkage, the writer conducted some tests in the same manner as those given in Table 1, a 1-in. square test bar being cast in green sand between graphite templets set accurately 12 in. apart. The method of gating these bars is shown in Fig. 21. The results obtained on some typical alloys, the pouring temperature in every case being approximately 1300° F., are given in Table 5.

These results are much higher than those indicated in Table 1, which may be the result of different pouring temperatures, difference in the mass of metal, and perhaps of the method of gating. But there is comparatively little difference in the actual shrinkage of the various alloys as obtained by this method, which agrees with the results given in Table 1. Here again it appears that the shrinkage obtained in this test is primarily due to solid contraction, the liquid shrinkage not entering at all, and the solidification shrinkage only slightly, if at all. In other words, this shrinkage can be calculated as the inverse of the coefficient of expansion of the alloys. For instance, Dittenberger's[22] formula for the coefficient of expansion of pure aluminum,

$$\frac{\Delta 1}{10} = (23.536t + 0.017071t^2) \times 10^{-6}$$

when applied to the temperature range from 650° C. to 20° C., gives a contraction of 0.249 in. per ft., which is considerably larger than the shrinkage shown in Table 5 for aluminum ingot. This formula admittedly gives rather high results and the shrinkage is undoubtedly influenced by the percentage of impurities present. Thus, roughly, the shrinkage given in Table 5 is merely the inverse of the coefficient of expansion of the alloys.

The usual patternmaker's shrinkage for aluminum alloys is $\frac{5}{32}$ in. per ft., or approximately 0.156 in. per ft. This is much lower than any of the values in Table 5 and is also lower than that obtained by the coefficient of expansion formula. It has been found through practical experience, however, that this allowance for shrinkage gives castings of approximately the correct size. To the writer, this is merely an indication that free shrinkage of the alloy is hindered by the shape of the castings, cores, molding, sand, etc. In some cases camshaft housings, approximately 6 ft. long, were of variable lengths though casting from the same pattern, caused perhaps by variations in pouring temperature and molding conditions.

Hence, to sum up, the greatest cause of cracking is the high contraction shrinkage of the aluminum alloys. There may be cases, where heavy sections are joined to very thin sections, where the solidification

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[22] Bureau of Standards, *Cir.* No. 76.

shrinkage also is a contributory cause to cracking, but in general where cracks are due to hard cores, or molds, the cause of the crack is solid shrinkage.

The writer agrees with the author that test-bar results of tensile strength and elongation, even at elevated temperatures, are not any criterion of a tendency for aluminum alloys to crack. For instance, the last alloy given in Table 5 has good strength and elongation at high temperatures, yet, because of its high shrinkage and perhaps other causes, it is very liable to crack. Fig. 20 gives an indication of the relative susceptibility of the several alloys to cracking. The 7 per cent. copper, 1 per cent. tin, balance aluminum alloy, which is admittedly weak at high temperatures, has pulled farther apart than the other alloys; whereas, the silcion-aluminum alloy has not cracked at all. Further investigations have shown that the alloys of silicon and aluminum, with and without copper, may have their composition varied through wide ranges and yet withstand this test without cracking. However, by the use of an impure ingot containing high iron, this alloy immediately became susceptible to cracks.

R. J. ANDERSON.—The definition of shrinkage given is the accepted one. Mr. Dix apparently does not attribute the cracking of aluminum alloys to the slight differences in linear contraction showed by pouring test bars in a mold; the author does, however. The ordinary 90:10 aluminum-copper alloy, where the contraction may be 1.25 per cent., will crack on being poured into a casting under given conditions, while the 87:13 aluminum-silicon alloy, where the contraction is about 1.10 per cent., will not crack. Rosenhain and others allege that the slight differences in linear contraction among the various light aluminum alloys cannot account for their different behavior on casting in the foundry; the author thinks that more importance should be attached to these small differences in contraction.

Edwards[23] has shown that the contraction in volume of the 92:8 aluminum-copper alloy from the beginning to the end of freezing—over the temperature interval 636° to 540° C.—is about 7 per cent. The liquid shrinkage of this alloy on cooling from 1,000° C. (density, 2.434) to 700° C. (density, 2.524) is about 3.5 per cent. The solid shrinkage from the freezing point (540° C., where freezing is complete) to room temperature is taken as about 0.156 in. per ft., or 1.30 per cent. The total contraction in volume on passing from the liquid state at any temperature to the solid state, say at room temperature, is the algebraic sum of the liquid shrinkage plus the solidification shrinkage plus the solid shrinkage.

[23] J. D. Edwards, Mechanism of Solidification of a Copper-aluminum Alloy. *Chem. & Met. Eng.* (1921) **24**, 217–220.

The test for patternmaker's shrinkage, or linear contraction on pouring into a mold, is useful and can be carried out with much more simplicity than the hot-shortness test devised by Gillett and Skillman.

The figures given for the linear contraction of some light aluminum alloys were determined by pouring into graphite molds, and subsequently about fifty alloys were tested in both graphite and sand molds. In the case of the bars poured in sand, bars were cast between the faces of a cast-iron templet exactly 12.000 in. apart and using three sizes of bars, viz., 0.5 in. square, 1.0 in. square, and 0.5 by 1.0 in. in cross-section. The data derived from these tests will be presented later, but it should be stated that the linear contraction is a function of a number of factors among which the pouring temperature and size of section are of predominant importance.

When making up patterns for the production of aluminum-alloy castings, the general figure 0.156 in. per ft. is used as the pattern allowance. If a given alloy is to be run for any lot of castings, it would be profitable to make linear-contraction tests over a wide range of section size and pouring temperature before making the patterns.

E. H. DIX, JR.—The author did not get my point, which is that 0.156 in. is so much smaller than the value obtained from the coefficient of expansion formula that the actual solid contraction more than takes care of the usual patternmaker's shrinkage.

R. J. ANDERSON.—That is true.

H. B. SWAN, Detroit, Mich.—In considering what is commonly known as 87:13 silicon alloy, you have practically a eutectic structure in silicon and aluminum and less shrinkage and the metal is stronger than in the No. 12. In an alloy of that type does not less strain develop between thin and heavy sections because the eutectic structure gives an alloy that cools almost at one point?

R. J. ANDERSON.—That is true. The 87:13 silicon alloy is in the vicinity of the eutectic, which lies at about 10.5 per cent. in the aluminum-silicon system. In the case of the 92:8 aluminum-copper alloy, where there are too many risers in the mold, the alloy cannot contract normally but is prevented from moving, and cracking occurs at high temperatures, i.e., in the solidification range.

T. D. STAY, Cleveland, Ohio.—One reason why the silicon alloys do not crack is they are not hot short at a point just under the solidification point. If there are a number of risers, which tend to prevent linear shrinkage, the alloy stretches a little instead of shrinking at a point just under solidification, and therefore eliminating strains and cracks. No. 12

aluminum is very hot short; in fact, a riser may be broken off easily at a point just below solidification; but silicon alloys may be hammered for several minutes and then sections will be badly dented without cracking.

JOHN A. LANGE, Western Springs, Ill.—When rolling metal containing silicon, the rollers may have to give one or two extra passes; but the silicon does not cause trouble in the melting, casting, or rolling. The trouble comes in the annealing operation, as this alloy must be given a short quick anneal, otherwise the sheet is harder and it does not act as well under the drawpress. That is my objection to the silicon. Of course, if we know we are getting metal with a high silicon content, we can arrange to handle it separately in our annealing furnace. But I would prefer not to have the metal in the plant because a large percentage of scrap is always coming back and I do not want to get it mixed with the other metals.

Gas Absorption and Oxidation of Non-ferrous Metals

By B. Woyski* and John W. Boeck,† Buffalo, N. Y.

Many writers, in discussing defects caused by oxidation and gassing of bronzes and red brasses advocate substantially the same cure for both. But from its nature, oxidation cannot take place if there is an excess of fuel in the furnace, while the gases absorbed by some alloys are products of incomplete combustion of the fuel. Hence the metal can be either oxidized or gassed, but it is doubtful whether these actions can take place simultaneously.

Oxidation can be easily overcome by the use of deoxidizers or it may be avoided by protecting the molten metal with suitable fluxes. Absorption of gases is more difficult to overcome: the regulation of the air and fuel, so as to produce complete combustion, and the protection of the bath by mineral fluxes are the most important factors. Gas expulsion, resulting in gas holes, porosity, unsoundness, and swollen or cauliflower-headed risers, is said to have several causes. One writer[1] lists the most common causes in the order of their importance, as follows: Superheated metal, dirty or slaggy furnace conditions, using metals low in quality or those previously subjected to bad melting practice, poor grade of fuel, use of newly lined or damp ladles, indiscriminate mixing of scrap or illogical combinations of metal.

Superheating the metal usually occurs when the heat is ready to be poured but for some reason must be held in the furnace. In this case, a careful furnace tender tries to avoid overheating the metal by keeping a mild fire; that is, a reducing fire which, unfortunately, results in gassing the metal.

A dirty, slaggy furnace may result in gassing the metal by retarding the heating and prolonging the time necessary to bring the metal to the proper temperature, thereby increasing the chances of gassing in uncertain furnace atmosphere.

It is doubtful whether metals of low quality have anything to do with gassing molten metal, except, possibly in the case of metals containing

[1] R. R. Clarke: *Foundry* (1919) **47**, 121.

sulfur. The sulfur may produce gas holes as a result of a reaction with occluded oxides. The reaction between sulfur and copper oxide is reversible, depending on pressure and temperature.

Metal that has been subjected to bad melting practice, whether oxidized or gassed, can be brought back to normal conditions by proper furnace practice. In the case of oxidized metal, a deoxidizer or a flux is necessary. In the case of gassed metal, a strongly oxidizing furnace atmosphere of relatively high temperature, so as to produce quick melting, will correct the gassed condition. This has been proved many times. In one case, when, on account of a change in the furnace tender, several heats of red brass were gassed, all the castings were returned to the furnace and melted under the conditions just described, the resulting castings were entirely satisfactory, both as to fracture and microscopical characteristics.

The use of newly lined and damp ladles may be one cause of gassing. This may be ascribed to hydrogen, from dissociated steam, being absorbed and then liberated on freezing, or after action on the occluded oxides with the formation of steam at lower temperatures; however, the dissociation of steam at the pouring temperature of bronzes is negligible.

Indiscriminate mixing of scrap and the illogical combination of metals should never occur in any foundry, but we hardly believe that it would result in gassed metal.

The most important cause of gassed metal is the atmosphere of the furnace; with the proper regulation of air and fuel, it will not take place in the case of most non-ferrous alloys. Unfortunately, the appearance of the flame outside the furnace is not a positive indication that the right condition exists inside the furnace, especially in the case of open-flame furnaces. The burners should allow intimate mixing of the air and fuel before they reach the furnace; otherwise, soluble fuel vapors may play on the surface of the molten metal.

The appearance of a casting poured from gassed metal depends much on its form and size, nature of the alloy, and the pouring temperature. In the case of red brasses, the riser may not "go down" at all, or it may go down considerably and then, at a lower temperature "come up." The casting may show gas holes under the riser, which will be different from shrink holes, or it may be solid under the riser but will not machine smooth and the fracture will be discolored. In the latter case, almost any foundryman will pronounce it oxidized. In the case of thin sections of zinc bronze, the casting may machine smooth but, if fractured, will show two layers that are easily separated; this condition also is often ascribed to oxidation. We have heard that this last condition may be overcome by hot pouring; this would indicate that the defect is a result of

shrinkage due to cold pouring but it does not disprove that the defect is not caused by gases absorbed by the metal, because to get the metal very hot it is necessary to use a hot (in other words, an oxidizing) fire, resulting in quick melting with a minimum chance for gassing.

We have strong evidence that the discolored fracture is not always due to oxidation. While making bearings of leaded gun metal, it was noticed that the castings showed discolored fractures. The metal was melted in a Detroit arc furnace, sealed air tight, so that any oxygen inside the furnace was used very rapidly by the electrodes with the formation of carbon monoxide; the oxidation of the metal under these conditions was inconceivable. The addition of phosphorus, magnesium, mineral fluxes, and charcoal did not help. The same alloy, melted in the Bailey electric-resistance furnace, when poured into the same kind of molds made by the same molder, produced castings of normal fracture. The Bailey furnace is not air tight; there is an inflow of air at the bottom, through the spout under the door, and an outflow of gases and zinc vapors at the top. If the alloy melted is of high zinc content, the pressure of the zinc vapors within the furnace is sufficient to prevent the inflow of air. Thus, in the Bailey furnace there is a greater opportunity for the oxidation of alloys of low zinc content than in the Detroit arc furnace; consequently, we attributed the trouble to the absorption of carbon monoxide, which exists, in a sealed arc furnace, under pressure. The vent was then left open and the trouble disappeared; castings exhibited a clean normal fracture. Later, another user of the Detroit arc furnace stated that if the furnace is air tight during melting, trouble will be experienced. The nature of the trouble was not explained, but it cannot be supposed to be oxidation.

It was also noticed on several occasions that discolored fractures appeared in bearings poured with metal melted in open-flame furnaces with decidedly reducing flames. With a decidedly oxidizing flame and a mineral-flux protection, castings of the same metal had normal fractures.

The so-called "tin sweat," or the exudation of a tin-rich alloy from bronze castings is also due to the liberation of gas absorbed in the furnace. This often happened in the past, but does not occur in our practice, since the detrimental effect of a reducing fire was realized. The tin sweat occurs most frequently in the riser. This may go down at first and then when almost solid an exudation takes place, either in the cavity of the riser, or at its edge, if the riser did not go down at all. Fig. 1 illustrates the first case and Fig. 2 the second. Fig. 1 represents a longitudinal section of the riser of a large casting containing 88 per cent. copper, 5.5 per cent. tin, 1 per cent. lead, and 5.5 per cent. zinc. The gray material is the tin-rich alloy and the black spots gas holes. In this case the evolution of a combustible gas, taking fire in the cavity of the riser on contact with the air, was seen plainly in partial darkness.

FIG. 1.—LONGITUDINAL SECTION OF RISER OF A LARGE CASTING, SHOWING EXUDATION OF TIN-RICH ALLOY.

FIG. 2.—LONGITUDINAL SECTION OF RISER OF A MEDIUM-SIZED CASTING.

Fig. 2 is also a longitudinal section of the riser of a medium-sized casting containing 84 per cent. copper and 16 per cent. tin.

In the case of aluminum bronze, various metallic deoxidizers are used to insure a sound casting. Their use seems to be illogical; defects in aluminum bronze are caused by gas holes or alumina inclusions. It is well known that alumina forms while pouring, and in the mold if there is any agitation. No deoxidizer will have the opportunity to reduce the alumina formed in the mold; if it does, the product of deoxidation, whether it be a gas or a solid, will be trapped under the skin of the casting.

Anderson says[2] "there is now known no commercial method for the deoxidation of aluminum." The oxidation of aluminum bronze while melting is negligible and only on the surface. Any oxide stirred in the metal during mixing rises to the surface very readily. This has been proved by melting aluminum bronze in a crucible holding about 3 lb., stirring the oxides into the molten metal, and then allowing the metal to freeze fairly rapidly in the pot. The button was cut vertically and a section polished. On examination, under the microscope, no alumina was found in the metal except immediately under the top surface. A number of heats of aluminum bronze were poured in this foundry without any deoxidizer, and those castings where there was no agitation of the metal in the mold were free from alumina. All those in which, on account of the design or method of gating, the metal was agitated while filling the mold, invariably, on machining, showed the presence of alumina immediately under the top surface. But the use of deoxidizers in the manufacture of aluminum bronze is quite common, probably because some of them act to a certain extent as degasifiers.

Aluminum bronze is very susceptible to gassing; that is, the copper which is melted first is, of course, highly susceptible, and the addition of the aluminum has not a corrective influence; but the resulting alloy is less sensitive to gasification than pure copper itself. When gassed, this alloy poured in a sand mold will not go down in the riser and the casting will show pear-shaped and elongated holes. Aluminum does not remove the gases absorbed by copper. This was also observed by Moore and Beckinsale.[3] That the gases absorbed by this alloy are combustible, and most likely hydrocarbons or carbon monoxide, was indicated in a few cases where the riser was watched and minute bubbles of gas were seen to emerge from the still liquid metal and take fire. By using a decidedly oxidizing flame in oil furnaces, the gassing of aluminum bronze is avoided completely. In coke-fired furnaces, gassing of this alloy never happens, as the atmosphere is rarely, if ever, reducing but probably almost neutral or slightly oxidizing.

[2] U. S. Bureau of Mines *Tech. Paper* **241** (1919).
[3] *Jnl.* Inst. of Metals (1921) **25**, 219.

That Carpenter and Elam[4] have not found any appreciable difference in the volume of gases from sound and unsound castings does not prove that blowholes and porosity in unsound castings cannot be ascribed to the larger volume of gas dissolved by the metal. The explanation for not finding very much difference may be that, since gas that cannot be held by the solidifying metal results in gas cavities and films between the crystals, it is removed from the metal while evacuating the furnace tube prior to heating. Unsound castings, being leaky, cannot hold gases that are not in solution, not only when in a vacuum but even under atmospheric pressure. On account of the high diffusibility of gases at high temperatures, much of the dissolved gases is replaced by air right after the solidification of the metal. The amount of gas kept in solution would then be the same in sound and unsound metal. This also may explain the discolored fracture of unoxidized but gassed metal. The air diffused in hot unsound metal would naturally discolor the walls of the cavities.

It is not clearly understood what kind of gases cause molten copper to come up in the sprue and riser, when poured in a sand mold. As a rule, to prevent that and obtain a sound casting, a deoxidizer is used, although it is evident that gases and not oxides are directly responsible for the trouble. The question arises: Is it a gas absorbed in the furnace which is soluble in liquid copper and insoluble or less soluble in solid copper; or, is it a gas which is soluble in solid as well as in liquid copper, and only after reaction with the oxides present in the metal or formed during pouring, is changed to insoluble? In the first case, the use of a deoxidizer is useless, except where the metallic deoxidizer—by alloying with the copper—renders it a non-solvent for the gas while in a liquid state. In the second case, the use of a deoxidizer is logical. That the gases soluble in copper are not only hydrocarbons is evident from the fact that even in an electric arc furnace the copper absorbs a gas, which could be nothing else but carbon monoxide.

The unreliability of the results obtained on the solubility of gases in copper, by the various investigators, is illustrated by the statement of Bamford and Ballard[5] in a survey of the literature on this subject that "the question as to whether nitrogen, carbon monoxide and carbon dioxide are soluble in copper is answered in the affirmative and in the negative by equally strong schools of thought." All investigators agree that the solubility of hydrogen in liquid copper is considerable.

It may be contrary to the general opinion, but we believe an oxidizing condition in the furnace is less troublesome than a reducing, provided the bath is protected with a thin layer of mineral flux; also that it is more

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[4] *Jnl.* Inst. of Metals (1918) **19**, 155.
[5] *Jnl.* Inst. of Metals (1920) **24**, 156.

economical in time and fuel. The loss of metal is not appreciably greater. The mineral fluxes protect the surface of the metal and may replace deoxidizers in many cases, but to be certain of the removal of oxides, formed before the metal is melted, the metal must be stirred in order to bring the oxides into contact with the flux, because all of the oxides do not float. Aluminum oxide cannot be expected to float in molten aluminum, its specific gravity, 3.73–3.99, being greater than that of aluminum, 2.6. Lead oxide, with a specific gravity of 9.2–9.5, cannot float in any bronze or copper-lead alloy up to 26 per cent. lead. Tin oxide, having a specific gravity 6.6–6.9, floats to the surface of all bronzes, but not as readily as zinc oxide, the specific gravity of which is 5.78.

Two small pots of copper-lead alloy (90 per cent. Cu, 10 per cent. Pb) were melted side by side in a gas-fired furnace and the metal badly oxidized; the metal was stirred often in order to mix the oxides formed; then one pot was fluxed with a mineral flux and both were left in the furnace to cool slowly. Both buttons were cut vertically and one half of each was machined and polished; the other half was fractured. The fluxed metal did not show any trace of oxide, while a red powder collected on the tool when machining the unfluxed one. Sufficient of this powder was collected, passed through a 200-mesh sieve to remove minute particles of metal, analyzed, and was found to contain Si 0.8 per cent., Cu 52.9 per cent., Pb 30.9 per cent. Nothing else could be detected, so the silicon, copper, and lead were calculated to oxides with this result: CuO 66.1 per cent., PbO 33.0 per cent., SiO_2 1.7 per cent. The machined unpolished section, when examined under the microscope at low power, exhibited holes and fissures from which red powder could be picked. After polishing, no oxide could be detected nor could the oxide cavities be distinguished from the lead. The fracture of the fluxed metal was clean and of normal color; the fracture of the unfluxed metal was discolored, but examination under a binocular microscope failed to reveal any oxide pockets in the fracture. After a long search some gray globules, which seemed to be lead, were touched with a pin and found to be red powder balls enveloped with a gray skin. Further examination revealed large numbers of them, which could be detached from the metal without bursting the skin.

In similar experiments with copper-tin alloys, it was found that with slow cooling the tin oxide floats to the surface in the form of crystals; the slower the cooling the larger the crystals. Theoretically, the tin oxide cannot exist in the presence of zinc, but in a number of cases we found tin-oxide crystals in red brasses. It is identified by its color, behavior with etching reagents, and its crystalline form. Zinc oxide is infusible at the temperature of molten bronze, and, therefore, could not attain crystalline form; evidently the reaction between tin oxide and zinc is not quantitative.

Our experience with monel metal is insignificant, but the detrimental effect of reducing fire on it was also noticed. We tried to follow the instructions of the monel-metal manufacturers, but obtained indifferent results. They state that oxidation must be avoided, but we find that sound castings can be made only by avoiding both oxidation and gassing. This can be accomplished by melting in an oxidizing atmosphere and protecting the metal with a mineral flux.

The three common causes of defects in a brass foundry, oxidation, gassing, and shrinkage, may produce effects very similar; consequently, the latter two are often erroneously taken for oxidation. The effect, that is the discolored fracture, in every case is due to oxygen; directly, when the metal is oxidized in the furnace, and indirectly, in the case of gas or shrinkage. Gas and shrinkage cause porous metal and intercrystalline fissures, so that air is admitted, which results in the formation of oxide films on the walls of the cavities and fissures.

On the other hand, the normal color of fracture is not a positive indication of the absence of oxides. Even in the same casting, the fracture is discolored at one point and normal at some other; while under the microscope any part of the casting exhibits the same unsound condition and presence of oxides. This was observed in solid bars from 1 to 3 in. in diameter and 12 in. long. The bars were gated into the riser located at one end. The fracture one-third distance from the riser was discolored and the fracture two-thirds the distance from the riser was normal. The polished cross-sections cut near fractures exhibited, under microscope, the same quantity of oxide crystals and films. No explanation has been found for this lack of correlation between the appearance of fracture and of polished section.

From these practical observations it can be concluded that: The most important cause of gassing molten metal is the furnace atmosphere. Gassed metal can be brought back to normal condition by remelting in an oxidizing atmosphere. A reducing atmosphere is more troublesome than an oxidizing one. Defects caused by gas or shrinkage may have a similar appearance to those caused by oxidation and are often mistaken for such.

DISCUSSION

PAUL D. MERICA, New York, N. Y.—In the case of copper-base metal, a clear distinction can be drawn between oxidation and gasification. If the metal is oxidized, it contains either zinc oxide (brasses) or tin oxide (in the bronzes). To introduce soluble gases into the metal, it must be done through the furnace atmosphere; in metals containing carbon, such as monel metal and steel, that is not necessary. The effect of oxidation is ultimately the same as that of gasification, as the first action of oxidation on metal containing carbon is to produce carbon

monoxide. Therefore it is not so easy to draw a distinction between oxidation and gasification in the latter case.

R. J. ANDERSON, Pittsburgh, Pa.—Among other conclusions, the authors seem to insist that we melt in oxidizing atmospheres rather than in reducing atmospheres. We have all been brought up to believe that a reducing atmosphere is the kind to use for melting non-ferrous alloys—that is the orthodox practice—and if there is any reason for changing I will be glad to learn it. On page 862, and elsewhere, the authors state that an alloy which has been subjected to bad melting practice, whether oxidized or gassed, can be brought back to a suitable state by proper furnace practice, from which it is assumed that if an alloy is burned or otherwise ruined in melting, it can be remelted and restored to a good condition for casting. Proof of this fact should have been given.

We have been talking for years about so-called oxidizing, neutral, and reducing atmospheres in furnaces, although in very few instances have any actual analyses been made of furnace atmospheres. It is technically incorrect to state that an atmosphere of any given composition is oxidizing, neutral or reducing without stating also all the conditions, e.g., temperature, actual chemical composition of the gases in the atmosphere, and the metal or alloy being melted. In an open-flame furnace, it is possible to have an atmosphere containing up to 14 per cent. CO_2 or CO. The importance of actual analyses of furnace atmospheres has been pointed out in a series of articles[6] covering briefly the results of an investigation in the Bureau of Mines.

There is an apparent contradiction in the paper: In the first paragraph of page 863, it is stated, "We have strong evidence that the discolored fracture is not always due to oxidation;" on page 868, it is stated, "The effect, that is the discolored fracture, in every case is due to oxygen." It is also stated on page 867 that "aluminum oxide cannot be expected to float in molten aluminum, its specific gravity, 3.73 to 3.99, being greater than that of aluminum, 2.6." The question of surface tension will enter into this case; if the aluminum oxide is not wetted by the aluminum, it can float on the surface of a liquid bath of aluminum under the influence of surface tension. A liquid bath of aluminum is always covered with a scum of aluminum oxide when exposed to the air or to a high-oxygen atmosphere.

BRUNO WOYSKI.—The terms oxidizing and reducing are used in their ordinary sense. The atmosphere is judged solely by the color of the flame, as the analysis of samples taken from various parts of the furnace varied to an extent sufficient to render them valueless.

[6] R. J. Anderson and J. H. Capps: Gases in Aluminum Furnaces and Their Analysis, Chem. & Met. Eng. (1921) 24, 1019–1021; and Constitution of Gas Atmospheres in Aluminum-alloy Melting Furnaces, Chem. & Met. Eng. (1921) 25, 54–60.

Where the authors say "We have strong evidence that the discolored fracture is not always due to oxidation," they use the term as it is generally used; that is, oxidation in the furnace. Later when they say "the discolored fracture in every case is due to oxygen," they differentiate as to whether it was in the furnace or due to the oxygen penetrating an unsound casting.

H. M. St. John,* Detroit, Mich.—I do not think, from the evidence submitted in the paper, we are justified in assuming that an oxidizing atmosphere is the right kind to use when melting bronze. I have been in touch with many people using Detroit furnaces, some of whom tried running with an open spout and also with the spout entirely closed. With the exception mentioned in this paper, I do not know of a case where a difference has been discovered in the quality of the metal, whether the furnace was open, partly open, or entirely closed. As a matter of fact, the authors seem to assume in the case of the arc furnace that if the spout is entirely open the atmosphere in the furnace will be oxidizing. Unfortunately no gas analyses are presented to show whether or not this might be true. But in several cases gas analyses have been made of the atmosphere of Detroit furnaces melting brass and bronze, and it has been found that whether the furnace is tightly sealed or operating with an entirely open vent, the difference in percentage of oxygen in the atmosphere is within the limits of experimental error. Apparently there is no difference. I do not mean, of course, with the door wide open, but with a wide-open spout. The atmosphere inside the furnace consists of nitrogen and carbon monoxide, practically nothing else. If the furnace is tightly sealed—and leaving aside for the moment the possible effect of zinc vapor or anything of that kind that may give added pressure— the pressure of carbon monoxide amounts to only a few ounces.

Carbon monoxide will dissolve in pure copper in the neighborhood of 2200° F. Copper heated below that point does not show pronounced gasification; copper melted above that point does show gasification, which is due to carbon monoxide. But when heated to 2200° and allowed to cool to about 2100°, copper does not show gasification; this is true regardless of whether the furnace is operated with a tightly sealed spout or is well vented.

The quantitative effect of carbon-monoxide pressure I know nothing about; I cannot state what it may be other than is indicated by the experience I have just mentioned, which perhaps is somewhat inconclusive. But I do not think perfectly venting the furnace will result in an oxidizing atmosphere.

Verne Skillman, Detroit, Mich.—I cannot conceive of running a brass furnace under really oxidizing conditions and do not want to say

whether it is good practice or bad. With a complete covering of flux I can see where we could possibly use an oxidizing atmosphere; but what would be the effect of running an oxidizing atmosphere in, say, a Simplex furnace, melting a bearing bronze containing 5 per cent. tin and 30 per cent. lead, heating up to 2200° or so, and pouring car brasses at 1800°. How many would be willing to run that alloy under oxidizing conditions? I would not want to be responsible for all the results that would possibly be obtained.

I do not believe that it makes much difference if the spout of an electric furnace is open or closed during melting; our practice is to have it partly open to relieve the pressure. If we do not relieve the pressure, it relieves itself through the electrode openings, heating the electrodes around the openings and causing trouble there.

WM. B. PRICE, Waterbury, Conn.—We were melting 1-ton copper billets in the Bennett electric furnace, using absolutely no flux. Later, a microscopic examination of the billets showed no evidence of any cuprous oxide, but the billets were full of gas holes, in some cases small, in others large. In running the furnace, we had some trouble with the electrodes, so that the copper was unavoidably superheated. My recollection is that the copper was heated to about 1400° C. To repair the electrodes, it was necessary to withdraw them from the bath; and to assist in cooling down the furnace, the charging door was opened so that the molten copper was exposed for over an hour, and was cooled to a relatively low temperature. The copper billet poured from this particular heat, weighing a ton, was absolutely sound from one end to the other, and showed no porosity or evidence of cuprous oxide.

We were not able to duplicate this billet because we tore down that particular furnace. Samples of the copper were sent to the Bureau of Standards and gas analyses made, which showed nitrogen, carbon dioxide, and carbon monoxide to be present. Unfortunately, the Bureau had some trouble with its vacuum furnace and has not been able to complete its analyses. I have always thought the reason for the very sound billet was that it was superheated and held at the proper heat in much the same way that steel is held in the ladle, which gave a chance for the expulsion of certain gases and an oxidizing atmosphere, which resulted in a sound billet with the proper pitch.

BRUNO WOYSKI.—We use an oxidizing atmosphere in preference to a neutral, because the latter, though the proper kind of atmosphere to use, is too difficult to maintain in practice. A strongly oxidizing atmosphere is not essential, except where the molten metal is extremely sensitive to free hydrocarbons; then it is necessary to insure rapid and complete combustion. We aim at a complete covering of the molten

bath by means of mineral fluxes, but from the nature of some alloys this is not essential.

The results of an oxidizing atmosphere are sound castings. Tin sweat and gas absorption are completely avoided, while with some alloys, for instance aluminum bronze, it is impossible to prevent gas absorption in a reducing atmosphere. The detrimental effect of free hydrocarbons is illustrated by our experience with the Bailey furnace, in which the resister trough is constructed by means of wooden forms. Before any metal is charged these forms must be burnt out; if these forms are not completely burned and we melt, say, a 90–10 bronze, we are sure to have tin sweat.

Under no circumstances do we use a reducing atmosphere. Melting with a reducing fire is slow and we know that most foundries are using oxidizing fires for this reason and are getting good results.

W. M. CORSE, Washington, D. C.—Some similar work on induction furnaces might help settle this question, as this type of furnace probably has as nearly a neutral atmosphere as any we can secure.

N. K. B. PATCH,* Buffalo, N. Y.—As the authors have said, the objective is to have as nearly neutral an atmosphere as possible but rather on the oxidizing side, the thought being that with a reducing atmosphere, the hydrocarbons from the fuel, if fuel oil is used, or CO in the case of an electric furnace, are not oxidized, so that they are soluble. Whereas, if they are oxidized by the slightly oxidizing atmosphere, the tendency to absorb such gases is obviated due to the fact that such fully oxidized gases are not soluble.

In the induction furnace, the condition is different. The temperature of the metal is produced by heat within and the action of the atmosphere can be almost entirely neutralized by a simple covering of flux; so that the effect of the atmosphere under such circumstances is an almost diametrically different problem from the one with the furnaces tried.

* Secretary and Works Manager, Lumen Bearing Co.

Temperature Problems in Foundry and Melting Room

JOHN P. GOHEEN,* PHILADELPHIA, PA.

CONSIDERABLE work has been done in developing a pyrometer to measure the temperature of molten brasses, bronzes, and aluminum in the crucible. On account of the high melting points and the amount of zinc and sulfurous fumes given off from molten brasses, the conditions are somewhat severe for a thermocouple of standard design; but satisfactory results are obtained with a portable pyrometer, 3 ft. (0.9 m.) of connecting lead wire, and a special 5-ft. nickel-nickel chromium thermocouple, composed of rods $\frac{1}{4}$ in. (6.35 mm.) in diameter. The lower 12 or 24 in., as desired, may be made detachable so that new tips may be connected at little trouble or expense. The portable instrument need not be of the extreme high-resistance type, as the movable parts of the galvanometer of a lower resistance instrument may be better constructed to withstand the continuous handling such an instrument receives. Platinum-platinum rhodium thermocouples, protected with porcelain and graphite tubes, have been used, but the lag in registering the temperature is too great.

It has been found that the range of temperature for various brasses is as follows:

Red brasses..............2200–2250° F. (1205–1234° C.)
Yellow brasses...........2250–2400° F. (1234–1318° C.)
Bronzes.................2300–2550° F. (1260–1400° C.)

A recent test with a portable pyrometer having a range of 2600° F. with a bare $\frac{1}{4}$-in. heavy exposed couple 5 ft. long and a handle attached at the upper end showed that the temperatures of molten aluminum under varying conditions in an oil-fired crucible furnace and one of the Simplex type were as follows:

	CRUCIBLE, OIL-FIRED FURNACE	SIMPLEX FURNACE
Temperature of metal in furnace.......	1540° F. (837° C.)	1530° F. (834° C.)
Temperature in ladle after skimming at furnace......................	1480° F. (805° C.)	1440° F. (784° C.)
Temperature in ladle after skimming at floor near mold.................	1440° F. (784° C.)	1430° F. (778° C.)

*Secretary, Brown Instrument Co.

A test, in a crucible furnace, of a special automobile aluminum gave the following temperatures:

Temperature of metal in furnace................................. 1305° F.
Temperature in ladle after skimming at furnace................... 1295° F.
Temperature in ladle after skimming at floor near mold........... 1280° F.

It will be noted that the temperature of the metal in the first two tests dropped 100° F. before the time of pouring. In the third test, the larger crucible and shorter carrying distance prevented as great a drop as in the other tests. Therefore, to insure consistent pouring temperatures, the temperature at the furnace and the distance to the molds must be regulated. The size of the crucible also must be considered, as a much slower rate of cooling prevails with the larger crucible of metal. It is possible to establish a melting-point temperature that will assure correct consistency of the metal at the mold.

Pyrometer Equipment for Electric Brass-melting Furnace

A permanently installed electric pyrometer may be placed on electric brass-melting furnaces and, if desired, an actual record secured of each heat. This shows the temperature at which each melt was made and also the time interval. The equipment developed for this purpose consists of a 60-day continuous recording pyrometer of the latest high-resistance type with chart range 0–2400° F. A record line is produced on this chart by means of a moving carbon ribbon so that a non-smearing permanent record is secured.

The thermocouple consists of extra heavy ¼-in. nickel-nickel chromium rods, similar to those used with portable pyrometers, but permanently installed. These heavy ¼-in. thermocouples are protected against the severe gases by heavy porcelain tubes, which in turn are inserted in firebrick tubes closed at one end.

Compensating leads carry the cold end of the thermocouple away from the effect of the radiated furnace heat. These compensating leads may be extended to a cold-end well several feet in the ground or direct to a recording pyrometer. While the temperature of the molten metal may reach 2450° to 2550°, the thermocouple, flush with the furnace lining, shows a relative temperature of the gases above the metal line.

Platinum-platinum rhodium thermocouples were tried but the tilting of the furnace when pouring caused the lighter gauge wire to break. Besides, the gases caused crystallization. The nickel-chromium as well as the platinum-rhodium thermocouple must be carefully protected to resist a zinc-oxide precipitation from the fumes of the brass at time of pouring.

SPECIAL EQUIPMENT FOR BRAZING BRASS

In the manufacture of bow sockets for automobile use, the difficulty of brazing the socket was overcome when the temperatures in the furnace and the effect of air drafts in recharging the furnaces were regulated. A specially designed furnace with two burners directly opposite the position of the sockets to be brazed was used, giving very close regulation. Ten to fifteen sockets at the ends of the hollow tubes were placed in this furnace at a time. To determine the relative flame temperature and brazing temperature, the following equipment was used:

A duplex recording pyrometer having a chart divided into two sections was installed near the furnaces. Two 18-in. platinum-platinum rhodium thermocouples in fused-silica quartz tubes were connected to the recorder, using compensating leads direct to the recording instrument. One of the thermocouples was inserted in the hollow tube close to the eyelet of the socket to be brazed, a mixture of borax and brass filings having been poured down the tube as a flux. It was found that the actual temperature of the furnace and of the couple were very close. The thermocouple was then removed from the tube and permanently located just to one side of the other burner, so that the hot end was in immediate juxtaposition to the sockets. It was found that a window immediately above the furnace, when open, permitted the air to be drawn through the opening of the furnace; an asbestos curtain was installed to prevent this.

The two galvanometer systems of several hundred ohms resistance, one for each section of chart, are mounted in a gray-iron casting, machined and enameled. As the thermocouples are connected by wire directly to each electrical system, there are no rotating switches to develop poor contact. A special 8-day clock drives the chart rolls as well as the few mechanical parts for depressing the pointer momentarily to the chart paper. A continuous roll of carbon ribbon produces a non-smearing dot every 30 sec., which permits the slightest change in temperature to be shown. Thermocouples of base metal or rare metal may be used.

The durability of the thermocouples depends greatly on the type of protecting tube selected. In galvanizing tanks, using a high percentage of zinc, considerable difficulty has been experienced in procuring a tube that will withstand the action of the zinc. Recently a test was made of nine types of tubes, closed at one end, installed in a galvanizing bath containing 99 per cent. zinc. After two months continuous immersion, all but two of the tubes were entirely eaten through by the action of the zinc. One of the two remaining tubes was of wrought iron with extra heavy wall; the other was a Resisteat tube composed of a high chromium-iron alloy. The wall of the wrought-iron tube was quite thin, but

the Resisteat tube showed very little effect and would likely hold up for several months longer.   Tubes of a nickel alloy rapidly gave out under the action of the spelter.

By a slight modification of the indicating pyrometer, a set of contacts may be inserted into the instrument so that by colored lights above each tank the foreman can tell at a glance just which tanks are correct and .which may require some adjustment.   By the substitution of a relay and valve mechanism, the temperature of the tank or kettle may be automatically controlled.

CORE-OVEN TEMPERATURE CONTROL

It was formerly common practice largely to disregard the preparation and systematic drying of the core, yet spongy castings due to blowholes or green cores were the bane of the foundry men.   Modern core-oven equipment and careful temperature regulation have contributed to the betterment of the casting shop.   Probably the most largely used temperature recorder on core ovens is the long-distance recording thermometer. One of these instruments should be used for each oven so that a continuous record of each may be secured.   Different-sized cores require a different drying temperature and a longer time interval.   With records available, it can soon be ascertained what time and temperature will be needed for the average sizes of cores the shop will be required to handle.

The equipment would consist of a recording instrument of the circular-chart type, clock-driven, actuated by a bulb containing a nitrogen gas.   Bronze or steel armored tubing is used to connect the bulb with the instrument, which if advisable may be located 150 ft. (45 m.) away. The temperature limits for core drying lie between 300° F. and 600° F. (150° and 315° C.) so that a chart range 50–600° F. or 50–800° F. will answer every requirement.   The bulbs are usually suspended through the tops of the ovens and kept in position by either a flange or metal strapping.   The armored flexible tubing should not be extended into the oven when the temperature exceeds 200 or 300° F.   Under conditions necessitating the installation of the flexible armored tubing part way through the oven, solid copper or steel tubing should be used.

Another efficient form of temperature-measuring equipment is a central-station indicating electric pyrometer.   A thermocouple of special sensitive construction is installed in each oven and connected by wiring to the central-station switch.   One or more recording pyrometers may also be connected through a plug-type switch, each plug representing an oven. In this way records may be made of any equal number of ovens.   Burned cores as well as green cores may be avoided by proper temperature regulation of the ovens.

Value of Annealing

In rolling brass, it is necessary to re-anneal several times before the product is ready for the warehouse. Electric pyrometers give indisputable evidence that the brass has been given the correct treatment. The furnaces, usually 18–20 ft. long, are loaded with the rolls of brass sheets and annealed to a predetermined temperature to produce soft or annealed brass as required. At least two thermocouples should be installed through the crown of each furnace near either end. The duplex recording instrument may be installed near each furnace or a central pyrometer house may be built, so that better protection will be given the recording instruments. Rapid strides have been made in the application of pyrometers for this work. The future will probably disclose many greater developments in simplifying the time and temperature control of annealing processes.

Automatic temperature control is being applied on nearly all heat-treating processes. An automatic control combined with an advancing time and temperature-schedule mechanism will place furnace operations in the future on a more nearly automatic basis.

Core-oven Tests

By F. L. Wolf* and A. A. Grubb,† Mansfield, Ohio

(Rochester Meeting, June, 1922)

THE tests here described were made to obtain information regarding costs, efficiency, etc. of baking cores in an oil-fired oven and two electric ovens, which were installed, early in 1920, in the core room of The Ohio Brass Co. and were used almost continuously during the rest of that year. Operating costs were rather high so a complete test was undertaken early in 1921.

The ovens will be referred to as Nos. 1, 2, and 3. Nos. 1 and 2 are electric ovens of different makes while No. 3 is an oil-fired oven made by the firm that made No. 2. The three are equipped with similar ventilating devices. They have full-height doors at front and rear. The cores are placed on iron racks, which are lifted by Cowan trucks and rolled into the ovens through the front doors. When baked, they are removed through the rear doors. Fig. 1 shows the general plan of the electric ovens while Fig. 2 shows the plan of the oil-fired oven.

Oven No. 1 is 91 in. (231 cm.) wide, 83 in. (210 cm.) high, and 79 in. (200 cm.) deep, inside measurements. The walls are constructed in panels about 18 in. (46 cm.) wide and $4\frac{3}{16}$ in. (10.6 cm.) thick, packed with blocks of insulating material and covered with sheet iron. The concrete floor on which the ovens set is 13 in. (33 cm.) thick. A recess about 5 in. (13 cm.) deep was cut in the floor to provide room for the heating units. Channels and plates supported on I beams serve as supports for the trucks and racks.

The ventilating system consists of a Sirroco No. 1½ fan, connected so as to draw air from the top of the oven or from the room, through damper C, and to deliver it either to the bottom of the oven or to the floor; the delivery is controlled by the vane A. The fan and pipes are covered with ¼-in. (6.3 mm.) asbestos-cement lagging.

The heating units have a capacity of 94.4 kw. and are located along

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the sides and in the floor. The temperature is controlled by a "Tycos" thermostat; there is also a glass thermometer on one of the doors.

The inside dimensions of oven No. 2 are each about 1 in. (2.5 cm.) shorter than those of No. 1. The walls are 2⅜₆ in. thick and built of 2 in. of asbestocel covered on each side with ⅛ in. of Firefelt and sheet iron. The heating units are of the same capacity and are located in the same manner as in oven No. 1. The ventilating system and temperature control are also the same.

Oven No. 3 is 102 in. (259 cm.) wide, 85 (216 cm.) high, and 78 in. (198 cm.) deep, inside measurements. The wall construction is the same as in oven No. 2; the floor is of ½-in. (12.7 mm.) steel plate laid on the concrete floor. There are two combustion chambers, one on each side of the doors; they are 21 in. (23 cm.) high, 21 in. wide, and extend the length of the oven. There are several openings along the side to permit hot gases to pass into the oven. An Anthony ¾-in. high-pressure burner is located at the front of each combustion chamber. Fuel oil is supplied at a pressure of about 70 lb. and air at about 60 lb. The ventilating system is similar to that of the other two ovens; the recirculated air, however, is delivered about 3 in. above the combustion chambers on each side and directed toward the middle of the oven. Sheet iron is supported over the chambers in such a manner as to direct the air toward the floor.

The oil-fired oven installation, however, was by no means ideal. The structure of the floor and supports under the ovens was such that the location of combustion chambers under the surface could be accomplished only with difficulty. For this reason we adopted the present system, which has not proved fully satisfactory and is to be reconstructed in the near future.

The cores baked in the tests were the regular run of cores used in our foundry. They consisted largely of valve bodies up to 3 in. in size, valve fittings, and overhead-trolley fittings. They varied in weight from a fraction of an ounce to a couple of pounds but averaged about 3½ oz. They were composed of lake and molding sands and sand-blast dust with linseed oil and a small quantity of glutrin as binder.

Each oven was carefully studied to determine the best method of manipulation, heat absorption, heat losses, etc. Test bakes were then made and analyzed as regards heat distribution and costs.

A Leeds & Northrup potentiometer and seven thermocouples located at various points in and about the oven were used to study temperature distribution. A pitot tube was placed in the flue to measure the volume of gases passing out. Power consumption in the electric ovens was registered by kilowatt-hour meters and an especially designed oil-measuring apparatus was used to measure the fuel-oil consumption in the oil-fired oven.

ELECTRIC OVENS

Radiation and Other Unavoidable Losses

Under this heading is included all unavoidable leakage of heat that occurs while the oven is closed as tight as is practicable. It was determined by saturating the oven with heat at the average baking tempera-
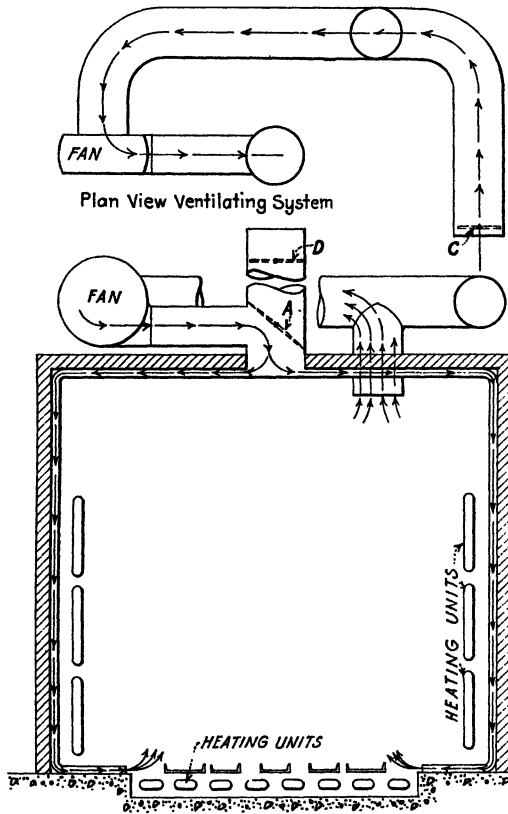
Fig. 1.—ELECTRIC CORE OVEN.

ture (400° F. by the door thermometer) and then measuring the power required to maintain that temperature with the oven completely closed. It was found that oven No. 1 consumed 19.7 kw. and No. 2 consumed 24.0 kw. The hottest points were along the sides of the ovens directly in front of the heating units, while the coolest points were near the floor. The thermostat was set to "kick in" and "kick out" within 10° F. but a variation in temperature of 40° was observed at points directly in front of the heating units, indicating that much heat is passed to the cores by direct radiation.

Heat Losses Under Operating Conditions

Inasmuch as operating conditions would not always permit the oven to be entirely closed, it was thought advisable to determine the heat losses due to ventilation. The measurements were made by setting the dampers, fan, etc. as desired and then observing the power consumption required to maintain the oven at baking temperature. The power consumption was in most cases greater than the figure observed when the oven was entirely closed. This was due to the loss of hot air through the flue and the admission of cool air either through damper C or about the doors. From this excess loss, the change in temperature of the air entering and leaving the oven, and the specific heat of air we were able to estimate the weight of air passing through the oven per minute. A fair check on several of these determinations was obtained by measuring the air flow in the flue with the pitot tube, thus practically confirming the method used for handling the heat losses.

Greater uniformity of temperature could be obtained in the electric ovens by operating the fans and recirculating the air, but a certain amount of hot air leaked past the dampers and up the flue, and more power was consumed. Under these conditions, about 1.9 lb. of air per minute passed through oven No. 1 and 22.5 kw. were consumed; for oven No. 2 the figures were 5.4 lb. and 32.2 kw., respectively. Evidently, considerable air was supplied to the cores even when all dampers were closed. Upon setting damper A partly to exhaust and partly to recirculate the air the temperatures were less uniform and much more power was consumed. When the damper was set to exhaust the air up the flue, the power consumption for each oven was about 40 kw., indicating that about 38 lb. of air were being drawn through the ovens each minute.

Heat Absorption

Tests were run to determine the quantity of heat absorbed by each oven in coming up to 400° F. from room temperature. The absorption of heat continued after the door thermometer registered 400°, so the power absorbed in holding this temperature was measured until the outer wall temperature reached an equilibrium, indicating that the walls were saturated. The tests were run with the ovens entirely closed. It was found that 48.0 kw.-hr. were required to heat oven No. 1 from room temperature to 400° F. in 28.5 min. An equilibrium was not reached, however, until 3 hr. later; during this period 8.4 kw.-hr. were absorbed in addition to the 19.7 kw.-hr. required each hour to compensate for radiation loss. Oven No. 2 consumed 46.0 kw.-hr. in coming from room temperature to 400° F. in 27 min. As in the case of No. 1, equilibrium was not reached for about 3 hr., during which time 10.0 kw.-hr. were absorbed

in addition to the 24.0 kw.-hr. required each hour to compensate for radiation loss. Oven No. 2 required slightly less power and rose in temperature slightly faster than No. 1, probably on account of its light construction. Its radiation losses, however, were greater.

Bakes

In baking cores, the electric ovens were handled as follows: They were heated to at least 450° F. and the loaded racks run in; this caused the temperature to drop 75° to 100°. The power was left on until the door thermometer indicated 425°, when it was cut off for the rest of the bake, which was completed in from 60 to 70 min. total time. Small bakes did not require the operation of the fan, but for the average bake, say 300 lb. and larger, the fan recirculated air while the temperature was rising above 400° F.

Three special bakes were made in each oven in an effort to determine just what became of the power consumed. Evidently part was used in heating the iron racks, plates, etc. from room temperature to the temperature at which they left the oven, part heated the air that entered the oven when the racks were put in and taken out, part heated the air drawn through the oven by the ventilating system, and part was radiated, while the rest was probably used in actually baking the cores. For each of the test bakes, the ovens were held at 400° before putting in the racks and were brought back to 400° immediately after the racks were removed. Time was measured from the entrance of a rack until the temperature was back to 400°, after its removal, and the power consumption, heat losses, etc. over the same period were considered. Following are the average figures per bake for each oven:

	Oven No. 1		Oven No. 2	
Weight of cores, net, pounds................	276.0		344.0	
Number of cores..........................	1515.0		1591.0	
Weight of rack, pounds....................	939.0		963.0	
Weight of plates, driers, core wires, etc., pounds...............................	1113.0		979.0	
Time in oven, minutes....................	71.0		73.7	
Power consumed, kilowatt-hours...........	59.0		67.1	
Power used in heating iron rack, plates, etc., kilowatt-hours.........................	22.9	(or 39%)	21.0	(or 31%)
Power used in heating air in oven, kilowatt-hours.................................	1.7	(or 3%)	2.5	(or 4%)
Power lost by radiation, etc., kilowatt-hours	24.7	(or 42%)	31.8	(or 47%)
Power used in baking cores, kilowatt-hours..	9.7	(or 16%)	11.8	(or 18%)
Power used per 100 lb. of cores, kilowatt-hours.................................	3.52		3.43]	
Total power per 100 lb. of cores, kilowatt-hours.................................	21.4		19.53	
Weight of cores, per kilowatt-hour, pounds..	4.57		5.13	

It should be noted that the total power consumption per hundred pounds of cores baked was lower in the case of oven No. 2 because the cores were larger and heavier loads were baked at a time. The heat actually used by the baking cores was practically the same in each case.

OIL-FIRED OVEN

This oven was more difficult to operate than the electric ovens, on account of the nature of the fuel and the attendant difficulties in ventilation and in heat measurement, so the results were not as uniform as
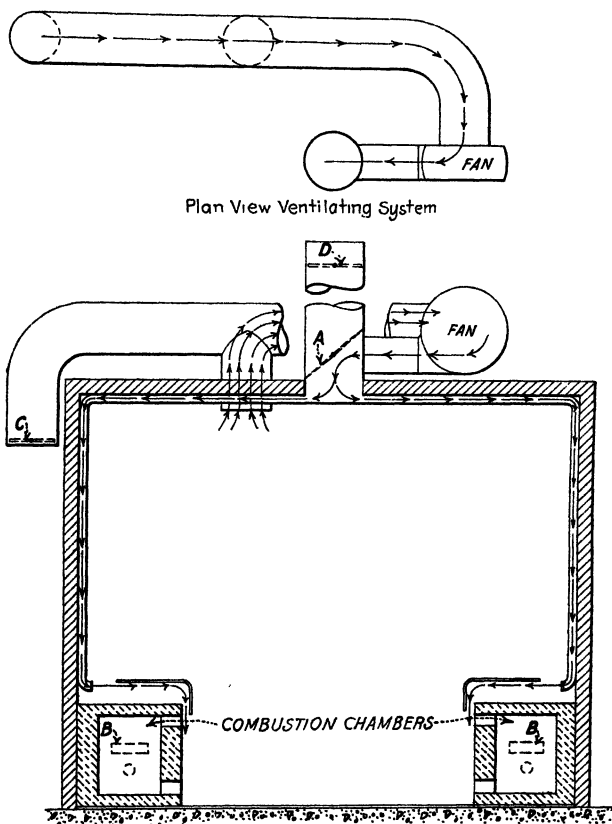
FIG. 2.—OIL-FIRED CORE OVEN.

desired. The fuel oil had a calorific value of about 19,560 B.t.u. per lb.; it was calculated that the available heat value of each gallon was about 138,800 B.t.u., which is equivalent to 40.6 kw.-hr. electric energy. The average baking temperature, as registered on the door thermometer, was 425° F.

Radiation and Other Unavoidable Losses

It was necessary to ventilate the oven and keep the fan going while the burners were operating, so the radiation loss could not be determined by the method used with the electric ovens. The method used consisted in heating the oven from 382° to 470° F., measuring the oil consumption, then shutting off the burners, closing up the oven, and measuring the time required for the temperature to return to the original reading. The dampers were open and the fan was exhausting air up the flue while the burners were operating, so the fuel consumption necessary to balance the losses and hold the temperature at 425° had to be determined; this was found to be 1.83 gal. per hr. To heat the oven from 382° to 470° in 32 min. 2.18 gal. of oil were consumed; the heat was lost by radiation, etc., with the oven closed, in 99 min. By calculation, it was found that radiation and other unavoidable losses, with the oven completely closed and at a temperature of about 425° F., were equivalent to about 0.736 gal. of fuel oil.

Heat Losses Under Operating Conditions

The burners operated best when the dampers were open and the fan was exhausting; then they consumed 1.83 gal. per hr. of oil, as mentioned. Attempts were made to obtain a more uniform temperature throughout the oven, by recirculating part of the gases and exhausting part, by closing the fresh-air inlet C and setting damper A in notch 3. This did not interfere seriously with the operation of the burners when the combustion chambers were hot and resulted in a lower heat loss; under these conditions 1.45 gal. of oil were consumed per hour. Nothing was gained, however, in uniformity of temperature; in fact, there was more variation. When the attempt to recirculate a larger proportion of the gases was made, the burners gave trouble. Evidently, considerable ventilation is necessary.

Heat Absorption

About 5 gal. of oil were required to bring the oven from room temperature to 450° F.; this was accomplished in about 50 minutes.

Bakes

Considerable trouble was experienced in obtaining a uniform bake, as the heat issuing from the vents in the combustion chamber struck the cores and burnt those near the edges of the racks while others farther in were green. Bricks were removed from the upper course, at intervals of about 1 ft., and iron sheets placed so that the air from the ventilating pipes was directed down over the edge of the chamber and up through the racks and cores. The fan was set to exhaust and all dampers were

open during the early part of the bake; but it was found advantageous to close damper C and partly recirculate during the latter part of the time when the burners were on. When the temperature, as shown by the door thermometer, reached 450° F., the burners were shut and the oven was completely closed.

Three bakes were made and analyzed as in the other tests; the averages of the results obtained were as follows:

Weight of cores, net, pounds....................	227.0
Number of cores...............................	691.0
Weight of rack, pounds........................	938.0
Weight of plates, driers, core wires, etc., pounds..	1240.0
Time in oven, minutes.........................	77.3
Fuel consumed, gallons........................	2.534
Fuel used in heating iron rack, plates, etc., gallons..	0.644 (or 25%)
Fuel used in heating air in oven, gallons..........	0.581 (or 23%)
Fuel lost by heat radiation, gallons..............	1.033 (or 41%)
Fuel used in baking cores, gallons................	0.276 (or 11%)
Fuel used per 100 lb. of cores, gallons............	0.122
Total fuel used per 100 lb. of cores, gallons.........	1.12

UNIFORMITY OF BAKES IN THE THREE OVENS

This test was run simultaneously in the three ovens, so as to make it strictly comparable. Thirty-nine plates of test cores, five cores on each plate, were made from the same batch of core sand. Thirteen plates were placed on each of three racks together with the regular loads of cores; one was placed on each corner of the upper, middle, and lower shelves and the other in the center of the middle shelf. The racks were then put into the ovens and baked under the best conditions we could obtain. The cores were then broken on a Wadsworth core-testing machine and the average strength of each plate was determined. Following are the results:

	Oven No. 1 Lb. per Sq. In.	Oven No. 2 Lb. per Sq. In.	Oven No. 3 Lb. per Sq. In.
Maximum strength...................	57.0	58.0	52.0
Minimum strength.................	42.0	38.0	28.0
Average strength...................	49.8	48.4	42.7
Maximum deviation from average, per cent........................	16.0	21.0	34.0

The cores on the lower shelf of oven No. 3 were overbaked by the hot gases issuing from the combustion chambers: this difficulty was experienced in nearly all the bakes in this oven. After sheets of metal were placed on the edge of the second shelf, so as to protect the lower shelf, none of the cores were burned but a few in the center came out a trifle green.

. The test was then repeated using only the upper and middle shelves of ovens Nos. 1 and 3; following are the results:

	Oven No. 1 Lb. per Sq. In.	Oven No. 3 Lb. per Sq. In.
Maximum strength.......	57.0	50.0
Minimum strength...............................	47.0	33.0
Average strength................................	50.3	44.8
Maximum deviation from average, per cent........	13.0	26.0

The tests indicate that greater uniformity can be obtained in the electric ovens and the cores are probably a little stronger than in the oil-fired oven.

COMPARISON OF OVENS

The analyses of bakes given are not directly comparable because racks of different weights and batches of cores of different size and weights were used. The figures below show an average bake and the heat consumption based on such a bake. They are, therefore, a fair comparison of the three ovens.

	Oven No. 1	Oven No. 2	Oven No. 3
Heat used in bringing oven from room temperature to baking condition, B.t.u..............	192,700	191,200	694,000
Heat radiated (or lost) while oven is closed as tightly as possible and held at baking temperature, B.t.u. per hour	67,200	81,900	102,000
Heat lost and radiated by oil-fired oven while maintaining baking temperature, burners on, fan recirculating, dampers open, B.t.u. per hour......................................			201,000
Average strength of test cores baked at various points in oven, pounds....................	49.8	48.4	42.7
Uniformity: maximum deviation from average strength, per cent.......................	16.0	21.0	34.0
Average bake, taken same for the three ovens:			
Weight of iron rack, pounds................	946		
Weight of plates, driers, etc., pounds........	1,111		
Weight of core, net, pounds...............	282		
Average weight of cores, gross, ounces.......	3.6		

	Oven No. 1	Oven No. 2	Oven No. 3
Heat consumption for average bake as above, B.t.u.			
For heating rack......................	36,300	36,300	36,300
For heating plates, driers, etc..........	42,600	42,600	42,600
For heating air in oven................	5,700	8,500	80,600
Radiation and other unavoidable losses..	84,500	108,200	143,200
For baking cores....................	33,900	33,000	47,700
Total............................	203,000	228,600	350,400
Cost per 1000 B.t.u., cents..............	0.8052	0.8052	0.0804
	(2.75c. per kw.-hr.)	(2.75c. per kw.-hr.)	(11.14c. per gal.)
Cost distribution for average bake			
For heating rack......................	$0.290	$0.290	$0.029
For heating plates, driers, etc..........	0.343	0.343	0.034
For heating air in oven................	0.046	0.068	0.065
Radiation and other unavoidable losses..	0.681	0.872	0.115
For baking cores....................	0.273	0.266	0.038
Total............................	$1.633	$1.839	$0.281

The factors other than cost of power or fuel that must be considered in comparing the ovens are uniformity and quality of bake, together with labor requirements for operation. The figures given show that more uniform bakes were obtained in the electric ovens. Work was rather slow in the core room when the tests were made, so cores varying greatly in size were baked together on the same rack. Fairly uniform bakes, however, were obtained in oven No. 1 but trouble was experienced in the oil-fired oven. Even when cores of uniform size were put on the rack, there was a tendency to burn those on the outer edges of the lower shelf while those in the middle were green. Sheets of metal hung so as to protect this shelf from the direct heat failed to remedy the condition entirely, so we quit using the lower shelf of the racks baked in this oven. Fairly uniform bakes were then obtained.

Core losses in ovens Nos. 1 and 3 were observed for comparison. Of 27,000 cores observed, 0.05 per cent. of those baked in the electric oven were burned and none came out green, while 1.95 per cent. of those baked in the oil-fired oven were burned and 2.14 per cent. came out green. This amounted to a loss of 30 or 40 cents on an average for each rack baked in the oil-fired oven against less than 1 cent for the electric oven.

The labor requirements for the oil-fired oven were much greater than for the electric ovens. The burners operated on high-pressure air and oil and required considerable attention to obtain the proper mixture for a clean hot flame; slight variations in pressure frequently extinguished the flames. On the other hand, the thermostat on the electric oven could be set to "kick out" at the desired temperature and little attention was required; in fact, racks of cores were baked with no attention being given the oven from the time the rack was run in and the power turned on until within a few minutes before it was removed. One man could handle seven or eight electric ovens while the same labor could handle not more than three oil-fired ovens of the type installed. At the wage paid to the oven tender, this made a difference in tending cost of between 15 and 20 cents per bake in favor of the electric oven.

The noise and smoke from the oil-fired oven are also worthy of consideration. With two burners going, it was almost impossible to converse in the core room. Although, when properly regulated, smokeless flames can be obtained from the oil burners, some smoke is almost certain to be made during the course of a bake. This leaves a dirty deposit on the cores, plates, rack, etc. which is a nuisance.

Conclusions

From the foregoing observations and data, we would conclude that power or fuel costs are decidedly in favor of the oil-fired oven while other considerations such as quality and uniformity of bake, core losses, con-

venience and cost of tending, cleanliness and noise are decidedly in favor of the electric ovens. Taking into consideration the higher losses in the oil-fired oven and the extra labor cost, the oil-fired oven cost about $0.80 per bake as compared with $1.63 for the electric oven No. 1. All of these considerations are based on the equipment and installation we now have and the present methods of manipulation, both of which we are trying to improve.

CRITICISM OF ELECTRIC OVENS

The test has shown that in the case of the better of the two electric ovens the heat and cost distribution on an average bake was as follows:

	B.T.U.	COST	PER CENT. OF TOTAL
For heating iron rack (946 lb.)...............	36,300	$0.290	17.9
For heating plates, driers, etc. (1111 lb.)......	42,600	0.343	21.0
For heating air (71.3 lb.) in oven.............	5,700	0.046	2.8
Radiation and other unavoidable losses........	84,500	0.681	41.6
For baking cores (282 lb.)..................	33,900	0.273	16.7
Total.................................	203,000	$1.633	100.0
Cost per 100 lb. of cores....................		0.580	

Only 16.7 per cent. of the heat is used for baking cores, the other 83.3 per cent. heats iron racks, plates, etc., or is lost into the room and up the flue. Some of this, however, is absolutely necessary. Plates and driers are indispensible, the ovens must be ventilated while hot in order to produce a good bake and some heat is certain to be lost through the walls and doors. But we believe that much of the heat consumption with its entailing cost is quite unnecessary, and can be saved by proper design. So long as oil or gas is used as the source of energy, the cost per B.t.u. is so low that these losses are hardly worthy of consideration, but with the high cost of electric energy, a great deal more care must be given to design and construction if economy is to be obtained.

Heating Rack

The racks furnished with the electric ovens averaged about 1000 lb. each. The weight was considerably reduced by removing all the tubular supports that could be spared, but even with this lighter weight the racks absorbed used 17.9 per cent. of the heat used for a bake. By using lighter angle iron, seamless tubes, etc., the weight could certainly be reduced 20 to 30 per cent. more without reducing their strength to the danger point; each 100 lb. of iron thus saved would mean a saving of 3 cents per bake in the electric oven.

Heating Plates, Driers, etc.

The same argument applies to plates and driers. The plates used in the test weigh from 20 to 30 lb. each. We are now trying a few plates made of No. 14 sheet iron stiffened with light angle iron; these plates weigh about 8 lb., thus saving of 60 to 70 per cent. in weight. Several driers, too, have been redesigned with a saving of 50 per cent. in weight. Such savings on the average rack of cores, plates and driers used in the test would mean a saving of about 20 cents per bake.

Heating Air in Ovens

Experiments conducted in our laboratory have shown that at least 4.5 lb. of air should be supplied during a bake for each pound of linseed oil in the cores; more air does no harm so far as the strength of the cores is concerned but requires a useless expenditure of heat energy. The average bake referred to above contains between 5 and 7 lb. of linseed oil and hence would actually require only 25 or 30 lb. of air. Evidently the air in the oven when the rack was placed there together with that which leaks in around doors and enters when doors are opened to examine the cores, is sufficient for a proper bake, so no additional ventilation need be supplied unless heavier bakes are made.

Radiation and Other Unavoidable Losses

While these heat losses were unavoidable in the present installations, they could be materially reduced in a properly designed oven. The loss of 84,500 B.t.u. per hr. from an electric oven costs about $0.68 each hour, in addition to the discomfort caused to those working in the room. The construction of the ovens tested may be fairly satisfactory so long as cheap fuel is burned, but it is far from economical when the more expensive electric energy is used. Greater care must then be given to door and damper construction, with view to cutting down heat loss due to air circulation, and effective wall insulation must be provided to cut down losses by conduction. The temperature observed at various points on the outer surface of oven No. 1 ranged from 100° to 180° F., depending on proximity to through iron. Taking the temperature of the entire surface at this lowest figure, it indicates that in a year of 275 working days at least 25,000,000 B.t.u. are lost, costing for electric energy about $200.

Baking Cores

The tests indicate that about 2000 B.t.u. are required to bake each 100 lb. of green core sand. It is highly probable that this figure cannot be

reduced as it depends on the binder used and the quantity of moisture in the green sand. By so arranging the oven and rack that larger loads may be baked at a time, it should be possible to increase the percentage of total heat used in actually baking cores and thus make a saving.

The figures and suggestions presented above indicate that there is a vast field for improvement in electric ovens—a field that is of growing importance because of the increasing use of electric energy for industrial heating.

DISCUSSION

W. M. CORSE, Washington, D. C.—How many dollars per day is the difference, when everything is considered?

A. A. GRUBB.—The fuel cost per 100 lb. of cores in electric oven No. 1 was 57.9 cents; in electric oven No. 2, 65.2 cents; and in the oil-fired oven No. 3, 9.9 cents. Other costs, labor and so forth, are almost double the power costs in the case of the electric ovens and several times the fuel cost in the case of the oil-fired oven.

W. M. CORSE.—What is the capacity per 100 lb. per oven, does it cost $10 or $100 a day more to operate that oven?

A. A. GRUBB.—That will depend on the number of bakes run; we bake about 275 to 300 lb. per bake and ten bakes per day per oven.

W. M. CORSE.—About $1\frac{1}{2}$ tons of cores?

A. A. GRUBB.—Yes.

Neumann Bands as Evidence of Action of Explosives upon Metal

By F. B. Foley,* Rolla, Mo., and S. P. Howell,† Pittsburgh, Pa.

Foreword

Not infrequently, in the case of the failure of a metal structure, such as a bridge, tank, airplane, gun carriage, etc., a doubt arises whether the failure was due, among other causes, to the effect of an overburden or of an explosive maliciously or accidentally exploded in contact with it.   As few means of discriminating between the effects of these causes have been known and these means were of a rough qualitative character, it was surprising to find in the record of an inquiry into the failure of a structure, the statement that, by observations on the Neumann bands in the structure, it was possible to determine the speed of breakage in the metal and to distinguish between a fracture caused by an explosion and one caused by an ordinary mechanical strain.

Such a criterion, if well proved, was of value to the explosives experts of the Bureau of Mines, the Ordnance Department of the United States Army and the Bureau of Ordnance of the United States Navy, but search showed no record of it in the literature nor did any of the metallurgists and metallographers consulted know of any data on which such a criterion could be based having been assembled and critically studied. However, it was universally held that there was promise in the idea and that there existed unique facilities at the Bureau of Mines Experiment Station at Pittsburgh for the preparation, under definitely known conditions, of material for study by metallographers, and that material of value might also be obtained from the proving grounds of the Army and Navy.   Therefore, with the approval of the Director of the Bureau of Mines and the Chiefs of Ordnance of the War and Navy Departments, the Chairman of the Division of Engineering of the National Research Council, on April 12, 1919, appointed a Committee on the Investigation of Neumann Bands, which has since been engaged in this study and which, after some slight change in its personnel, consists now of the following:

Dr. William Campbell, Advisory Metallurgist, Bureau of Ordnance, Navy Department.

F. B. Foley, Metallurgist, Bureau of Mines.

S. P. Howell, Explosives Engineer, Bureau of Mines.

Comdr. O. M. Hustvedt, U. S. N., Bureau of Ordnance, Navy Department.

Dr. Charles E. Munroe, Chief Explosives Chemist, Bureau of Mines.

* Metallurgist, U. S. Bureau of Mines.
† Explosives Engineer, U. S. Bureau of Mines Experiment Station.

Dr. Albert Sauveur, Professor of Metallurgy and Metallography, Harvard University.

Col. W. H. Tschappat, Technical Staff, Ordnance Department, U. S. A.

The work, the results of which are presented in this paper, was done by Messrs. Foley and Howell, but the plan of operation was considered and approved by the committee as a whole and the details have, during the progress of the investigation, been inquired into by the individual members of the committee, and by the committee at called meetings. The investigators also had the benefit of the supervision and advice of Dr. Henry M. Howe, a large part of the metallographic study having been carried out in his laboratory. The study was completed in the Metallographic Laboratory of the University of Minnesota.

The Committee on the Investigation of Neumann Bands approves the publication of this first report, which is offered for publication by authority of the Division of Engineering of the National Research Council, but the authors alone are responsible for the statements presented and the conclusions found.

<div align="right">CHARLES E. MUNROE,
Chairman, Committee on Investigation of Neumann Bands.</div>

HISTORY AND PRESENT STATUS OF THEORY

In 1848, Neumann, in studying the meteoric iron of Braunau, noted the existence of certain geometrically arranged markings in the kamacite to which the name "Neumann lines" was given. Kamacite is a solid solution of iron and nickel having a nickel content running as high as 6 or 7 per cent. Crystallographically it does not differ from terrestrial iron; its individual grains, however, are much larger than those usually found in ferrite. Prestel[1] is credited with having first noted the existence of similar lines of Neumann in terrestrial iron. Many writers refer to these markings as lines. Osmond and Cartaud[2] thus refer to them in their study of the meteoric irons of Timbuktu and Caille, but in their study of the occurrence of these markings in ferrite[3] they use the word "lamellae." Howe called them Neumann bands in order to convey the idea of two dimensions, as the markings possess appreciable width under moderate magnification. In this article we shall follow Howe[4] in the use of Neumann bands as being descriptive of the appearance of their markings on a plane section.

In meteoric iron there are three constituents, kamacite, tænite, and plessite, characteristically arranged, in what has been called Widmannstättian structure. This structure is not to be confused with that of Neumann bands in kamacite that has been exposed to shock. Widmannstättian structure has been found often in steel as the result of the crystallographic arrangement of the ferrite while lines of the same kind

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[1] Cohen: "Meteoritenkunde," 89. Stuttgard, 1894. Schweizerbart.

[2] Sur les fers météoriques. *Mem. Rev. de Met.* (1904) 69.

[3] Crystallography of Iron. *Jnl.* Iron and Steel Inst. (1906) 71, 444.

[4] H. M. Howe: "Metallography of Steel and Cast Iron." N. Y., 1916. McGraw-Hill Book Co.

as those found by Neumann in meteoric iron have been readily produced in ferrite by exposing it to shock. In 1874, Tschermak,[5] in studying the nature of Neumann bands, explained their existence as being due to congenital twinning, that is to say, twinning that had taken place during the process of crystallization from the liquid state; but subsequent investigators, particularly Osmond and Cartaud,[6] have concluded that they are mechanical twins produced in ferrite, so far as we know, by shock only and the more readily the lower the temperature of the metal. Neumann, followed later by Tschermak and by Linck, considered that it was a question of fluorspar twinning.
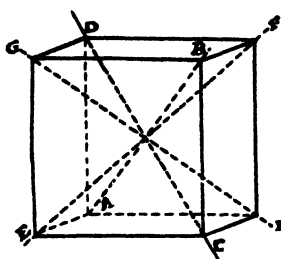
FIG. 1.—SHOWING TERNARY AXES.   FIG. 2.—OCTAHEDRAL PLANE OF CUBE; CD, TERNARY AXIS.   FIG. 3.—FLUORITE (FLUORSPAR) TWINNED.
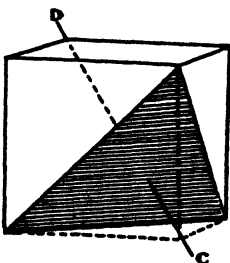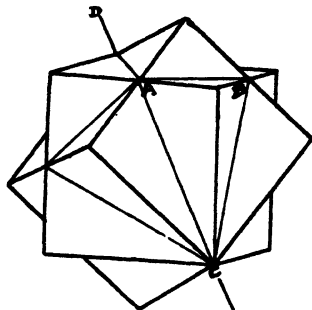
Fluorspar, or fluorite, the name by which this mineral is usually called in systematic mineralogy, belongs, crystallographically, to the isometric system, the system to which iron also belongs; like α iron, it is cubic. There are two forms of twinning, contact twins and penetration twins. In the former, a simple revolution of 180° about the twinning axis produces a simple crystal; while in the latter the twinned individuals interpenetrate one another. Penetration twins are common in fluorspar. The twinning plane is the octahedral (111) plane, and the axis of twinning is the ternary or trigonal axis. If a crystal in one complete revolution about an axis repeats itself three times in space, once at each 120°, that axis is called a ternary or trigonal axis. A cube has four such axes (Fig. 1). The octahedral, or twinning, plane of fluorspar is shown in Fig. 2 with the ternary, or twinning, axis CD passing through it. Imagine that two cubes, instead of one, occupy the space of the cube represented in Fig. 1, and that one of these cubes remains stationary while the other revolves about axis CD in the plane shown in Fig. 2; such motion would produce the twin shown in Fig. 3. This twinning produces contact or junction planes ABC, Fig. 3, along which the face of the one cube cuts

[5] *Sitzungsberichte* Akad. Wiss. Wien, Math. Nat. Classe (1874) **70**, 443.
[6] *Op. cit.*

through the other. Neumann lamellae or bands are found parallel to such planes of junction. Osmond and Cartaud, finding a difference of opinion among earlier investigators regarding these planes, polished, etched and photographed six faces of a rectangular parallelopiped of iron and so mounted these photographs on a wooden model that they were able to trace the Neumann bands, which were numerous in the specimen, on two or more faces of the model. From this they determined that the bands were always parallel to the planes of the trapezohedron (211), and confirmed the theory previously advanced by Linck.[7]

Fig. 4 represents a (211) trapezohedron. The notation (211) indicates, after Miller, that all the planes bounding the figure intercept the three crystallographic axes at a relative distance from the center of 1:2:2.
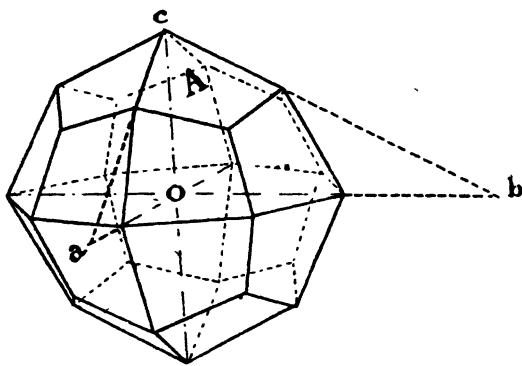
FIG. 4.—TRAPEZOHEDRON (211).

The axes in Fig. 4 are marked a, b, c. If plane A is extended so as to intercept axes a and b, and Oc is taken as unity, Oc will be related to Oa and Ob as 1:2:2. To identify this plane with Miller indexes, the reciprocals of the relation of its intercepts on the three axes, i.e., $\frac{1}{1}:\frac{1}{2}:\frac{1}{2}$ or 211, are used. Where it is desired to indicate that all the planes bounding the figure are 211 planes the notation appears in brackets, (211).

Fig. 5 represents a cube (the form in which α iron crystallizes) truncated by a 211 plane; a–b–c is a 211 plane. If a value of 2 is given to the sides of the cube then Oc will be related to Oa and Ob as 1:2:2, from which it follows that Oc is $\frac{1}{2}$ Od. Thus, we observe that a 211 plane intercepts the surface of a cube either parallel to a diagonal of the face, as ab, or parallel to a line running from a corner to the middle of the opposite edge, as ac and cb. As the Neumann bands are parallel to such 211 planes, they must likewise intercept the faces of the cube in directions parallel to the diagonal of the face or to a line running from a corner to the middle

[7] *Zeits. für Crystallographie* (1892) **20,** 209.

of the opposite edge of a face. This would result in the six directions shown in Fig. 6, which are the directions we would expect Neumann bands to assume on a plane parallel to the face of a cubic crystal of ferrite. In addition, Howe finds evidence that Neumann bands may also follow the cubic cleavage; however, in a random section taken through a specimen, the above conditions would rarely be realized.

In the case of "annealing twins," deformation is not followed immediately by twinning because while the deformation produces a translation there is not sufficient molecular mobility in the metal to permit the symmetrical orientation of the molecular polyhedra with respect to the original position; but when such deformed metal is annealed that necessary molecular mobility is attained which permits the consummation of the twinning. We find no record of annealing twins being produced in
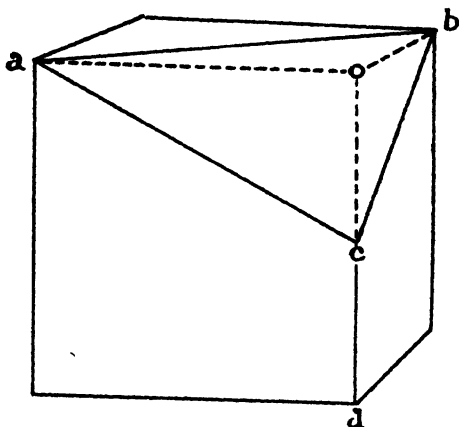
FIG. 5.—CUBE TRUNCATED BY 211 PLANE.   FIG. 6.—INTERSECTION OF CUBE FACE BY 211 (NEUMANN) PLANES.
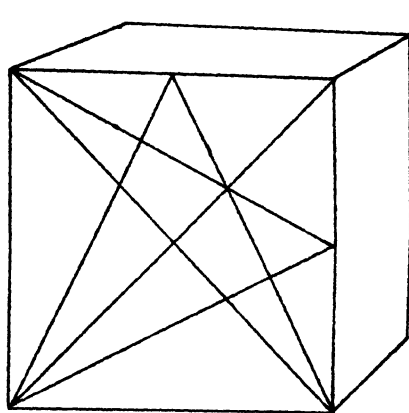
ferrite although they are common in some of the non-ferrous alloys, particularly in brasses.

If a polished face of a piece of metal be examined after deformation by a slowly applied stress, two systems of lines will be found to have been produced.[8] One system will be symmetrical with reference to the direction of the applied stress; its lines are called "Lüders lines." As they are symmetrically arranged with respect to the direction of the applied stress it is evident that they can bear no relation to the crystallinity of the metal, and this is true because the same kind of lines will be found regardless of the structure of the material, whether crystalline, cellular, or amorphous. Other lines will be found also to have been

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[8] Osmond, Frémont et Cartaud: "Les modes de déformation et de rupture des fers et des aciers doux." *Mem. Rev. de Met.* (1904) 11.

developed within the crystal units; these lines, called "slip bands," are usually curved but roughly parallel within a single grain, and they usually change their direction in passing from one grain to another. If the stress is applied suddenly, as by the blow of a sledge, a third set of markings will be observed by microscopic examination. These will differ from the other lines in that they will be much straighter. If the surface is repolished and etched, the Luders lines and the slip bands will no longer be visible, whereas the third set of lines will be brought out more prominently; the lines that persist after etching are called Neumann bands. If a specimen containing Neumann bands is heated to a temperature of 800° C. for a short time, the twinning will be effaced and etching after polishing will not reveal them. Osmond-Wuth, Heyn, Bauer, and Goerens have shown that the rapidity of solution of iron increases steadily with the degree of mechanical treatment, from which Tamman[9] concludes that the solubility of iron will therefore increase with the number of twin lamellae, or Neumann bands, present in it.

The susceptibility of ferrite to the formation in it of Neumann bands from sudden deformation appears to be governed by the condition of the ferrite, particularly by its brittleness (which is affected by phosphorus and other elements) and by its temperature. Osmond and Cartaud, however, found that Neumann bands were not produced in it at the temperature of blue temper, although it has been found by tests, conducted over a range of temperatures, that the ductility, as measured by reduction of area and elongation, decreases in this region. Hadfield[10] showed many such bands formed in Swedish iron near the point of fracture produced without elongation at liquid-air temperature. Robin,[11] working with a steel containing 0.06 per cent. carbon and 1 per cent. phosphorus, produced the bands at a number of temperatures between 20° and 600° C., but he failed to find any produced at a temperature of 700° C. His iron, of course, was embrittled by the phosphorus. Howe[12] produced them, by slowly applied pressure, in a 4 per cent. silicon steel, which he called quite brittle. By the application of a single heavy blow, and also after a number of moderately heavy blows, Stanton and Bairstow[13] produced them in Swedish iron, but there were none found after breaking with a great number of lighter blows. The blow of a sledge has produced them for Howe.

[9] Tamman: "Lehrbuch der Metallographie," 129. Leipzig, 1914. Voss.

[10] R. A. Hadfield: *Jnl.* Iron and Steel Inst. (1905) **67**, 248.

[11] F. Robin: Etude micrographique sur la fragilité des aciers à l'écrasement. *Mem. Rev. de Met.* (1911) **8**, 436.

[12] *Op. cit.*

[13] Stanton and Bairstow: *Engineering* (1908) **86**, 731; *Proc.* Inst. Mech. Engrs (1908) Pt. 3–4, 889.

NEUMANN BANDS IN OTHER MATERIALS THAN FERRITE

We have been discussing, so far, the occurrence of Neumann bands only in ferrite. It is not to be understood that such mechanical twins cannot be produced in other materials. They should be found in any crystalline pure metal or solid solution that could satisfy the conditions for their production. Carpenter and Edwards[14] concluded that tin, zinc and cadmium give mechanical twins immediately on straining. After slight shock and after local deformation of the metal by a needle, G. Timofeef[15] found twinning in zinc. They have been developed in bismuth by the spreading of a crack by a sawcut. Indeed, Benedick suggests that twin lamellae are formed at every surface of slip in strained metal.

Neither is ferrite the only constituent in steel subject to the formation of mechanical twins. Osmond and Cartaud[16] state that there are no known mechanical twins in β iron but that γ iron twins under severe deformation and they suggest that martensite is the result of the formation of an infinity of twins produced by the stresses incident to the transformation of γ to α iron at low temperature on sudden quenching. The α iron thus produced is pseudomorphic on the martensitic structure of γ iron. Such twins might probably be classified as congenital, since they are formed simultaneously with the α phase.

MATERIAL USED AND PREPARATION OF SPECIMENS

This investigation is restricted to a study of the production of Neumann bands in the ferrite of low-carbon steel by stresses of varying rates of propagation in the specimens. These specimens were of mild steel in order that they should contain a considerable proportion of ferrite; hence they were prepared from cold-rolled shafting having a carbon content of less than 0.15 per cent. The specimens, each 1½ in. in diameter and ¼ in. thick, were cut from the shafting. When they were partly cut off, they were numbered and photographed, as shown in Fig. 7. All specimens having identical designating letters (A, B, C, D, or E) were cut from the same piece of shafting, and the disks were numbered consecutively, from 1 to 10, beginning at that end of the piece that showed the shorter projection. The specimens were then cut into disks of the designated dimensions and were annealed by being placed on edge in an electric furnace and annealed at 871° C. (1600° F.), held at

[14] C. A. Edwards: Metallic Crystal Twinning by Direct Mechanical Strain. *Jnl.* Inst. of Metals (1915) **14**, 116.

Edwards and Carpenter: Hardening of Metals. *Jnl.* Iron and Steel Inst. (1914) **89**, 138.

[15] *Compte rendus* (1889) **109**, 680–692.

[16] *Op. cit.*

that temperature ½ hr., and allowed to cool down overnight with the furnace. This was done for the purpose of removing any stresses or twinning that might have been present in the material. The quality of each specimen was dependent on the nature of the final heat treatment, so no special consideration was given to any prior heat treatment of the cold-rolled steel shafting. Readings were taken with a Hoskins pyrometer. As this heat treatment produced a scale on the disks, it was necessary to grind all surfaces; the top and bottom surfaces in order to make them smooth, and the edges in order to bring them to a diameter of 1½ in. This removal of the scale reduced the thickness of the disks to 0.231 ± 0.004 in. The disks weighed 50 ± 1 gm. each.
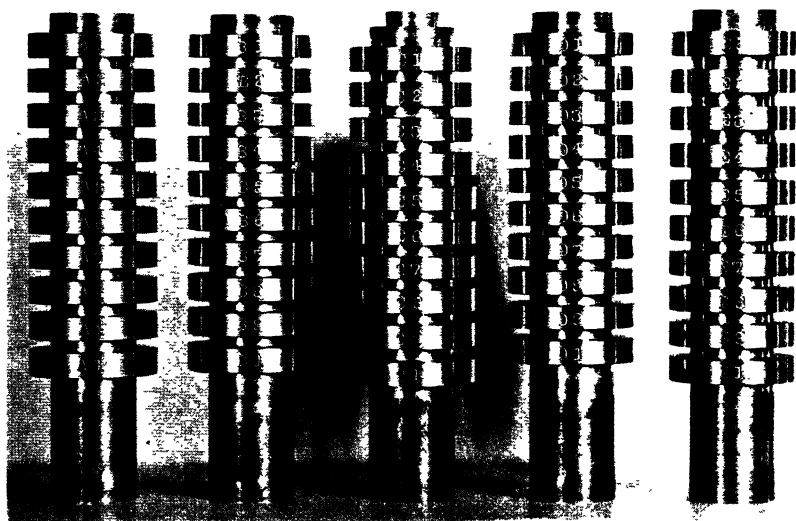
FIG. 7.—TEST SPECIMENS, SHOWING METHOD OF PREPARING THEM.

The composition of one, D-9, was as follows: Carbon 0.110 per cent., phosphorus 0.09 per cent., manganese 0.597 per cent., silica 0.105 per cent., sulfur 0.099 per cent.

METHODS OF PRODUCING DEFORMATION

As it was thought that the abundance of Neumann bands in ferrite would vary with the speed of deformation with or without fracture, it was at first decided to make tests with several explosives, each having a known but different rate of detonation, and also by the impact of a falling weight. Tests with explosives were made by using the Bureau of Mines standard test with small lead blocks.[17] In this test the

[17] S. P. Howell and J. E. Tiffany: Methods for Routine Work in Explosives Physical Laboratory of Bureau of Mines. Bureau of Mines *Tech. Paper* 186 (1918) 36–37

specimen, being a disk 1½ in. in diameter and approximately ¼ in. thick, is placed on a lead cylinder 1½ in. in diameter and 2½ in. in height, and on the top of the specimen is placed 50 gm. of the chosen explosive in the form of a cylinder and fired with a No. 7 electric detonator. In each test the specimen was placed with the letters and numbers upright. The direction of the impact of the explosive on the specimen was downward. The height of the 50-gm. charge of explosive varied with the explosive and is shown in Table 1. The arrangement of the explosive, specimen, and lead block is shown in Fig. 8. The whole arrangement was enclosed in a paper tube so as to retain each unit in proper position. This paper tube formed the container of the explosive and was prepared prior to putting the explosive in position. Before firing, the lead block was placed on a firm horizontal base.

For the impact the specimen was placed on a firm horizontal steel base and a falling weight of 448.75 lb. dropped onto it from heights of 24 and 12 ft. (7.3 and 3.6 m.), the corresponding velocities being 39.29 and 27.78 ft. (11.98 and 8.47 m.) per sec. The tup, which was secured to the bottom of the falling weight, was a hemisphere having a radius of 2½ in.; but as the tests progressed, this tup was deformed until the point which came in contact with the specimen had a radius of 8 inches.

The specimens tested on the small lead block were not, in some cases, especially with the explosives having the lower rates

FIG. 8.—ARRANGEMENT OF EXPLOSIVE, SPECIMEN, AND LEAD BLOCK USED IN NEUMANN BAND INVESTIGATION.

of detonation, deformed as much as seemed desirable, for the surface of the specimen in contact with the lead block was effectively supported by it, and for this reason additional trials were made with the explosive on specimens that were not supported. For this purpose the charges were made as described above, using 50 gm. of the explosive and firing it with a No. 7 electric detonator, and with the explosive, disk, and lead block surrounded by paper. The lead block was then slipped out of the paper tube and the tube slightly crushed under the steel specimen so as to hold it firmly in place against the charge of explosive. The paper tube served also to retain the explosive in the shape of a cylinder. The explosive and specimen were then suspended in the bomb-proof by a wire passing through the paper tube above the explosive. The specimen therefore

hung below and in contact with the explosive. Upon firing, the specimen was projected downward and was caught in a box of sawdust from which it was recovered intact or in fragments. Tables 1, 2, and 3 show the details of tests on 33 specimens grouped into series A, B, and C.

TABLE 1.—*Series A, Specimens Supported by Lead Blocks*
Tested Aug. 12, 1919

Specimen Number	Test Number	Explosive Designated	Compression of Lead Block, Millimeters
A 1	B-1440	D-2385	21.00
B 2	B-1441	D-2385	21.00
C 3	B-1442	D-2385	20.50
B 1	B-1443	D-2335-A	16.50
C 2	B-1444	D-2335-A	17.50
D 3	B-1445	D-2335-A	17.50
C 1	B-1446	D-2407-A	10.50
D 2	B-1447	D-2407-A	10.50
E 3	B-1448	D-2407-A	11.00
D 1	B-1449	D-2408	3.50
E 2	B 1450	D-2408	5.00
A 3	B-1451	D-2408	5.00
E 4	B-1480	D-2617-C	7.50
A 5	B-1481	D-2617-C	8.00
B 6	B-1482	D-2617-C	8.00

TABLE 2.—*Series B, Specimens Not Supported by Lead Blocks*
Tested Dec. 29, 1919

Specimen Number	Test Number	Explosive Designated
B 4	B-1483	D-2385
C 5	B-1484	D-2385
D 6	B-1485	D-2385
C 4	B-1486	D-2335-A
D 5	B-1487	D-2335-A
E 6	B-1488	D-2335-A
D 4	B-1489	D-2407-A
E 5	B-1490	D-2407-A
A 6	B-1491	D-2407-A
A 7	B-1492	D-2617-C
B 8	B-1493	D-2617-C
C 9	B-1494	D-2617-C

TABLE 3.—*Series C, Impact Tests with 448.75-lb. Falling Weight*

Specimen Number	Test Number	Height of Fall, Feet	Velocity, Meters per Second
E 1	II-465	24	11.98
A 2	II-465	24	11.98
B 3	II-465	24	11.98
A 4	II-465	12	8.47
B 5	II-465	12	8.47
C 6	II-465	12	8.47

Table 4 gives the salient characteristics of the explosives used in Series A and B.

TABLE 4.—*Certain Characteristics of the Five Explosives Used*

Grade and Class of Explosive	Explosive Designated	Apparent Specific Gravity	Height of 50-gm. Charge 1½ In. in Diameter, Centimeters	Rate of Detonation in 1¼-in Diameter Cartridges, Meters per Second
60-per-cent. straight nitroglycerin dynamite....................	D-2385	1.21	3.6	5716
40-per-cent. straight nitroglycerin dynamite....................	D-2335-A	1.15	3.8	4470
Nitroglycerin permissible........	D-2407-A	1.00	4.4	3190
Nitroglycerin permissible........	D-2617-C	1.01	4.3	2296
Nitroglycerin permissible........	D-2408	1.01	4.3	1523

PREPARATION AND MICROSCOPIC EXAMINATION OF SPECIMENS

After the specimens had been subjected to impact, they were halved by sawing in a plane through a diameter of each disk. A surface, thus exposed, was ground flat, polished, and etched with a solution of amyl alcohol and nitric acid in the proportion of 10:1. It is our experience that some care must be exercised to avoid the formation on the surface to be examined of a film of amorphous metal, which may result from the polishing operation and which may not be wholly removed in the etching process. Such a film may mask the Neumann bands that may be present. It was this that prevented the development of what appear to have been Neumann bands in some tin specimens, after repolishing and etching, in some experimental work conducted by Edwards.[18]

Rosenhain pointed out that since the lines did not return on repolishing and etching, they could not possibly have been Neumann bands, but later Edwards was able to reproduce them by etching off a layer, about $\frac{1}{16}$ in. deep, of metal, which had evidently been amorphatized in the rough polishing, following this by a careful polishing on chamois and another etching. We have observed in some instances that a repolishing on the finishing wheel and re-etching have given a much better result than that obtained on the first etching. Neumann bands, in their physical appearance, bear no resemblance to any etching figure in steel that we know of except scratches on the surface of the specimen; therefore it is necessary to have a surface free from scratches; but a scratch of any length can readily be told from a Neumann band by observing that the Neumann band will almost invariably stop or change direction at a grain boundary, while a scratch will maintain a straight course through many

[18] C. A. Edwards: Metallic Crystal Twinning by Direct Mechanical Strain, *Jnl.* Inst. of Metals (1915) **14**, 116.

Edwards and Carpenter: Hardening of Metals. *Jnl.* Iron and Steel Inst. (1914) **89**, 138.

grains.   For this reason, it is essential that the grain boundaries of the ferrite be developed, otherwise one cannot be certain that Neumann bands are not present.   The amyl-nitric reagent used appears to develop the Neumann bands and the grain boundaries of the ferrite with equal rapidity.   With picric acid, which gives good results also, the development of grain boundaries appears to be rather slow at times.

All the specimens, which, for each deformation, were in triplicate, were examined microscopically and a representative specimen from each set was photographed at 200 diameters at three points in the section, one at 1 mm. from the face subjected to impact, one at the middle of the specimen, and one 1 mm. from the back, all of them being taken along the axis of the specimen.

RESULTS OF MICROSCOPIC EXAMINATION

Those specimens that were subjected to the impact from explosion while supported by a lead block will be designated as series A, and those that were without substantial support will be designated as series B. Table 5 gives a summary of the effect of the impacts on the shape of the specimens.

TABLE 5.—*Effect of Impact on Shape of Specimen*

	Speci-men Number	Figure Number	Impact Velocity, Meters per Second	Reduction in Thickness, Inch	Reduction in Thickness, Per Cent.	Shape of Resultant Diametrical Section
Series A.........	E2	10–12	1523	0.0108	4.7	Flat
	B6	13–15	2296	0.0133	5.8	Flat
	D2	16–18	3190	0.0138	6.0	Slightly cupped
	D3	19–21	4470	0.0480	20.9	Slightly cupped
	C3	22–24	5716	0.0593	26.8	Slightly cupped
Series B........	A7	25–27	2296	0.003	1.3	Flat
	A6	28–30	3190	0.0128	5.6	Cupped
	C4	31–33	4470	0.0218	9.5	Cupped and cracked
	B4	34–36	5716	0.0271	11.8	Cupped and broke apart

The thickness of the disks of series A was reduced in a greater degree for the same explosive velocity than that of the disks of series B.   This reduction in thickness is important and must be taken into consideration in estimating microscopically the number and character of the Neumann bands produced.   The greatest reduction in thickness was that of specimen C3, series A, which was subjected to an explosive impact of 5716 m. per sec., and the least was that of specimen A7 of series B,

which was subjected to an impact of 2296 m. per sec. None of the specimens in series A, which were supported by a lead block, broke under the impact, whereas the higher velocities ruptured unsupported specimens in series B.

Photomicrographs taken 1 mm. from the front face, which was the face on which the explosive was detonated, are marked position 1; those taken at the center of the section are marked position 2; and those taken 1 mm. from the back face, which was the face that was down, are marked position 3. The elongated black patches are pearlite areas running perpendicular to the face of impact. The sharp, black, usually straight lines, terminating or changing direction at grain boundaries, are Neumann
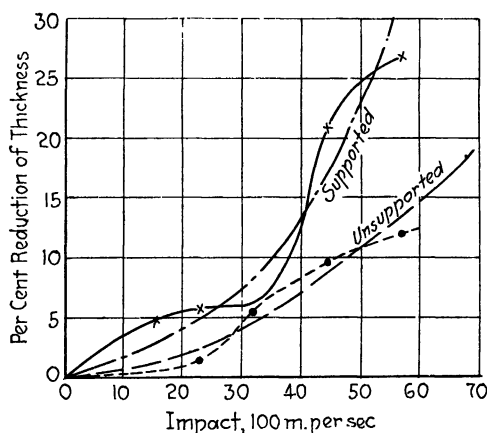
FIG. 9.

bands. In series A, photomicrographs of position 1 generally show a greater number of these bands than the photomicrographs of position 3, indicating that, as a rule, the number of twins decreases progressively from the neighborhood of the face of impact toward the back of the specimens which were supported. Specimen D3, Figs. 19–21 is an exception. The magnification in all of the Figs. 10 to 37 is 100 diameters and it is not extraordinary that, considering the actual size of the field under observation, there should be this one reversal in five instances. This does not seem to hold true in series B; of the four instances, two (specimens A7, Figs. 25–27 and A6, Figs. 28–30) show fewer bands in position 1 than in position 3. The phenomenon is the more marked in specimen A7 and a careful exploration of its mates, specimens B8 and C9, verified the fact that the number of bands near the surface of impact was less than at the back.

Counting the actual number of Neumann bands in a given field would be an arduous task and of doubtful value, for frequently the bands are so

close together that it would be difficult to decide whether to count one band or more; besides there is the question of discontinuity so that

FIG. 10.

FIG. 11.

FIG. 10.—SERIES A, SPECIMEN E2, POSITION 1. ×100. IMPACT VELOCITY
1523 M. PER SEC.

FIG. 11.—SERIES A, SPECIMEN E2, POSITION 2. ×100. IMPACT VELOCITY
1523 M. PER SEC.

FIG. 12.

FIG. 13.

FIG. 12.—SERIES A, SPECIMEN E2, POSITION 3. ×100. IMPACT VELOCITY
1523 M. PER SEC.

FIG. 13.—SERIES A, SPECIMEN B6, POSITION 1. ×100. IMPACT VELOCITY
2296 M. PER SEC.

what may appear as two or more separate twins may actually be but one. Therefore the authors have confined themselves to an ocular comparison

of the different fields. There is no questioning that 1523 m. per sec. (specimen E2, Figs. 10–12) produced the least number of twins. With a
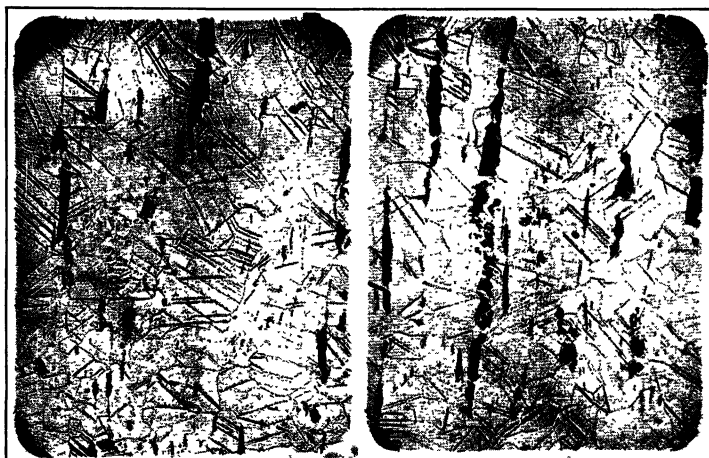
FIG. 14.   FIG. 15.

FIG. 14.—SERIES A, SPECIMEN B6, POSITION 2. ✕100. IMPACT VELOCITY
2296 M. PER SEC.

FIG. 15.—SERIES A, SPECIMEN B6, POSITION 3. ✕100. IMPACT VELOCITY
2296 M. PER SEC.

FIG. 16.   FIG. 17.

FIG. 16.—SERIES A, SPECIMEN D2, POSITION 1. ✕100. IMPACT VELOCITY
3190 M. PER SEC.

FIG. 17.—SERIES A, SPECIMEN D2, POSITION 2. ✕100. IMPACT VELOCITY
3190 M. PER SEC.

little less assurance one would be inclined to admit that 5716 m. per sec. velocity produced the greatest number (specimen C3, Figs. 22–24);

but, consider the effect the deformation of specimen C3 has on the number
of Neumann bands in a field.   All the specimens in series A were flat-

FIG. 18.   FIG. 19.
FIG. 18.—SERIES A, SPECIMEN D2, POSITION 3.   ×100.   IMPACT VELOCITY
3190 M. PER SEC.
FIG. 19.—SERIES A, SPECIMEN D3, POSITION 1.   ×100.   IMPACT VELOCITY
4470 M. PER SEC.

FIG. 20.   FIG. 21.
FIG. 20.—SERIES A, SPECIMEN D3, POSITION 2.   ×100.   IMPACT VELOCITY
4470 M. PER SEC.
FIG. 21.—SERIES A, SPECIMEN D3, POSITION 3.   ×100.   IMPACT VELOCITY
4470 M. PER SEC.

tened somewhat by the impact; specimen C3 was reduced in thickness
about 27 per cent. and from the shape of the diametrical section after

impact most of this reduction was localized near the face of impact, which means that there are more grains packed in unit area in specimen C3 than

FIG. 22.

FIG. 23.

FIG. 22.—SERIES A, SPECIMEN C3, POSITION 1. ×100. IMPACT VELOCITY 5716 M. PER SEC.

FIG. 23.—SERIES A, SPECIMEN C3, POSITION 2. ×100. IMPACT VELOCITY 5716 M. PER SEC.

FIG. 24.

FIG. 25.

FIG. 24.—SERIES A, SPECIMEN C3, POSITION 3. ×100. IMPACT VELOCITY 5716 M. PER SEC.
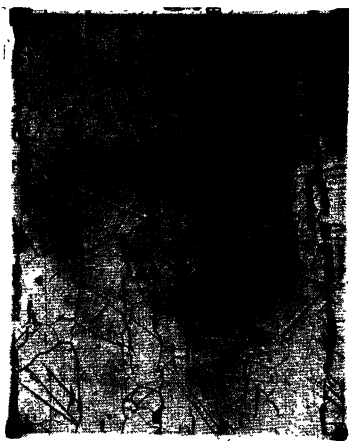
FIG. 25.—SERIES B, SPECIMEN A7, POSITION 1. ×100. IMPACT VELOCITY 2296 M. PER SEC.

in specimen D3, and more per unit area in specimen D3, which was reduced about 21 per cent., than in specimen D2 which was

reduced only 6 per cent.   The difference in the number of bands present in specimen C3, Figs. 22–24 and in the number present in specimen

FIG. 26.   FIG. 27.

FIG. 26.—SERIES B, SPECIMEN A7, POSITION 2.   ×100.   IMPACT VELOCITY
2296 M. PER SEC.
FIG. 27.—SERIES B, SPECIMEN A7, POSITION 3.   ×100.   IMPACT VELOCITY
2296 M. PER SEC.
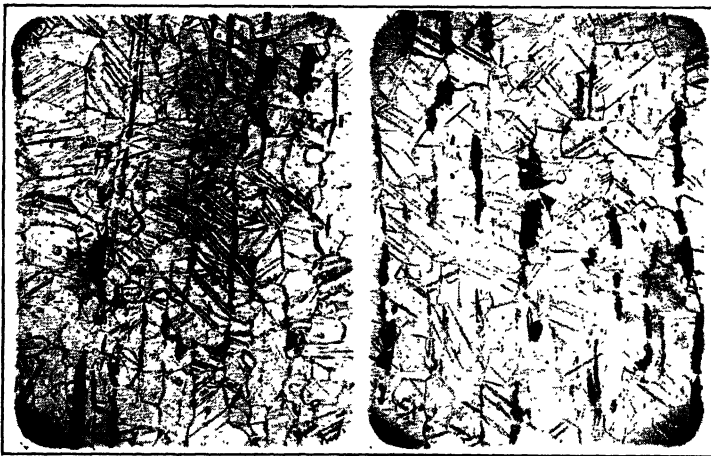
FIG. 28.   FIG. 29.

FIG. 28.—SERIES B, SPECIMEN A6, POSITION 1.   ×100.   IMPACT VELOCITY
3190 M. PER SEC.
FIG. 29.—SERIES B, SPECIMEN A6, POSITION 2.   ×100.   IMPACT VELOCITY
3190 M. PER SEC.

D2, Figs. 16–18, is not great and might be accounted for by the relative reductions of the specimens.   As between specimens B6 and E2, Figs.

13 and 10, the much greater difference in the number of bands can be accounted for only by the difference in rate of impact.

FIG. 30.

FIG. 31.

FIG. 30.—SERIES B, SPECIMEN A6, POSITION 3. ×100. IMPACT VELOCITY
3190 M. PER SEC.

FIG. 31.—SERIES B, SPECIMEN C4, POSITION 1. ×100. IMPACT VELOCITY
4470 M. PER SEC.

FIG. 32.

FIG. 33.

FIG. 32.—SERIES B, SPECIMEN C4, POSITION 2. ×100. IMPACT VELOCITY
4470 M. PER SEC.

FIG. 33.—SERIES B, SPECIMEN C4, POSITION 3. ×100. IMPACT VELOCITY
4470 M. PER SEC.

The reduction in thickness of specimens in series B was less than that in series A. The number of Neumann bands in series B does not appear

to be greater in one specimen than in another, unless it be that specimen C4, Figs. 31–33, has more than the others.   Neither does the number

FIG. 34.   FIG. 35.

FIG. 34.—SERIES B, SPECIMEN B4, POSITION 1.   ×100.   IMPACT VELOCITY 5716 M. PER SEC.

FIG. 35.—SERIES B, SPECIMEN B4, POSITION 2.   ×100.   IMPACT VELOCITY 5716 M. PER SEC.

FIG. 36.   FIG. 37.

FIG. 36.—SERIES B, SPECIMEN B4, POSITION 3.   × 100.   IMPACT VELOCITY 5716 M. PER SEC.

FIG. 37.—SPECIMEN B7, SHOWING ABSENCE OF NEUMANN BANDS IN MATERIAL PRIOR TO EXPOSURE TO IMPACT.   ×100.

of bands in series B appear to differ to any noticeable extent from the corresponding specimens in series A.

Considering the entire set of specimens without regard to whether they were of series A or series B, the number of Neumann bands does not appear to be commensurate with the degree of plastic deformation. Otherwise series A would have far more bands present than series B. The number of bands present does not seem to bear any relation to the degree of deformation.

No information relating to Neumann bands was obtained from the set of specimens which was subjected to deformation from a falling tup, weighing 448.75 lb. striking with a velocity of either 8.47 or 11.98 m. per

FIG. 38.—SHOWING THREE DIRECTIONS IN ONE GRAIN. ×930.

sec. These specimens were reduced to the thickness of a knife edge and no Neumann bands were recognized in them.

Fig. 38, taken at a magnification of 930 diameters, was taken to show three directions in a single grain. At first it was thought that there were more than this number of directions present in this particular grain, but it was noted on closer examination that what at first glance appears to be but one grain is in reality two and the very thin boundary line of another grain (ABC) becomes apparent on close examination. The two grains are oriented so similarly that the twins in one continue into the other with only a slight change of direction in crossing the very thin

grain boundary.   These two grains are probably on the verge of coalescence.   Although the instances of three directions in a given grain were rare, the phenomenon is present in at least two instances in Fig. 31. Most frequently we find but two directions for Neumann bands in a single grain.   Four directions were exceedingly rare, and persistent exploration failed to find any instance of more than four directions.

Neumann bands may be either ridges or furrows on an etched surface; this is well illustrated by Fig. 39.   Bands *AB*, *CD*, and *EF* are ridges with the light shining on the lower side casting a deep shadow.   These

FIG. 39.—BANDS *B*, *CD*, *EF* SHOWN AS RIDGES, BANDS *GH* AND *IJ* AS FURROWS; NOTE CHANGE IN RELATIVE HIGH LIGHTS AND SHADOWS.   × 930.

three twins continue in almost a straight line into the adjoining grain but there the relation of high light to shadow is reversed.   This is exactly what one would expect to find in a case of twinning.   Grains are said to etch more rapidly in directions normal to certain crystallographic faces than to others.   Sometimes, the dominant crystal may present a readily attacked face to the etching solution and the twinned elements present other less readily attacked faces, a condition which results in Neumann bands appearing as ridges.   The evidence here does not confirm the belief of Tamman[19] that the appearance of Neumann bands is due to the fact that the mechanically overstrained metal that causes the twin dis-

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[19] Tamman: "Lehrbuch der Metallographie," 129.   Leipzig, 1914.   Voss.

solves more quickly than the surrounding mass. According to Stead,[20] the cubic planes are most resistant to etching, the octahedral plane the least resistant, and the other random planes intermediate. Accordingly, one would expect that where a given grain presented an octahedral face to the acid the face would be most readily attacked and the Neumann bands would appear as ridges and that when a cubic face of the grain was presented to the acid the Neumann bands would be attacked the more rapidly and would therefore appear as furrows.

CONCLUSIONS

This investigation furnishes evidence in support of the following conclusions:

1. The abundance of Neumann bands increases with the velocity of impact till this passes 2296 m. per sec. With increase in velocity of impact beyond this point, there is no appreciable increase in abundance (compare Figs. 13 to 15 and 25 to 27 with Figs. 34 to 36).

2. There appears to be a decrease in the number of Neumann bands with the distance from the face of impact to the back of those specimens that were supported by a lead block.

3. Impact velocities up to 5716 m. per sec. do not produce Neumann bands having a number of directions in a single grain greater than four.

4. Neumann bands are found to be both ridges and furrows and their appearance on an etched surface does not seem to result from an etching out of mechanically overstrained metal, which would result in furrows only.

5. In comparing series A with series B the number of Neumann bands does not bear any apparent relationship to the amount of plastic deformation caused by the impact.

SUGGESTIONS FOR EXTENSION OF RESEARCH

One of the difficulties connected with an investigation in which the object to be attained is the quantitative relationship of the number of Neumann bands to the velocity of impact that produced them is to find a means of determining the amount of twinning. The twins cannot be counted. For example, in Fig. 39, between Neumann bands CD and EF two or three short twins protrude from the grain at a very slight angle with its surface. Obviously the magnification used in this case, 930 diameters, could not be used if counting were to be resorted to, yet a magnification of 200 diameters would not make such small bands sufficiently prominent for them to be noticed. A suggestion is offered through the findings of Osmond-Wuth, Heyn and Bauer, and Goerens that the rapidity of solution of iron increases steadily with the degree of mechanical

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[20] See H. M. Howe: "Metallography of Steel and Cast Iron," 269.

treatment.   One might therefore determine by measurements of the rate of solution the degree of mechanical twinning.

A photomicrograph of Swedish iron showing Neumann bands (Fig. 40) was furnished by Prof. C. Y. Clayton, who states that the specimen was heated to 1000° C., cooled slowly in the furnace, and sectioned with a saw; it was not subjected to any other mechanical deformation yet, after etching the sawed surface, the Neumann bands were found.   Gaichi Yamada, assistant professor of metallurgy, Kyoto Imperial University,

FIG. 40.—NEUMANN BANDS IN SWEDISH IRON.   ×200.

says that a specimen of iron heated to 1000° C. and held at that tempcra- ture for 30 days showed Neumann bands, though it had not been subjected to mechanical deformation other than that incident to sectioning.   We have found no Neumann bands as a result of sectioning with a saw any speci- mens of the steel used in this investigation; Fig. 37 shows the structure of the metal before subjection to impact.   Perhaps the presence of some embrittling influence has been the cause of the production of twins in the instances just cited.   These findings, and there have been others of this kind, suggest the advisability of determining in some manner the effect of inherent brittleness of ferrite on the formation of Neumann bands.

We realize the difficulty of getting a close estimation of relative brittleness but suggest that the impact test might serve as a first approximation.

It would also seem advisable to determine what effect the presence of mechanical twins, in small or large numbers, has on the physical properties of steels having an excess ferrite constituent. Do they cause a loss of ductility, a reduction or increase in tensile strength or elastic limit, or do they render the material less resistant to shock?

Further experiments should be conducted with the material used in this investigation using velocities below 2296 m. per sec. in order to determine the point at which the velocity does not produce the suddenness necessary to cause Neumann bands to appear in this material, and to determine also whether there is a critical velocity above which no further increase in the number of bands occurs.

DISCUSSION

ALBERT SAUVEUR, Cambridge, Mass. (written discussion).—Neumann bands in ferrite are the result of sudden deformation produced by impact or explosion, which may or may not cause rupture; in the absence of deformation Neumann bands are not formed. An explosion of such violence, therefore, as to cause fracture without any deformation whatever will not be accompanied by the occurrence of Neumann bands. As the force of impact increases, if rupture results, the deformation becomes more localized in the immediate vicinity of the fracture and so do the Neumann bands. They are not found a slight distance from the fracture. The closeness of the Neumann bands to the fractured surfaces may be taken as an indication of the violence of the explosion or shock. Other things being equal, the closer the bands to the fracture or, to put it differently, the smaller the area away from the fracture in which Neumann bands are found, the more violent is the impact that produced them.

Conditions of Stable Equilibrium in Iron-carbon Alloys

BY H. A. SCHWARTZ, H. R. PAYNE, A. F. GORTON, M. M. AUSTIN, CLEVELAND, OHIO

(San Francisco Meeting, September, 1922)

FROM time to time, one of the authors has had occasion to investigate the graphitizing reaction and has published the results mainly as discussion[1] of the work of other investigators. In view, therefore, of the diversity of both observations and interpretation, it seemed well to attempt a revision of the entire subject in the Research Laboratory of the National Malleable Castings Co. The work consisted of a careful study of a single, impure, iron-carbon alloy. The material, which may be considered typical low-carbon white cast iron of the malleable industry, was an air-furnace iron of the following composition: carbon 2.30 per cent., silicon 1.20 per cent., manganese 0.29 per cent., phosphorus 0.156 per cent., sulfur 0.048 per cent. The freezing range extended from 1213° to 1132° C., the latter figure agreeing with Gontermann's data.[2]

We have attempted to define the equilibrium diagram of this alloy, in the stable condition as related to the metastable diagram, from a temperature somewhat below the eutectic freezing point to a temperature below Ar_1 and for a graphitic-carbon content from practically nothing to the original total carbon of the specimen (see Fig. 6). More especially we desired to locate the line in the stable diagram corresponding to the Ac_m line of the metastable system, which is designated by analogy, the At_m line (t_m from temper carbon); also to determine the location of At_m with respect to Ac_m toward their lower extremities at Ar_1 in order to determine whether or not graphitization is completed by the separation of an iron-carbon eutectoid analogous to pearlite in the iron-cementite system. This would involve further a comparison of the locations of the stable and metastable A_1 points.

As the carbon concentrations of the solid solutions of the two systems are probably not identical, an explanation of their differences and of the

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[1] Merica and Gurevich: Graphitization of White Cast Iron upon Annealing. *Trans.* (1920) **62**, 509. R. S. Archer: Graphitization of White Cast Iron. *Trans.* (1922) **67**, 445. White and Archer: Annealing of Malleable Castings. Am. Found. Assn. *Trans.* (1919) **27**, 351.

[2] *Zeit. Anorg. Chemie*, **59**, 4.

mechanism of their conversion into one another was to be sought. The experimental methods of attack included the following:

1. Determination of thermal critical points of stable and metastable alloys.

2. Microscopic study of solid solutions of the two systems.

3. Chemical investigation of solubility of carbon in stable system, as related to temperature.

4. Study of changes in electrical resistance accompanying graphitization.

5. A few scattering observations incidental to other work that have a bearing on this problem.

6. Preparation and examination of a series of alloys varying progressively in carbon and having otherwise the composition of the hard iron to serve as a means of defining the location of Ac_m, or at least its terminus at A_1.

The thermal critical points were investigated by Gorton, using the Roberts-Austen differential method and Leeds & Northrup apparatus. The platinum-platinum-rhodium thermocouple measuring the temperature was checked at intervals against one calibrated to $\pm 3°$ C. by the Bureau of Standards by means of precision potentiometer measurements. The two couples were identical at all times and at all temperatures, within the precision of the standard couple.

The microscopic investigations were conducted by Payne, using the usual methods. The micrographic equipment was of the Bausch & Lomb inverted type, with optical equipment permitting of photography up to 4000 diameters. All photography was by monochromatic light, usually red or yellow, on panchromatic plates. Etching media are indicated in connection with the micrographs.

The solubility of carbon was investigated by Austin using essentially the same methods employed by Archer and by Schwartz in previous investigations.

The following precautions, however, were observed. The heat treatments were conducted in oxygen-free nitrogen in glazed quartz tubes heated in a Hoskins tube furnace. The temperatures were autographically recorded and automatically controlled to $\pm 5°$ C. For the operation of the tube furnace, an iron-constantan couple was required, which, however, was calibrated by comparison with a platinum-platinum-rhodium standard.

Graphitization, in general, was initiated at temperatures of 900° to 1000° C. and completed by a hold of 100 to 300 hr. at the desired temperature.

As a criterion of the attainment of equilibrium, a second specimen was held at the final temperature about 100 hr. longer than the first to test whether the reaction was still going on measurably. Another guide

is that, obviously, the reaction at a higher temperature will be complete in a time permitting the attainment of equilibrium at a lower temperature.

An attempt has also been made to check the conclusions by quenching the completely graphitized metal from a few typical temperatures. The data should differ from those obtained on cooling only by the lag between Ac and Ar. It was observed that in nearly perfectly graphitized material the agraphitic carbon[3] decreases at 750° and increases at 780°.

The application of electrical-resistance measurements to the study of the changes during graphitization and to the identification of the solid solutions is, we believe, new. The work was carried on by Gorton, who measured with a Kelvin bridge the resistance of a specimen enclosed in a glazed quartz tube in an atmosphere of nitrogen.

Temperature measurements were made with the thermocouple used for determining the critical points. The current was reversed during each resistance measurement to exclude the effect of stray thermal electromotive forces. Characteristic changes, both as to specific resistance and thermal coefficient of resistance, were found on the conversion of the metastable to the stable system and accompanying the solution of carbon (?) in ferrite. We believe that these electrical measurements proved a most fruitful field of investigation by throwing light on the interpretation of the results of the better known methods of investigation.

The plan of attack was supervised by the senior author, who is responsible also for the form of publication and the interpretation of results. In the latter task, however, he has been assisted by his co-workers.

FIG. 1.—THERMAL CRITICAL POINTS OF MALLEABLE IRON.

CRITICAL-POINT CURVES OF ORIGINAL WHITE CAST IRON

The critical-point curves of the original white cast iron and of the resulting completely graphitized product on the first reheating and on subsequent heatings are shown in Fig. 1. In the metastable system, Ac_1 is definitely at 776° C. (1430° F.) and Ar_1 at 749° C. (1380° F.), the mean value being 762°. In the stable alloy, on heatings subsequent to the first, the sharply marked points are Ac_1 768° C. (1415° F.) and Ar_1 721° C.

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[3] Agraphitic after H. M. Howe, to indicate non-graphitic carbon irrespective of any conception as to whether it is combined with or in solution in iron.

(1330° F.), the mean value being 745° C. We believe these points to be the critical points of the metastable system, for on cooling from, say, 1650° F. at the rate used in this investigation the metallic matrix is largely pearlite, caused by the reestablishment of the metastable system during the heat treatment. On the first heating of the stable system, Ac_1 is
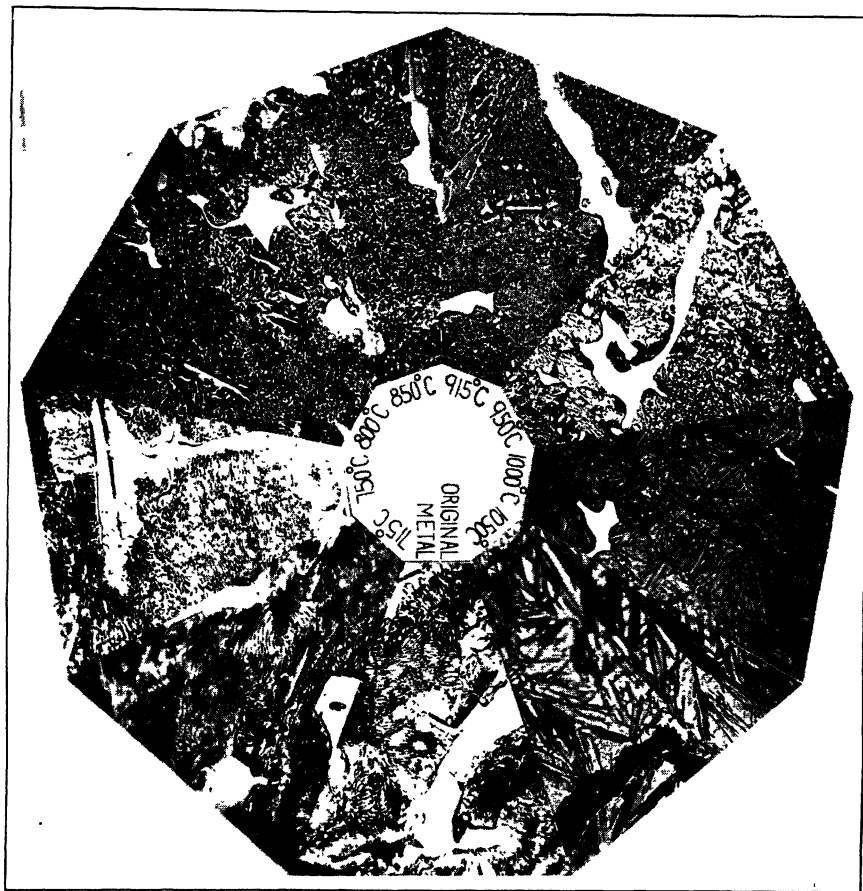
FIG. 2.—STRUCTURE OF WHITE CAST IRON QUENCHED AFTER REHEATING TO VARIOUS TEMPERATURES. ETCHED WITH PICRIC ACID. × 1000, REDUCED ONE-HALF IN REPRODUCTION.

found at 782° C. (1440° F.), the peak being but faintly marked, because of the small amount of dissolved carbon participating in the reaction. This is probably the true Ac_1 stable thermal critical point and is above Ac_1 metastable, which was not expected.

There appears in the graphitized specimen a faintly marked inflection in the cooling curve, beginning at about 749° C. (1380° F.), which may perhaps be a vestige of Ar_1 stable. The interpretation of the small peak

below Ac_1 at about 755° C. is not known; it is due to the metal but apparently not connected with the change of system. By careful heat treatments, it was possible to show Ac_1 stable and metastable in the same reheated specimen and demonstrate the gradual decrease in the former and increase in the latter as the intensity of reheating was increased. We are led to assign, tentatively, a mean value of 765° C. to A_1 for the stable system, a value 3° C. higher than A_1 metastable.
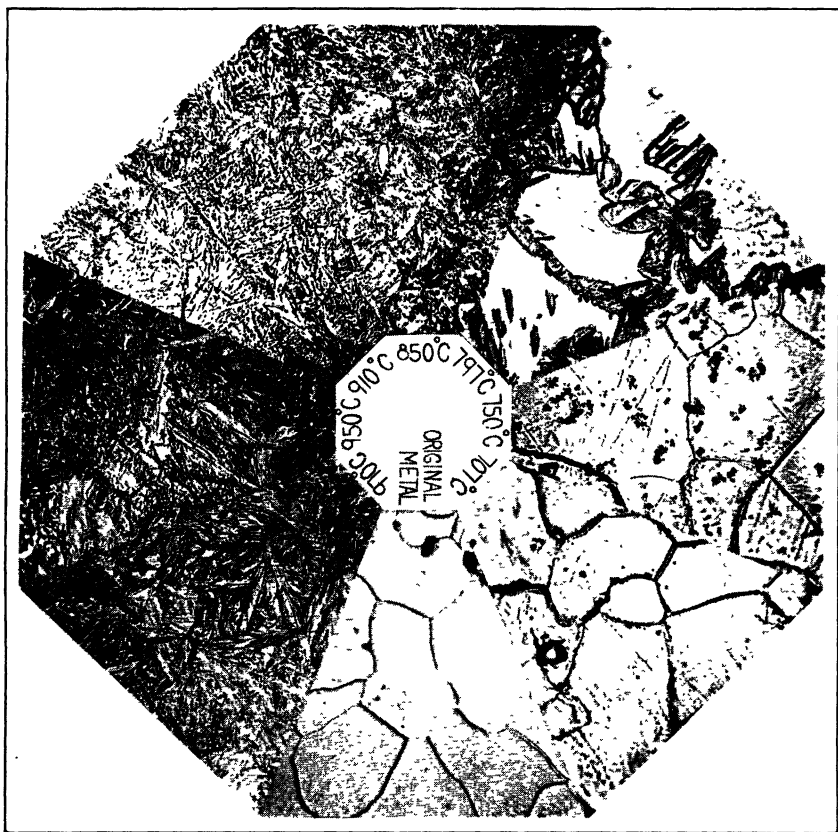
FIG. 3.—STRUCTURE OF MALLEABLE CAST IRON QUENCHED FROM VARIOUS TEMPERATURES. ETCHED WITH ALCOHOLIC HNO₃. × 1000, REDUCED ONE-HALF IN REPRODUCTION.

STRUCTURE OF REHEATED MALLEABLE IRON

A number of observers have photographed the structure of reheated malleable and demonstrated the penetration of so-called combined carbon from the grain boundary into the ferrite grain. We have not seen any publication pointing out the tremendous differences between the structures so obtained and the pearlite, sorbite, troostite, martensite series

characteristic of the metastable system. Fig. 2 shows the structure of the original white cast iron and of this metal briefly reheated to a series of designated high temperatures and quenched. The photomicrographs are at 1000 diameters, reduced in reproduction to 500, and the specimens were etched with picric acid. Fig. 3 shows a similar series starting with the completely graphitized metal; the etching medium is alcoholic nitric acid.

The differences in the appearance of the stable and metastable solutions do not arise from the etching media, as picric acid produces on the stable solutions a muddy stain that cannot be properly photographed and is in no sense similar to the metastable structures. Long-continued heat treatment at 900° or 950° C. will convert the structure from that characteristic of the temperature in Fig. 2 to that of Fig. 3. The structure of malleable reheated to 797° C. would, with a longer heating, become uniform throughout by diffusion and would then be identical with that at the next higher temperature.

We conclude that the stable and metastable solid solutions are metallographically distinct from one another. The photomicrographs indicate that Ac_1 metastable is below and Ac_1 stable above 750° C., i.e that the stable critical point is above, not below, the metastable.

TABLE 1.—*Solubility of Carbon in the Stable System*

Temperature, Degrees C.	Per Cent. Combined Carbon	Decrease in Combined Carbon	
		Per Cent.	During Last Hours
	Beginning with Hard Iron		
740	0.08	0.00	144
750	0.12	0.00	72
775	0.57	0.09	96
790	0.78	0.04	96
790	0.66	0.05	96
825	0.68	—0.02	96
840	0.81	—0 02	96
850	0.73	0.01	72
950	1.10 Same time as 850° equilibrium		
	Graphite burned down to 0.12 per cent.		
	Beginning with Annealed Iron		
		Increase in Combined Carbon	
740	0.05	−0.05	4
750	0.03	−0.04	4
780	0.56	0.27	5
1100	1.75	0.06	½
	Graphite exhausted to 0.01 per cent.		

The thermal data seem to have lagged about 30° C. behind the microscopic data for Ac_1. The precision of temperature measurements was probably slightly better in the former case, although the temperatures in the microscopic investigation are strictly comparable among themselves.

Attempts to determine Ac_1 stable by noting the highest temperature at which ferrite would be formed on reheating a partly graphitized and quenched specimen of hard iron indicated a figure approximating 745° C. Microscopic inspection of specimens made by Austin and similar in composition to the hard iron but of various lower carbon contents indicated that the eutectoid composition lies between 0.86 and 0.99 per cent. carbon, estimated metallographically at 0.92 per cent. The Ac_m line could not be followed. The results of the determinations of the agraphitic carbon content of stable alloys in equilibrium with free carbon at various temperatures are given in Table 1, which shows the solubility of carbon in the stable alloys and indicates that for the stable system A_1 is slightly above 750° C.

We believe that the study of electrical resistance as an expression of the graphitizing reaction is novel. It is a most useful means of checking conclusions based on other methods for determining critical points, differentiating between various solid solutions and investigating the mechanism of graphitization.
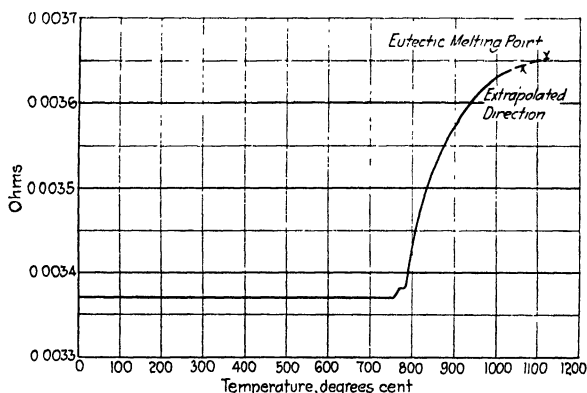
FIG. 4.—RESISTANCE AT 22° C. OF MALLEABLE IRON AFTER COOLING RAPIDLY FROM VARIOUS TEMPERATURES.

TEMPERATURE-RESISTANCE CURVE OF WHITE CAST IRON

The temperature-resistance curve of white cast iron (on heating) shows discontinuities at 280°–290° C. and 810°–820° C. The location of the latter point suggests that it is an expression of the alpha-beta transformation. On the graphitized material a similar curve shows a break at about 790° C., while one taken during very slow cooling diverged

from the preceding in the direction of greater resistance at temperatures above 710°–720°; the discontinuities, however, were hardly sharp enough to be distinctive in any of the three cases.

If a completely graphitized specimen is heated to successively higher temperatures and cooled fairly rapidly after each heating, the Ac_1 (stable) point should be indicated by a permanent change in resistance. In Fig. 4 is plotted the resistance at room temperature (23° C.) of a specimen as related to the temperature of reheating. An extremely sharp break is found at 760° C.; above this point, the curve suggests a solid solution of progressively increasing carbon concentration and not the redissolving of eutectoid carbon. It may be that the time of heating was insufficient to secure homogeneity at temperatures above A_1, so that we were measuring the resistance of a mechanical mixture of solid solution and ferrite at the lower temperatures. Any decomposition of the stable system into the metastable, because of insufficient quenching, is not significant in this connection where only the form of the curve and its point of inflection are involved.

The ratio of the hot to the cold resistance of the specimen shows great discontinuity between 850° and 870°, the significance of which we have not studied. It is merely suggested that this may represent the alpha-beta or beta-gamma transformation. The specific resistance (20° C.) of a number of ferrous alloys has been determined as follows:

	Specific Resistance	Temperature Coefficient (20°–100°)
Armco iron	10.9×10^{-6}	0.0047
Eutectoid steel (quenched from 1700° F.)	33.1×10^{-6}	0.0017
Malleable (ferrite and temper carbon)	29.5×10^{-6}	0.0021
Malleable (quenched from 1700° F.)	34.9×10^{-6}	0.0016

The agraphitic carbon content of the quenched malleable was in excess of the eutectoid ratio of the metastable system; equal carbon concentrations in solid solution in the two cases have radically different effects on the electrical resistance. Approximately 0.9 per cent. carbon raises the specific resistance 22.2×10^{-6} ohms, or over 200 per cent. of the carbon-free resistance in the metastable system; and less than 5.4×10^{-6} ohms or about 35 per cent. in the stable system. In the first case, the effect of this agraphitic carbon content is to reduce the thermal coefficient by nearly 67 per cent.; in the second, by less than 25 per cent. Further work on the resistance of iron-carbon alloys has been done but the results are reserved for publication in other connections and do not bear on the present problem.

CHANGES OF RESISTANCE AT CONSTANT TEMPERATURE

. We have observed the changes of resistance during the process of graphitization at constant temperature; success in this experiment is

possible only if great uniformity of temperature with respect to time and throughout the specimen can be attained. These precautions being taken and stray thermal electromotive forces eliminated, changes of resistance with time were observed as plotted in Fig. 5.

There is a decided drop in resistance, but in 2 or 3 hr. a minimum is reached and the resistance rises, apparently approaching a maximum asymptotically. If, the reaction being over, the temperature is lowered, the resistance falls with temperature and then with time, approaching
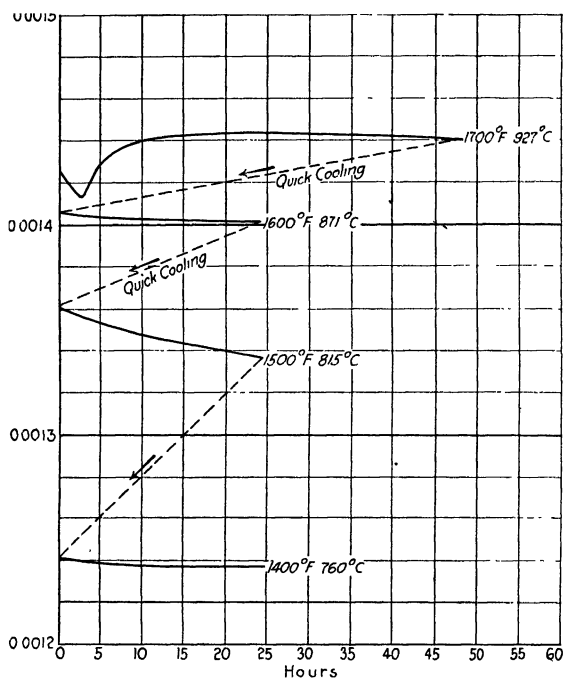
FIG. 5.—CHANGE OF RESISTANCE AT CONSTANT TEMPERATURE OF WHITE CAST IRON DURING GRAPHITIZATION.

a minimum asymptotically; this is repeated with each successive fall in temperature, presumably to Ar_1. Microscopic examination indicates the disappearance of free cementite at approximately the minimum point on the first curve although the exact coincidence has not been demonstrated.

RATE OF GRAPHITIZATION

Schwartz, in April, 1915, had occasion to measure the rate of graphitization at constant temperature as a function of the concentration and found that, once initiated, this rate remained nearly constant for quite an interval and began to fall only when the concentration of agraphitic carbon approached values corresponding roughly to the carbon ordinate

of the Ac_m line at the given temperature. The relationship was not studied further as it had no bearing on the purpose for which that investigation was being conducted.

Payne has observed, qualitatively, that free cementite graphitizes extremely rapidly, compared with the white iron from which it was isolated.

Mrs. Anna (Nicholson) Hird, in connection with an investigation of the composition of cementite, by accident studied a specimen in which very little graphite had been formed. It was shown analytically that this carbon had been formed apparently at the expense of the originally eutectoid and procutectoid cementite. A detailed discussion of her work is impossible at this time; if brought to a successful conclusion it may be published as a separate paper.

In the microscopic examination of many specimens of hard iron in which graphitization is just beginning, we have observed that the first evidence of the formation of temper carbon is never entirely within a cementite grain, is very rarely in contact with a cementite grain, and is usually entirely within the austenite (mix crystal). Phillips and Davenport[4] present micrographic evidence to this effect. We, in common with others,[5] have frequently observed that when graphitization proceeds below A_1, the temper carbon is entirely surrounded by ferrite. It is common knowledge that the distribution of temper carbon in malleable iron in no way suggests the dendritic structure of the original hard iron.

AUSTENITE AND BOYDENITE

We believe that the investigation just outlined justifies the conclusion that the solid solutions of the stable system are distinct in structure, electrical properties, and temperature-solubility relation from those of the metastable system. For this new stable solid solution, we propose the name boydenite, in commemoration of the metallurgist who first practiced the graphitizing reaction.

As boydenite differs from austenite (mix crystal) of equal carbon concentration in every measurable characteristic, they cannot both be solid solutions of either cementite or carbon. The most logical conclusion appears to be that austenite is a solution of cementite and boydenite a solution of carbon; though we are aware of Jeffries' and Archer's opinion[6] that cementite is not in solution as such. The evidence is based primarily on the relation between the size of the cementite molecule and of the iron atom. We would not wish either to affirm or deny this conclusion, feeling

[4] Malleableizing of White Cast Iron. *Trans.* (1922) **67**, 466.

[5] Merica and Gurevich, *loc. cit.* Bean, Highriter and Davenport. Amer. Found. Assn. *Trans.* (1921) **29**, 306.

[6] *Chem. & Met. Eng.* (1921) **24**, 1057.

that our own knowledge of the mechanism of solid solution is as yet most rudimentary.

A conception that austenite, in a generic sense of solid solutions in gamma iron, is a solid solution in equilibrium with, though not of, cementite and that boydenite is a similar solution with respect to carbon would be equally acceptable. The solutes in austenite and boydenite, however, are apparently not the same. Our experiments seem to indicate that carbon migrates in boydenite more rapidly than in austenite, although at low temperatures the rate of migration even in the former is not high.

In view of Cesaro's demonstration[7] that molten iron is a solution of Fe_3C in Fe_2 and of the fact that cementite separates from the frozen solution in further cooling, we adhere to the opinion that the solution in austenite cannot well be free carbon, especially as austenite and boydenite could not both be solutions of carbon in the same form of combination.

Some discrepancies exist as to the critical points determined by various methods. The results by the thermal method differ somewhat from the concordant results of the other methods; this we ascribe in part to lag (*i.e.* super-cooling or super-heating) and more largely to the difficulty of deciding what constitutes the beginning of the break in the differential-temperature curve. All the evidence, however, agrees that A_1 metastable is below A_1 stable; the difference between the two temperatures is probably between 5° and 10° C., the thermal data for Ac_1 furnishing the best available evidence as to the magnitude of this difference.

We conceive of the graphitization taking place in the following stages: At fairly high constant temperatures, say 900° to 1000° C., the process is initiated by the separation of a graphite nucleus, possibly due to carbon precipitated out by some local conversion of austenite into boydenite and excess carbon. The bulk of the solid solution, however, remains austenitic and maintains its carbon concentration by dissolving cementite. More carbon precipitates and more cementite dissolves until the latter is used up, when there remains a matrix of austenite and grains of free carbon. This reaction is relatively rapid. Subsequently the austenite is graphitized into boydenite and free carbon and reaction ceases when the solid solution is boydenite of a carbon concentration corresponding to the saturation point at the temperature chosen. If the temperature falls slowly enough, carbon precipitates along the At_m line, the boydenite decreasing in carbon content. Just above Ar_1 the solubility of carbon in boydenite is about 0.50.

The agraphitic carbon becomes almost nil below Ar_1 stable and ferrite is formed, the remaining carbon precipitating as a ferrite-carbon eutectoid. As a matter of fact, carbon is probably soluble in ferrite so that

[7] *Jnl.* Iron and Steel Inst. (1919) **99,** 447.

the carbon concentration at Ar_1 has a small finite value; in this case it was less than 0.03 per cent. If the temperature falls too rapidly to permit the boydenite to decrease its carbon content by precipitation, austenite will be formed, the metastable mix crystal having a higher carbon concentration than the stable. If the temperature falls somewhat rapidly below Ac_1 metastable pearlite may form and graphitization will still proceed, though very slowly because of the slight carbon solubility at that temperature. The ferrite ring around the temper carbon under these conditions, which is really a ring of boydenite of extremely low carbon content, suggests the rapid migration of carbon when dissolved as such.

In the light of the facts at our disposal, we believe that the stable-equilibrium diagram of the alloy in the area studied is probably as shown in Fig. 6, the probable metastable diagram being shown by the solid line. The At_m line is always to the left of Ac_m, but differs from that suggested by Archer by departing materially from the latter in the lower temperatures. It differs from the form tentatively suggested by one of the present authors, in the apparent existence of a eutectoid of considerable carbon concentration instead of the At_m line approaching very near to nil carbon at A_1. We have never seen microscopic evidence of the existence of a eutectoid, and it is

FIG. 6.—STABLE AND METASTABLE EQUILIBRIUM DIAGRAM FOR HYPOEUTECTOID CARBON-IRON ALLOYS CONTAINING ±1 PER CENT. SILICON.

possible, though not probable, that there is a sharp inflection in At_m just above A_1 permitting the lower end of At_m to bend over and reach nil carbon at A_1. We are seeking evidence in this direction but have found none. Work is also in progress toward determining the effect of other elements on the relative positions of the eutectoid ratio in both systems and also on the mechanism on which the initiation of graphitization depends. The latter evidence is being sought in the direction of a study of the equilibria in the ternary system Fe-C-Si.

DISCUSSION

E. S. DAVENPORT, Bloomfield, N. J. (written discussion).— New and interesting data on the problem of graphitization are found in this paper and the authors are to be congratulated on their application of electrical-resistance measurements to the study of the graphitizing reaction. The designation At_m line for the graphite-austen-

ite equilibrium line is, perhaps, to be regretted, for the term "temper carbon" is not widely used outside of the malleable-castings field. Inasmuch as x-ray work has demonstrated the identity of temper carbon and natural graphite, would it not be better to call it the Ag_r line, particularly as this term is already in use to some extent?

The critical-point curves for the original white cast iron seem to have been omitted from Fig. 1. With the exception of Ac_1 the white cast iron, the critical points obtained by the authors agree quite well with those obtained by Bean[8] and Phillips and Davenport[9] on similar material. The points obtained by the authors on the stable alloys, heatings subsequent to the first, are undoubtedly the metastable points.

The micrographic evidence supporting the theory of the existence of boydenite is interesting. Of particular interest is the statement that "long-continued heat treatment at 900° or 950° C. will convert the structure from that characteristic of the temperature in Fig. 2 to that of Fig. 3." If this could be accomplished with specimens covering a range of compositions, it would constitute a strong argument in favor of boydenite. However, martensite is one of the most difficult to etch of all ferrous structures. Slight variations in etching technique are to be carefully guarded against, as such variations might well cause the apparent differences in structure when only a few specimens are being examined.

On the other hand, the specific-resistance data may be interpreted as evidence against the existence of boydenite. The eutectoid steel and the malleable, quenched from the same temperature, have specific resistances of the same order of magnitude (33.1×10^{-6} and 34.9×10^{-6} respectively). The temperature coefficients also show a similar agreement. Is it not the simplest course to assume that the same constituent is present in both alloys? The slight discrepancies may be attributed to the excess temper carbon, over the eutectoid ratio of the metastable system, present in the quenched malleable. It would hardly be expected that Armco iron and malleable (ferrite and temper carbon) would have specific resistances or temperature coefficients of the same order of magnitude. It is not clear why any great importance should be attached to the amount of change produced by the heating and quenching operations when the original reference materials were not strictly comparable.

More work must be done before the existence of boydenite can be conclusively proved. The theory will be of great value however in stimulating further investigation and discussion. An x-ray examination of distinctly martensitic and "boydenitic" irons should yield enlightening data.

Table 1 contains valuable data on the graphite-solid solution equilib-

[8] W. R. Bean: Discussion of H. A. Schwartz, Some Physical Constants of American Malleable Cast Iron. *Proc.* A. S. T. M. (1919) Pt. 2, 247.

[9] Malleableizing of White Cast Iron. *Trans.* (1922) **67**, 466.

rium line. If, based on these data, the At_m line in Fig. 6 is correctly located, we have yet to account for the graphitization of the remaining 0.5 per cent. carbon in completely annealed malleable. As there is no evidence at hand indicating the existence of a graphite eutectoid, either the At_m line turns sharply to the left just above A_1, as suggested by the authors, or we must assume that approximately one-fifth of the temper carbon in malleable is precipitated at temperatures below A_1. A close investigation of this line in the vicinity of A_1 is desirable.

Regarding the mechanism on which the initiation of graphitization depends, we venture to call attention to our suggestion in an earlier paper, that slight fluctuations in temperature during the "soaking" period of the anneal might be the cause of the initial precipitation of temper carbon.

H. A. SCHWARTZ (authors' reply to discussion).—The structures shown in the photomicrographs have been consistently observed in many experiments; they are not apparently the result of peculiarities in the etching methods. Assuming that both boydenite and austenite are solutions in gamma iron, it seems reasonable that the "chevron" structure corresponding to cleavage planes in the original solid solution should be preserved as a pseudomorph in cooling.

In the matter of electrical resistivity we reason thus: The resistivities of malleable and Armco irons are far different, because of the presence of free carbon in the former; therefore, the resistivities of quenched malleable and quenched eutectoid steel should be quite different. Actually they are nearly identical. Hence the resistivity of the metallic matrix in the former must be enough lower than that of quenched eutectoid steel to balance the increased resistance caused by the presence of carbon, or of the voids it formerly occupied. This difference constitutes the point of our experiment.

Although x-ray investigations on boydenite and austenite have been attempted, they have yielded no conclusive data.

The nomenclature At_m was adopted, possibly ill advisedly, to suggest a difference between material containing primary graphite and one containing secondary graphite. We are not sure whether the exterior form of this phase enters into the solubility. The term "temper carbon" has, of course, an association with the malleable industry. It was coined, however, by Ledebur to define the particular form of free carbon with which we were concerned; hence our selection of terminology.

Effect of Nickel-chromium on Cast Iron

By Richard Moldenke, E.M., Ph.D., Watchung, N. J.

(San Francisco Meeting, September, 1922)

The presence of nickel-chromium in pig iron made from Cuban ores is so well known that the word "Mayari" is unconsciously associated with it. Since 1904, when exploration of the Mayari deposits of iron ore in northeastern Cuba was undertaken, there has been increasing interest in these ores. The Transactions contain numerous references to them.[1] This iron ore, which is shipped to the United States in the nodulized state, has been thoroughly tested and found very valuable; the deposit covers over 25,000 acres and there is in sight over 500,000,000 tons of ore that, by systematic mining, runs practically constant in composition. With nickel and chromium present, also titanium in sufficient quantities to attract attention, as well as extremely low phosphorus and sulfur contents, the iron founder is naturally interested in the utilization of Mayari ore for pig iron for the better grades of finished iron products.

The present point of consumption of Mayari ore is at Sparrow's Point, where Mayari pig iron of any desired composition is made. The composition of the dried ore is about as follows: Iron 49 per cent., silica 3 per cent., nickel 1 per cent., chromium 1.5 per cent., alumina 11 per cent., and combined water 13 per cent. When nodulized, the iron runs up to nearly 56 per cent., the silica, alumina, nickel, and chromium are slightly higher than the amounts just given. Practically all the combined water is driven off and the nodulized ore carries only a few tenths per cent. of

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[1] C. M. Weld: Residual Brown Iron Ores of Cuba. *Trans,* (1909) **40**, 299.

    C. K. Leith and W. J. Mead: Origin of Iron Ores of Central and Northeastern Cuba. *Trans.* (1911) **42**, 90.

    C. W. Hayes: Mayari and Moa Iron-ore Deposits in Cuba. *Idem,* 109.

    J. E. Little: Mayari Iron Mines, as Developed by the Spanish-American Iron Co. *Idem,* 152.

    J. F. Kemp: Mayari Iron-ore Deposits. *Trans.* (1915) **51**, 3.

    C. K. Leith and W. J. Mead: Additional Data on Origin of Lateritic Iron Ores of Eastern Cuba. *Trans.* (1915) **53**, 75.

carbon. Careful furnacing of this ore has enabled the production of a silicon range quite within regular foundry requirements, so that the use of Mayari pig iron has been extended to the making of such light castings as automobile piston rings, exhaust valve cages, etc.

On account of the chromium content of the ore and the large amount of slag produced in smelting it, a high fuel ratio is required. All of the nickel and practically all of the chromium are reduced, which results in a high-grade iron, quite free from oxidation, and with what might be termed a "charcoal-iron value" to the foundryman. Foundry melting processes are oxidizing in character, hence metal to go into cupola or air furnace charges cannot be too free from previous oxidation, as the dissolved iron oxide in the molten foundry metal cannot be removed as readily as in the case of steel. It is essential, therefore, that the foundry-man use as much pig iron in his mixture as he can afford and yet compete; further, that this pig iron be an "honest" one—that is, made without skimping the fuel in the blast furnace. The more remelted stock used—such as the several purchased scrap varieties, the gates, sprues, defective work, and "over iron" of the plant—the greater the internal shrinkage troubles, lost castings, and inferior work produced. Foundry-men, therefore, tie to irons they know are reliable, and these on investigation will be found only where pure ores and abundant and good fuels are used. The fact that nickel, chromium, and titanium are present in this iron is a guarantee of careful smelting.

COMPOSITION OF NICKEL-CHROMIUM IRON

These nickel-chromium irons run high in total carbon. For instance, in a series of analyses of Mayari pig iron, with silicon running from 0.38 up to nearly 2.00 per cent., the total carbon remains above 4.00 per cent. and goes as high as 4.50. This is a favorable indication, for when foundry-men have trouble from their pig iron, the iron will usually be found low in total carbon, say, between 2.50 and 3.00 per cent. instead of between 3.25 and 3.75. The nickel content in Mayari pig iron is practically half that of the chromium at all times. A typical analysis is as follows:

	Per Cent.		Per Cent.
Total carbon	4.25	Nickel	1.20
Silicon	0.50 to 3.00	Chromium	2.40
Manganese	0.90	Titanium	0.18
Phosphorus	0.065	Vanadium	0.05
Sulfur	0.03		

The natural classification of this iron is with reference to the silicon content (low silicon and high silicon), as there is little variation in content of the other elements.

The nickel-chromium content naturally has an influence on the physical structure of the pig iron. Being cast into iron molds, or machine-cast, the lower silicon pigs would be dead white in fracture, and the higher silicon irons at least a very close-grained gray, if not mottled in the center and white at the edges. The low-silicon pigs crack easily and are remarkably dense, the specific gravity being from 7.28 to 7.30 as compared with 7.20 to 7.24 for ordinary iron. This condition will be understood better when it is remembered that nickel promotes the formation of graphite in cast iron, while the action of chromium is just the reverse.

If it is desired to increase the depth of the chill in a casting when the test piece shows it is too light, ferrochrome should be added to the ladle of molten iron. But with Mayari pig iron there is a natural marked tendency to chill the casting; the effect of the chromium is dominant because twice as much chromium as nickel is present. For this reason, also, only small percentages of Mayari pig iron should be used in the softer varieties of mixtures, in order to get the strength imparted by the nickel and chromium without sufficient added hardness to interfere with easy machining. The results of the tests described later graphically show the situation.

One interesting feature of the Mayari ores is the fact that nickel and chromium, or valuable metals other than the usual manganese, are introduced into pig iron directly from the ore in uniform distribution, and should therefore be more efficient than when these metals are added to molten iron as expensive ferroalloys, not only because of the impossibility of distributing metals of higher melting points than cast iron uniformly in that medium by simple addition, but also because of the loss of a percentage of the ferroalloy by oxidation and absorption by the slag when added in the ladle.

The fundamental conception of gray-fracture cast iron as a steel plus mechanically mixed graphite indicates that for the same graphite content, equally sized and distributed, any improvement in the steel portion by alloying with nickel, chromium, titanium, vanadium, etc. must mean a high-grade product. Introduction through the ore has the advantage of regularity of results through uniformity in distribution.

In the early days, Mayari pig iron low in silicon, with all the carbon in the combined form, and the pigs brittle from casting strains, was hardly considered desirable foundry iron. However, in the search for new markets for this high-grade iron, the roll-maker was induced to try it, and so gradually the founders of the lower silicon ranges of cast iron got to know the value of nickel-chromium in their mixtures. Its use should, however, be extended, for the transportation systems of the country are demanding better cast-iron car wheels and the producers of crushed stone and the users of ball mills would not object to a longer

life for the wearing parts of their machinery. Hence, improvement of the steel portion of cast iron by alloying, for heavy-duty machinery, should be encouraged, and new avenues found where a smaller weight of higher quality iron means an industrial advance for the nation.

To obtain definite information on the action of nickel-chromium in the Mayari pig iron derived from their Cuban ores, the Bethlehem Steel Co. retained the author to lay out and conduct a series of tests. By
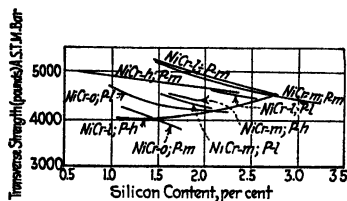
FIG. 1.—TOTAL CARBON, LOW.
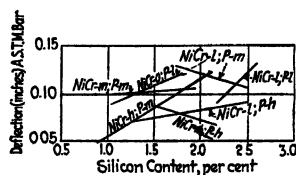
FIG. 4.—TOTAL CARBON, LOW.
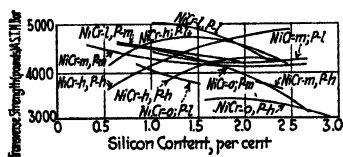
FIG. 2.—TOTAL CARBON, MEDIUM.
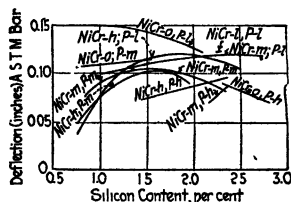
FIG. 5.—TOTAL CARBON, MEDIUM.
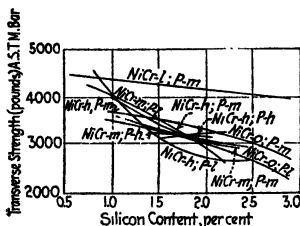
FIG. 3.—TOTAL CARBON, HIGH.
TRANSVERSE TEST CURVES.
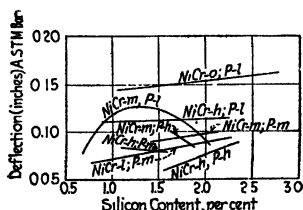
FIG. 6.—TOTAL CARBON, HIGH.
DEFLECTION TEST CURVES.

thus going outside of its own organization, the element of self-interest was eliminated; for the company desired that the foundry industry should be given unbiassed information for the benefit of consumer and producer. The method of planning and carrying out the work and the conclusions derived from these tests (probably the most extensive so far made in the foundry industry) are here given. The work occupied more than a year and the cost of the detail shop and laboratory

operations was large, for which the Bethlehem Steel Co. should be given credit by those interested in cast iron and its uses. The engineer who designs any cast-iron articles and the foundryman are both interested in the information given by the curves in Figs. 1 to 12.
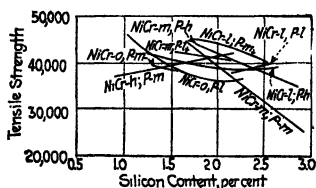
FIG. 7.—TOTAL CARBON, LOW.
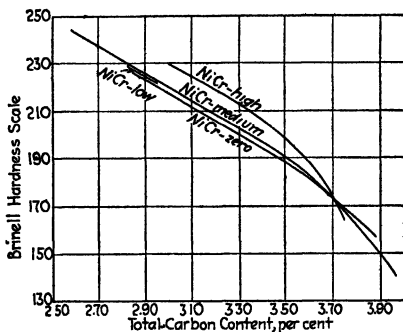
FIG. 10.—PHOSPHORUS, LOW.
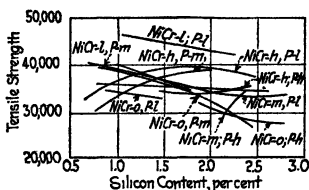
FIG. 8.—TOTAL CARBON, MEDIUM.
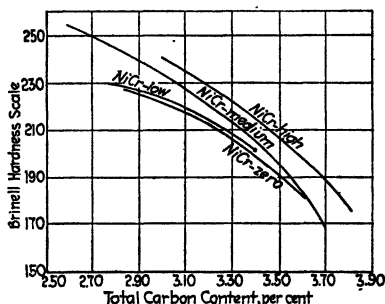
FIG. 11.—PHOSPHORUS, MEDIUM.
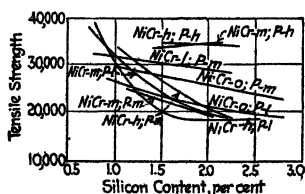
FIG. 9.—TOTAL CARBON, HIGH.
TENSILE TEST CURVES.
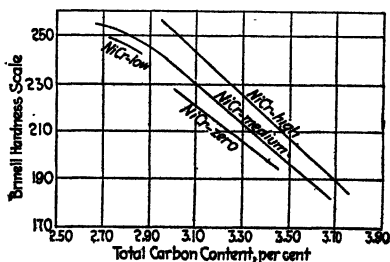
FIG. 12.—PHOSPHORUS, HIGH.
BRINELL NUMBER CURVES.

USES OF NICKEL-CHROMIUM IRON

Rolls, Car Wheels, Crushers, etc.—Low-silicon mixtures are essential for these products, as heavily chilled surfaces are wanted. The castings are subjected to extreme bending and crushing stresses, and the machining problem is exceptionally difficult. Strength and ability to resist wear are the essential features of the design. The following ranges of composi-

tions would be indicated: Silicon, below 1.50 per cent.; total carbon, low to medium; phosphorus, low to medium; nickel-chromium, low to medium. Where exceptional hardness is needed and strength is provided by increasing the section sufficiently, the total carbon and phosphorus should be low and the nickel-chromium high.

Strong gray-iron castings, just machinable without too much difficulty. This work is subject to bending and reciprocating stresses, as in engine frames, and the iron should be strong and resilient; hence a low total carbon through steel additions to the mixture, with silicon from 1.00 to 1.50 per cent., or just enough to hold the combined carbon to where machining is possible. Phosphorus should be low to medium and nickel-chromium low to high.

Strong, but Soft, Gray-iron Castings.—This work is not subject to severe bending or reciprocating stresses. The Brinell hardness must be fairly low, so that a high total carbon is essential, with silicon from 2.00 to 2.75 per cent. (the low silicon for thick sections and the high silicon for thin work). Phosphorus should be low to medium and the nickel-chromium in the low ranges.

Piston rings, and light work with much elasticity. Here hardness must be avoided and the Brinell numbers should be low. As, however, the work must be springy enough not to set when bent, a medium total carbon is better than a high one. Good deflections indicate a low to medium phosphorus; and to machine well a carbon condition resulting from high silicon is best. The nickel-chromium content should be medium to high.

The foregoing are but a few of the many lines of cast-iron products. In whatever branch of the industry nickel-chromium may be applied, the daily routine should be checked up carefully with the standard test-bar procedure. This will soon give a series of values that will allow the selection of the proper range of composition adapted to the work in hand.

EFFECT OF REMELTING MAYARI PIG IRON IN CUPOLA

The chromium content of the Mayari pig irons is double that of the nickel, but when remelting in the cupola, the nickel is hardly affected while the chromium is oxidized and removed to some extent. The analyses of the test bars show this plainly—the nickel persisting where the chromium is gone entirely. In the general run of results, the chromium has been cut in half and about equalized with the nickel; only in the higher nickel-chromium ranges are the ratios found more nearly normal. This condition is important and shows that in the repeated remelting of the daily sprues, gates, and defective work the nickel content becomes cumulative while the chromium is decreased, with corresponding higher strengths and lessening hardness. This result is noted in the foundry,

for the best improvements from nickel-chromium additions show themselves only after the daily remelt has been thoroughly impregnated with these elements. A pig iron made with a superabundance of fuel can always stand the effects of poor cupola practice better than one made without this advantage. In the latter instance, the iron cannot stand further oxidation without interior shrinkages, pin and gas holes, cracks, and other troubles due to iron without "life." The excellent quality of charcoal irons can be attributed to perfect burning of the fuel in the furnaces making them. Similarly, Mayari pig iron, because of the high fuel ratios required to effect the reduction of the nickel, chromium, titanium, and vanadium from the ores, produce a good fluid metal, full of life, and capable of running the thinnest castings with less than 0.1 phosphorus. This means clean iron free from entrained slag and giving good castings that have no hard spots, are of even and fine-grained structure and hence wear well.

Mayari pig iron was first used in the production of chilled rolls, and here the effect of composition and manufacture can be plainly traced. The added strength and elasticity given by nickel, the greater depth of chill and hardness produced by chromium, the fine-grained structure making for smoothness and polish due to clean metal and freedom from excessive oxidation gave rolls of longer life for heavy duty, as well as doubling the life of finishing rolls through the increased resistance to abrasion and freedom from fire-cracks. Nickel-chromium irons are now used in grinding plates for feed mills, flour-mill machinery, stone crushers, piercing plugs and die bushings for projectile work, mill guides, wire-drawing dies, bending blocks, pipe balls, balls for tube mills, bottom plates for grinding mills, sand-blast nozzles, etc. One place where excellent service is given is in the chilled cast-iron car wheel.

In the case of heat-resisting castings, the nickel-chromium content together with freedom from oxidation results in longer life under exposure to high temperatures. Such castings as glass molds, hot-gas valves and bells for gas producers, stoker parts, grate bars, superheater headers, lead melting pans, and furnace castings last much longer than when of the ordinary irons. Unquestionably the low phosphorus content helps in this, but the breaking up of the graphite crystals by the chromium present tends to check the inward injurious action of heat, such as is observed where the graphite crystals are large, as in Outerbridge's experiments in "growing" gray cast iron. Incidentally, the nickel content increases the strength of the casting.

Greater acid-resistance follows the use of chromium in cast iron, the chemical industry being well supplied with castings containing this element. Similarly, where heavy work is made of metal from both cupola and air furnace, nickel-chromium in the iron is a valuable addition. In general, the greater freedom from oxidation of well-made

irons, when these are melted properly, results in a longer period of fluidity of the metal before setting, as compared with irons made in blast furnaces that are pushed very hard. The consequence is that the slower the molten metal sets in the mold, the better it feeds up through gates and risers with corresponding increase in soundness. Nickel-chromium in such well-made irons gives the additional advantage of high strength to the work.

DESCRIPTION OF TESTS AND DISCUSSION OF CURVES AND TABLES

At first, the tests were made at independent foundries. In laying out a series of tests with Mayari pig iron—using the word "Mayari" as a brand name, just as in the iron industry pig irons have been given various names by those making them—the object was to differentiate the effect of the nickel-chromium content of the iron from that of the other elements present in the usual range of composition of cast iron. Considering that quite a number of chemical and physical factors are present and have varying influences on the end result of a melt, the problem was not easy.

In the first place, the tests must bring out facts easily interpreted by the foundryman. He can understand the breaking weight and deflection incident to the transverse test of standard test bars. He cannot always, however, fairly judge wearing quality, relative machinability, and properties on which there may be honest differences of opinion. Hence the tests selected consisted of transverse (with deflection), tensile strength, Brinell, detailed chemical analysis, and the preparation of photomicrographs on a sufficient number of standard A. S. T. M. bars made under standard conditions of practice. Inasmuch as one reason why it is impossible to predict the strength of cast iron from its composition is the change in physical structure resulting from varying proportions of pig iron and remelted metal, this point had to be standardized. It was done by not only adopting a straight 60 per cent. pig to 40 per cent. scrap relation, but by making this scrap. This meant the melting of a standard grade of pig iron in the cupola and pouring the metal into pigs to serve as the scrap of the mixtures used. This was done in 5-ton batches to secure uniformity in product for the make-up of the mixtures.

In planning the mixtures themselves, not only were the silicon, manganese, sulfur, phosphorus and total carbon taken into consideration, but also the nickel-chromium content. The usual foundry procedure was followed. The foundryman fixes his phosphorus first—whether low, medium, or high, depending on the class of work he is to make; in the mixtures, it was aimed to have one set of test bars with phosphorus running around 0.20 per cent., a set with about 0.60 per cent., and a third set with 0.90 per cent.

The sulfur was held as nearly between 0.10 and 0.12 per cent. as possible, and manganese around 0.60 per cent. These elements need not, therefore, require special attention in the discussion.

Next, it was necessary to fix the total carbon limits, so that several divisions in total carbon could be made; this was accomplished by steel-scrap additions. In carrying out the plans, there was first a series of straight pig-and-scrap mixtures without steel. Next, these mixtures with 10 per cent. of their scrap replaced by steel; and then by 20 per cent. of steel. This was to give three sets of total-carbon divisions. Afterwards, in actual practice, it turned out that the varying total-carbon determinations obtained necessitated the division of these results into three groups of low, medium, and high total carbon, respectively.

The succeeding step was to provide for the use of varying proportions of Mayari pig iron in the mixtures. This was accomplished, in a similar manner, by taking the mixture for every total càrbon as part of each division of phosphorus first without Mayari iron, then with 10 per cent. Mayari forming part of the pig irons used, then 15 per cent., and finally 20 per cent. Mayari pig. Here, also, it was necessary, after the tests were made, to divide the results into four general groups; namely, iron with no or very low nickel-chromium; iron with low, with medium, and with high nickel-chromium content.

Last of all, there had to be a grouping by the silicon content from 0.50 to 3.50 per cent., in steps of 0.25 per cent. As the silicon content in greatest measure affects the division of the total carbon into graphite and combined carbon, this was the most particular of the divisions and, with the exception of the Brinell hardness results, all curves were oriented in accordance with this silicon range. To sum up: There were four nickel-chromium divisions for each total carbon and three of these (or 12 altogether) to each silicon. There were 13 silicon divisions for each phosphorus group, or a total of 465 tests as planned. It was planned to make repetition tests with Mayari impregnated sprues twice in each case, to compare with the original material, but unfortunately this plan could not be carried out.

The molten metal in each case was to be poured into the standard A. S. T. M. 1¼-in. diameter test bar vertically with top pour in dry-sand molds, three bars to the mold. Arrangements had been made in a number of foundries for these tests with those of the mixtures that would fit the class of work made at these plants. To reduce the number of separate heats for so large a program, the regular day's run of the foundry was to be preceded by five separate charges of different mixtures, with 50 per cent. more coke between each than required. When, however, the fourth foundry failed to give results sufficiently in line with what should have been the case, it was recognized that in spite of the excellent cupola

practice prevailing in each of these plants, cupola melting was not as accurate as it was supposed to be. The results of these tests have been given in another paper as they were of more especial interest to foundrymen. By this time the importance of having separate melts was realized, and as the entire crucible department of the Bethlehem Steel Co. was not in operation at the time, one of the large regenerative furnace units, which cares for twenty-four crucibles at a time, was given over for this research; also some of the mixtures were eliminated where no immediate benefit to the foundry industry could be seen. Among these were the low-silicon, high-phosphorus mixtures. On the other hand, as the crucibles had to be over one-half full (or with about 60 lb. metal) for satisfactory service, double the number of molds were made, and six test bars cast for each melt instead of three as planned. This made the results more reliable.

As the crucible-melting process is the best (as well as most expensive) and as the mixtures were weighed and run through by operatives expert in such work, the results are more comparable than would have been the case with cupola iron. So well protected were the charges within the crucible that, in the case of the first day's run, when it became necessary to let four crucibles in the end pit remain over night because the metal was considered too cold to be properly comparable, this metal, though in the molten state 16 hr., was carefully traced through the chemical and testing laboratories and found as good as the rest.

As previously stated, the scrap used was remelted pig iron, of bessemer analysis. Ferrosilicon and ferrophosphorus were used to make up what silicon and phosphorus were required. The Mayari pig iron gave the nickel-chromium, and steel clippings cut the total carbon. The remaining pig iron required was taken from the regular low-phosphorus furnace product of the plant. The regular routine of the crucible melting process was observed: the crucibles were heated after charging before going into the pits, the melting was observed from time to time, and when ready, the crucibles were drawn, shaken down a little, and poured. The cores for three bars each were placed vertically, side by side, in long rows on the platform adjoining the furnace, and held rigid by steel beams with sand rammed up where needed. The test numbers were chalked on the crucible, floor, and core as the process went on, and as the molds were shaken out the bars went into numbered pans and were taken immediately into the grinding room for cleaning and stamping, so that no lots would be mixed. Thence the bars went to the laboratories of the plant.

The pouring temperature was necessarily high—not far from that of molten steel in appearance, though not hot enough to burn the molding sand used. Comparatively few of the bars were defective. The stand-·ard A. S. T. M. bars were $1\frac{1}{4}$ in. in diameter and 15 in. long. They were placed upon supports 12 in. apart and broken by applying pressure

at the center. The deflection was noted at the moment of rupture. The broken piece of the bar representing the lower end as it stood vertically in the mold—poured from the top—was then put into the lathe and a cut made across the fractured surface, the turnings being carefully caught below. These turnings were ground and mixed, *in toto*, by the standard methods and analyzed for the constituents shown in the table. A further piece was then sliced off the broken test bar to serve for the Brinell hardness test, two impressions being taken half way between center and rim and the average reported in the table. Before cutting up the test bar, however, the two diameters were carefully calibrated—the vertical one in position of test and the horizontal—so that the breaking weight could be properly corrected. Care was taken in the first place to put the bar on the supports for breaking so that the smallest diameter was the vertical one. The remainder of the broken bar was then turned up for the tensile test piece, according to the A. S. T. M. standard, and then pulled in the machine.

All the bars were broken transversely and their deflection was taken. Hence the data given in the table for transverse strengths are the average of six bars in nearly all cases. But one bar of each mixture was subjected to the tensile test, analyzed and the Brinell hardness taken. Also only one micrograph was made from the bar, showing the average character of the metal across the surface.

Even with all the advantages of the crucible process for accuracy in mixture making, the results, on analysis, indicated compositions varying widely. It took much time and patience to regroup the results, so that an orderly sequence might be established. A summary of the results is given in Table 1.

These results were obtained by checking off each determination separately against a table giving the entire series of the particular element as gotten in the tests. The bunching of the check marks easily differentiated the high points from the scattering, and hence the ranges given became available.

It was now possible to group the results as originally intended, and Table 2 gives the general groups of low, medium, and high phosphorus each in combination with a similar condition in total carbon, respectively; and each of these sets in combination with either no, low, medium, or high nickel-chromium, respectively, as far as these checked with the actual results. Further, this grouping made it possible to construct sets of thirty-three separate curves each, for the transverse, deflection, and tensile results, each with the silicon range as the basis. The phosphorus, total carbon, and nickel-chromium are standardized individually as shown in the curves. The Brinell curves form the exception to the above arrangement, as they show no relation whatever to the silicon range. Lack of space required the grouping of

these curves to a considerable extent, but careful scrutiny will reveal the characteristics without difficulty.

A few compression tests were made at the time, on the full-sized section, with columns 1½ in. long. The results are given in Table 3. In connection with the diameters of the test bars, it may be said that even with every care taken to make strong molds, when these are grouped in large numbers they open up slightly under the strain of the molten metal, hence the bars come out a little large. Instead of 1.25 in. diameter, they may run to 1.30 in. If a heavy coat of blacking has been applied, the bars may run light—down to 1.20 in. diameter. Hence the necessity of careful calibration for the vertical and horizontal diameters of the test piece, as placed upon the supports, and at the point of fracture. The formula for a round-section beam, supported at both ends and broken at the middle, is then applied to reduce the result to the standard 1.25-in. diameter. The table gives these results in their corrected form.

In place of the originally planned 465 tests with 1395 bars, there were actually made 245 tests in the crucible, with 1155 bars. This was after 36 tests had been made in the cupola, with 135 bars therefrom, all of them valuable in themselves but not serving the object of the investigation, as already stated.

TABLE 1.—*Summary of Results of Tests*

	Number of Bars	Per Cent. Wanted	Range of Bulk of Results in Per Cent.	Remarks
Phosphorus..................	94	0.10	0.10 to 0.122	Low
	103	0.40	0.40 to 0.460	Medium
	48	0.80	0.80 to 0.868	High
Total carbon.................	28	2.75	2.81 to 2.89	Low
	131	3.25	2.94 to 3.49	Medium
	84	3.75	3.54 to 3.76	High
Nickel chromium.............	60	0.00	0.05 to 0.07	None
	62	0.12	0.12 to 0.14	Low
	62	0.18	0.18	Medium
	61	0.24	0.22 to 0.25	High
Silicon.....................	31	0.75	0.76 to 0.90	
	35	1.00	1.05 to 1.20	
	36	1.25	1.30 to 1.45	
	36	1.50	1.50 to 1.65	
	33	1.75	1.78 to 1.97	
	28	2.00	2.03 to 2.13	
	33	2.25	2.25 to 2.35	
	9	2.50	2.50	
	5	2.75	2.80	

TABLE 2.—Results of Tests

Division	Test No.	Si, Per Cent.	S, Per Cent.	Mn, Per Cent.	P, Per Cent.	Ni, Per Cent.	Cr, Per Cent.	Ti, Per Cent.	T.C., Per Cent.	C.C., Per Cent.	Graph., Per Cent.	Trans., Lb.	Defl., In.	Ten., Lb. Per Sq. In.	Brinell No.
Phos., low. T.C., low. Ni-Cr, none.	60	1.05	0.087	0.61	0.096	0.07	0.00	0.14	2.847	0.645	2.202	4.675	0.11	47.350	229
	62	1.80	0.068	0.62	0.094	0.03	0.00	0.13	2.683	0.339	2.344	4.245	0.12	36.800	217
	66	2.25	0.082	0.64	0.100	0.05	0.00	0.14	2.881	0.509	2.372	4.238	0.12	37.000	217
Phos., low. T.C., low. Ni-Cr, low.	147	2.17	0.076	0.63	0.078	0.16	0.00	0.09	2.862	0.639	2.290	4.524	0.09	37.800	223
	65	2.59	0.086	0.60	0.084	0.13	0.01	0.09	2.616	0.190	2.426	4.460	0.13	39.790	210
Phos., low. T.C., low. Ni-Cr, med.	145	1.52	0.088	0.62	0.074	0.15	0.19	0.08	2.836	0.639	2.197	4.714	0.09	41.600	226
	148	1.75	0.084	0.64	0.078	0.18	0.16	0.10	2.780	0.600	2.180	4.375	0.07	41.780	229
	149	2.14	0.090	0.59	0.092	0.17	0.15	0.09	2.590	0.382	2.208	4.300	0.07	40.000	241
Phos., low. T.C., med. Ni-Cr, none.	31	0.85	0.097	0.62	0.106	0.07	0.00	0.13	3.201	0.698	2.503	4.522	0.14	36.810	215
	6	0.90	0.094	0.60	0.106	0.07	0.00	0.13	3.490	0.704	2.786	4.765	0.15	31.440	185
	30	0.91	0.098	0.61	0.100	0.07	0.00	0.12	3.310	0.780	2.530	4.190	0.15	35.220	200
	33	0.91	0.086	0.60	0.108	0.06	0.00	0.13	2.998	0.157	2.944	4.400	0.14	33.220	210
	32	1.15	0.094	0.61	0.106	0.06	0.00	0.13	3.101	0.696	2.645	4.100	0.16	34.600	197
	35	1.14	0.084	0.62	0.104	0.07	0.00	0.12	3.341	0.807	2.634	3.724	0.14	33.820	197
	38	1.28	0.085	0.64	0.106	0.06	0.01	0.14	3.508	0.728	2.780	3.722	0.14	31.200	197
	34	1.33	0.085	0.64	0.100	0.08	0.00	0.12	3.234	0.618	2.616	3.889	0.14	35.020	195
	36	1.30	0.090	0.65	0.104	0.06	0.00	0.13	3.436	0.622	2.814	3.617	0.15	35.020	195
	37	1.44	0.082	0.63	0.100	0.08	0.00	0.13	3.436	0.618	2.836	3.726	0.14	31.040	188
	64	1.47	0.089	0.62	0.092	0.03	0.15	0.13	3.071	0.699	2.372	4.157	0.13	36.410	215
Phos., low. T.C., med. Ni-Cr, med.	57	1.07	0.087	0.58	0.092	0.14	0.02	0.07	2.998	0.900	2.098	4.760	0.09	40.780	832
	58	1.61	0.003	0.56	0.090	0.12	0.03	0.07	3.054	0.920	2.134	5.050	0.10	47.760	246
	63	2.26	0.092	0.58	0.084	0.13	0.01	0.08	2.944	0.810	2.134	3.890	0.12	39.790	212
Phos., low. T.C., med. Ni-Cr, med.	142	1.00	0.078	0.57	0.090	0.22	0.15	0.07	3.026	0.889	2.137	4.520	0.05	41.600	322
	140	1.12	0.094	0.56	0.094	0.14	0.19	0.10	2.972	0.736	2.236	4.550	0.09	41.780	217
	144	1.25	0.087	0.59	0.086	0.22	0.19	0.08	2.998	0.730	2.268	4.319	0.09	39.780	229
	119	1.34	0.086	0.60	0.092	0.18	0.19	0.08	3.054	0.726	2.328	4.600	0.12	42.980	204
	120	1.91	0.075	0.67	0.110	0.15	0.16	0.08	3.436	0.710	2.726	4.022	0.12	31.830	204
	121	2.15	0.075	0.62	0.114	0.20	0.15	0.08	3.490	0.600	2.890	3.495	0.13	25.270	183
	122	2.15	0.080	0.62	0.106	0.18	0.15	0.09	3.216	0.490	2.726	4.006	0.13	33.830	207
	151	2.48	0.086	0.62	0.068	0.20	0.18	0.09	3.436	0.682	2.754	3.858	0.12	38.880	185
	150	2.63	0.082	0.63	0.054	0.20	0.18	0.08	3.208	0.454	2.454	4.605	0.11	38.000	228
	94	2.76	0.085	0.54	0.122	0.15	0.15	0.09	3.463	0.301	3.162	4.351	0.11	19.500	150
Phos., med. T.C., med. Ni-Cr, high.	197	0.64	0.075	0.57	0.100	0.17	0.38	0.08	1.948	1.378		4.127	0.05	37.800	321
	225	0.73	0.077	0.56	0.108	0.35	0.34	0.08	3.044	1.906	1.282	4.277	0.04	27.800	415
	198	0.79	0.087	0.56	0.084	0.25	0.34	0.08	1.906	1.586				35.420	854
	226	0.85	0.078	0.54	0.100	0.30	0.26	0.06	2.944	2.585	0.359	4.825	0.05	38.290	889

401	32,830	0.05	4,608	0.326	2.946	3.272	0.07	0.32	0.30	0.102	0.54	0.082	0.95	228
255	40,190	0.08	5,243	1.936	1.172	3.108	0.07	0.33	0.30	0.100	0.55	0.078	1.09	227
239	43,780	0.10	4,828	2.998	0.790	2.998	0.07	0.24	0.40	0.070	0.55	0.083	1.13	229
229	41,800	0.12	4,733	2.344	0.784	3.108	0.08	0.32	0.14	0.140	0.56	0.083	1.15	230
195	31,800	0.14	4,085	3.054	0.488	3.490	0.08	0.36	0.25	0.118	0.58	0.079	1.21	203
220	39,790	0.12	4,293	2.562	0.636	3.272	0.08	0.32	0.35	0.098	0.57	0.080	1.40	234
215	35,810	0.11	4,494	2.856	0.710	3.490	0.09	0.35	0.18	0.120	0.59	0.081	1.43	204
201	37,410	0.13	4,426	2.736	0.556	3.108	0.09	0.38	0.30	0.114	0.57	0.085	1.50	207
228	39,790	0.11	4,620	2.454	0.654	3.108	0.07	0.29	0.40	0.138	0.60	0.082	1.85	231
223	39,390	0.11	4,145	2.328	0.780	3.216	0.08	0.32	0.30	0.144	0.59	0.081	1.95	232
220	37,410	0.11	3,852	2.508	0.708	3.326	0.08	0.20	0.22	0.096	0.59	0.081	2.10	233
197	27,860	0.11	4,852	2.780	0.546	3.136	0.08	0.31	0.45	0.096	0.59	0.081	2.40	206
217	37,600	0.11	4,396	2.344	0.792	3.136	0.07	0.36	0.45	0.044	0.56	0.077	2.44	235
187	27,460	0.16	3,586	2.896	0.741	3.637	0.13	0.00	0.05	0.102	0.64	0.084	0.94	5
181	27,460	0.15	3,581	3.070	0.742	3.812	0.12	0.00	0.05	0.100	0.61	0.080	0.97	1
187	23,870	0.16	3,620	2.950	0.769	3.719	0.12	0.00	0.04	0.116	0.60	0.084	1.00	11
176	25,470	0.15	3,424	3.043	0.834	3.877	0.13	0.00	0.05	0.104	0.61	0.092	1.08	2
185	27,860	0.16	3,620	2.934	0.785	3.719	0.13	0.02	0.07	0.102	0.63	0.081	1.05	8
168	23,600	0.14	3,193	3.054	0.600	3.654	0.08	0.00	0.08	0.106	0.58	0.089	1.05	86
163	22,880	0.15	3,121	2.754	1.025	3.779	0.12	0.21	0.16	0.104	0.60	0.084	1.20	4
175	21,090	0.15	3,020	2.999	0.720	3.719	0.07	0.17	0.15	0.102	0.59	0.090	1.23	7
158	21,090	0.16	2,490	2.917	0.802	3.719	0.13	0.17	0.17	0.106	0.62	0.085	1.25	9
151	19,500	0.17	2,626	3.097	0.488	3.583	0.13	0.17	0.15	0.100	0.72	0.096	1.30	10
197	33,430	0.06	4,387	2.890	0.764	3.654	0.07	0.19	0.17	0.106	0.64	0.084	1.45	114
252	31,830	0.12	3,343	1.936	1.754	3.881	0.07	0.21	0.12	0.116	0.61	0.080	1.50	113
167	23,870	0.09	3,770	3.054	0.926	3.980	0.09	0.19	0.18	0.102	0.60	0.084	1.05	85
197	35,410	0.16	4,291	2.836	0.980	3.816	0.09	0.20	0.23	0.104	0.61	0.092	1.20	86
207	29,600	0.13	3,455	2.890	0.784	3.654	0.09	0.21	0.18	0.112	0.63	0.081	1.23	115
199	33,420	0.16	3,162	3.162	0.784	3.734	0.07	0.17	0.16	0.114	0.60	0.089	1.25	87
179	27,460	0.12	2,916	2.916	0.572	3.572	0.08	0.21	0.15	0.104	0.59	0.084	1.30	117
185	29,050	0.12	3,590	3.054	0.666	3.702	0.08	0.17	0.17	0.102	0.62	0.090	1.45	88
192	31,440	0.12	3,893	2.998	0.648	3.881	0.08	0.22	0.15	0.106	0.70	0.085	1.50	116
160	21,440	0.11	3,982	2.890	0.683	3.544	0.07	0.17	0.09	0.108	0.72	0.096	2.05	118
170	25,470	0.12	3,660	3.108	0.592	3.700	0.11	0.17	0.11	0.114	0.63	0.091	2.08	96
168	21,480	0.14	3,665	3.136	0.462	3.598	0.08	0.15	0.16	0.128	0.60	0.094	2.20	90
157	27,860	0.10	3,360	3.162	0.678	3.740	0.11	0.16	0.15	0.108	0.58	0.093	2.25	89
152	19,100	0.12	2,694	3.272	0.436	3.708	0.08	0.16	0.10	0.118	0.64	0.086		91
151	19,500	0.13	2,831	3.054	0.600	3.654	0.11	0.17	0.12	0.120	0.63	0.093		92
160	19,895	0.11	2,600	3.108	0.524	3.632	0.11		0.10	0.122	0.64	0.086		95
	19,500		2,750	2.928	0.671	3.599						0.101		93
229	33,830	0.10	4,224	2.780	0.818	3.598	0.08	0.38	0.24	0.124	0.58	0.073	0.77	181
210	35,220	0.11	4,202	2.852	0.693	3.545	0.08	0.29	0.25	0.119	0.61	0.079	0.80	208
190	28,210	0.11	3,560	2.836	0.878	3.711	0.09	0.29	0.18	0.106	0.64	0.083	0.93	169
215	37,800	0.09	4,390	2.754	0.900	3.654	0.08	0.28	0.20	0.116	0.60	0.084	1.00	201
205	32,680	0.11	4,196	3.184	0.361	3.545	0.08	0.35	0.21	0.106	0.58	0.083	1.00	199
204	33,430	0.12	4,420	2.836	0.818	3.654	0.07	0.21	0.24	0.098	0.65	0.081	1.10	200
179	26,260	0.09	2,764	3.054	0.654	3.708	0.08	0.24	0.22	0.288	0.64	0.074	1.12	100
197	31,830	0.08	3,676	2.808	0.818	3.626	0.08	0.32	0.22	0.092	0.54	0.083	1.18	205
170	21,490	0.13	3,320	3.108	0.600	3.708	0.09	0.33	0.25	0.112		0.077	1.29	172

Phos., low ......
T.C., high ......
Ni-Cr, none ......

Phos., low ......
T.C., high ......
Ni-Cr, med ......

Phos., low ......
T.C., high ......
Ni-Cr, high ......

TABLE 2.—Results of Tests (Continued)

Division	Test No.	Si, Per Cent.	S, Per Cent.	Mn, Per Cent.	P, Per Cent.	Ni, Per Cent.	Cr, Per Cent.	Ti, Per Cent.	T.C., Per Cent.	C.C., Per Cent.	Graph., Per Cent.	Trans., Lb.	Defl., In.	Ten., Lb. Per Sq. In.	Brinell No.
Phos., med.	171	1.42	0.080	0.58	0.110	0.21	0.34	0.09	3.708	0.672	3.036	3,688	0.16	25,470	176
T.C., low	175	1.44	0.084	0.61	0.120	0.21	0.27	0.08	3.576	0.740	2.836	3,000	0.11	19,500	163
Ni-Cr, none	174	1.54	0.085	0.58	0.118	0.24	0.30	0.08	3.762	0.654	3.108	2,933	0.11	17,900	163
	173	1.63	0.076	0.61	0.120	0.24	0.39	0.09	3.762	0.654	3.108	3,047	0.10	19,100	163
	176	1.71	0.083	0.58	0.118	0.25	0.42	0.08	3.680	0.572	3.108	2,700	0.09	19,100	152
	177	1.97	0.085	0.63	0.114	0.16	0.34	0.08	3.708	0.710	2.998	2,853	0.10	19,900	163
	178	2.30	0.083	0.64	0.122	0.18	0.37	0.08	3.680	0.626	3.054	2,732	0.11	19,500	166
	179	2.50	0.073	0.64	0.108	0.18	0.32	0.09	3.654	0.600	3.054			17,900	154
Phos., med.	45	1.30	0.107	0.56	0.456	0.08	0.01	0.08	2.836	0.328	2.508	4,180	0.12	39,390	215
T.C., low	46	1.62	0.105	0.59	0.490	0.08	0.00	0.07	2.944	0.328	2.616	3,765	0.12	37,600	212
Phos., med.	73	1.73	0.112	0.57	0.386	0.12	0.02	0.08	2.808	0.736	2.072	5,164	0.12	43,770	226
T.C., low	74	2.07	0.111	0.54	0.374	0.10	0.01	0.10	2.890	0.600	2.290	4,983	0.12	43,380	238
Ni-Cr, low	76	2.11	0.101	0.55	0.404	0.11	0.00	0.08	2.836	0.710	2.126	4,650	0.13	44,570	229
	77	2.34	0.104	0.56	0.408	0.09	0.02	0.07	2.780	0.572	2.208	4,860	0.13	41,600	229
	75	2.38	0.115	0.55	0.400	0.14	0.00	0.08	2.836	0.600	2.236	4,870	0.10	42,000	229
	78	2.50	0.116	0.58	0.452	0.11	0.00	0.08	2.862	0.572	2.290	4,532	0.11	39,790	229
Phos., med.	246	1.63	0.105	0.52	0.434	0.14	0.28	0.08	2.590	0.382	2.208	5,121	0.10	45,600	255
T.C., low	245	1.74	0.105	0.55	0.512	0.14	0.30	0.09	2.792	0.556	2.236	5,126	0.12	42,180	245
Ni-Cr, med	242	2.92	0.092	0.56	0.476	0.15	0.29	0.08	2.836	0.683	2.180	4,480	0.10	24,270	241
Phos., med.	153	0.88	0.110	0.52	0.458	0.28	0.19	0.08	2.901	2.193	0.708	4,959	0.05	36,810	381
T.C., low	243	2.11	0.091	0.55	0.452	0.21	0.38	0.09	2.916	0.354	2.562	4,634	0.12	41,780	232
Ni-Cr, high	41	0.76	0.108	0.63	0.456	0.06	0.00	0.12	3.289	0.781	2.508	4,536	0.13	37,800	229
Phos., med.	40	0.78	0.088	0.59	0.456	0.07	0.00	0.14	3.272	0.754	2.518	4,498	0.12	39,790	215
T.C., med.	22	1.08	0.131	0.60	0.450	0.05	0.00	0.13	3.310	0.562	2.748	3,864	0.19	35,620	202
Ni-Cr, none	12	1.11	0.105	0.62	0.420	0.04	0.00	0.12	3.468	0.861	2.607	4,290	0.15	33,400	215
	42	1.13	0.099	0.59	0.466	0.08	0.00	0.09	3.272	0.710	2.562	4,430	0.13	38,800	207
	71	1.15	0.095	0.57	0.414	0.05	0.01	0.14	3.027	0.847	2.180	4,998	0.11	37,800	235
	44	1.16	0.094	0.61	0.426	0.05	0.00	0.13	3.310	0.824	2.486	4,287	0.15	37,800	212
	72	1.25	0.098	0.58	0.398	0.07	0.01	0.15	3.011	0.831	2.180	4,898	0.11	41,780	235
	43	1.54	0.119	0.60	0.500	0.07	0.00	0.13	3.250	1.178	2.072	4,306	0.11	37,200	223
	48	1.78	0.112	0.62	0.444	0.07	0.00	0.13	3.234	0.616	2.618	4,091	0.14	34,250	210
	52	1.88	0.133	0.61	0.664	0.06	0.00	0.14	3.032	0.742	2.290	4,030	0.14	37,600	232
	18	1.96	0.118	0.58	0.480	0.06	0.00	0.15	3.272	0.327	2.945	3,222	0.15	28,000	197
	67	2.00	0.080	0.64	0.434	0.07	0.01	0.13	3.087	0.579	2.508	4,220	0.13	34,420	207
	50	2.20	0.110	0.63	0.396	0.04	0.01	0.13	3.180	0.688	2.592	4,085	0.12	35,820	217
	19	2.26	0.131	0.60	0.470	0.04	0.00	0.13	3.414	0.442	2.972	2,736	0.17	24,670	192
	20	2.40	0.121	0.58	0.460	0.04	0.00	0.13	3.408	0.436	2.972	2,780	0.16	23,870	170
Phos., med.	70	0.65	0.102	0.54	0.416	0.11	0.02	0.07	2.998	1.144	1.854	4,563	0.08	36,720	269

Class	No.														
T.C., med. / Ni-Cr, low	69	0.68	0.101	0.57	0.454	0.09	0.03	0.07	2.998	1.226	1.772	4.527	0.09	42,970	242
	128	1.73	0.096	0.62	0.450	0.10	0.09	0.09	3.380	0.818	2.562	4.278	0.12	33,820	199
	47	1.79	0.105	0.61	0.442	0.09	0.01	0.08	3.298	0.682	2.616	4.122	0.12	33,030	210
Phos., med. / T.C., med. / Ni-Cr, med.	124	0.78	0.115	0.55	0.460	0.18	0.20	0.08	3.348	0.786	2.562	4.544	0.11	37,800	229
	152	0.84	0.081	0.54	0.430	0.28	0.18	0.09	3.354	2.400	0.954	4.745	0.07	25,820	375
	125	0.93	0.116	0.58	0.440	0.18	0.18	0.08	3.326	0.818	2.508	4.634	0.10	39,790	223
	98	0.96	0.099	0.58	0.464	0.16	0.26	0.10	3.626	0.518	3.108	3.981	0.09	29,200	187
	182	1.05	0.095	0.63	0.418	0.16	0.17	0.07	3.054	0.328	2.726	3.734	0.09	29,850	192
	154	1.08	0.118	0.56	0.446	0.24	0.17	0.20	3.070	0.834	2.236	4.637	0.07	39,790	235
	127	1.10	0.094	0.62	0.414	0.18	0.16	0.10	3.380	0.708	2.672	4.521	0.12	37,800	217
	126	1.14	0.100	0.57	0.422	0.15	0.26	0.08	3.490	0.764	2.726	4.158	0.11	35,020	207
	183	1.27	0.108	0.58	0.444	0.18	0.17	0.09	3.436	0.446	2.890	3.885	0.11	26,660	187
	155	1.32	0.100	0.60	0.364	0.15	0.16	0.09	3.026	0.878	2.148	4.832	0.07	42,980	238
	136	1.32	0.120	0.59	0.578	0.18	0.17	0.08	3.490	0.036	2.454	4.050	0.09	33,830	210
	101	1.41	0.101	0.64	0.400	0.14	0.08	0.11	3.162	0.436	2.726	2.856	0.10	21,490	179
	102	1.44	0.111	0.58	0.426	0.22	0.16	0.07	3.490	0.436	2.536	2.940	0.12	23,870	179
	244	1.44	0.099	0.55	0.446	0.12	0.31	0.09	3.080	0.544	3.054	4.741	0.07	44,570	232
	156	1.57	0.104	0.58	0.414	0.12	0.16	0.09	2.970	0.680	2.290	4.634	0.09	41,390	229
	157	1.65	0.112	0.56	0.450	0.28	0.12	0.09	2.998	0.762	2.236	4.823	0.10	39,800	229
	159	1.77	0.095	0.59	0.374	0.18	0.16	0.08	2.970	0.626	2.344	4.161	0.11	33,620	202
	129	1.80	0.100	0.60	0.450	0.12	0.00	0.10	3.326	0.710	2.616	3.981	0.07	36,400	212
	218	1.84	0.090	0.64	0.382	0.38	0.15	0.09	3.342	0.720	2.236	4.783	0.09	42,800	282
	158	1.86	0.097	0.59	0.390	0.18	0.19	0.08	3.408	0.982	2.426	3.450	0.09	24,870	190
	133	2.12	0.104	0.58	0.426	0.18	0.21	0.08	2.970	0.626	2.536	4.300	0.10	38,390	229
	162	2.31	0.095	0.60	0.386	0.23	0.24	0.09	3.162	0.736	2.328	4.637	0.09	19,900	179
	160	2.32	0.103	0.63	0.410	0.20	0.21	0.09	3.054	0.682	2.808	4.777	0.10	39,195	229
	134	2.37	0.091	0.60	0.422	0.23	0.36	0.09	3.490	0.752	2.312	3.845	0.09	28,650	202
	161	2.48	0.100	0.54	0.438	0.14	0.34	0.07	3.064	0.580	2.856	3.670	0.11	27,460	382
	123	2.60	0.065	0.58	0.310	0.40	0.36	0.09	3.016	2.613	0.403		0.05	23,480	422
	237	2.76	0.096	0.62	0.442	0.13	0.36	0.07	3.189	3.119	0.070			30,640	192
Phos., med. / T.C., med. / Ni-Cr, high	236	0.80	0.105	0.56	0.452	0.23	0.36	0.09	3.436	0.492	2.944	3.707	0.09	30,640	388
	180	0.89	0.095	0.58	0.476	0.17	0.26	0.08	2.998	0.818	2.180	4.316	0.05	28,250	204
	238	0.92	0.088	0.54	0.440	0.14	0.31	0.08	3.436	0.492	2.944	4.060	0.10	35,220	291
	209	1.08	0.097	0.54	0.444	0.22	0.32	0.08	3.436	0.818	2.180	4.538	0.06	38,280	218
	239	1.12	0.092	0.57	0.460	0.38	0.27	0.08	2.998	0.492	2.616	4.534	0.10	39,390	220
	212	1.14	0.110	0.57	0.428	0.25	0.36	0.07	3.272	0.656	2.726		0.10	37,410	217
	213	1.56	0.103	0.57	0.434	0.38	0.26	0.08	3.418	0.692	2.836	4.596	0.10	35,810	217
	217	1.58	0.092	0.58	0.434	0.25	0.31	0.08	3.436	0.600	2.868	4.485	0.14	35,810	
	214	1.66	0.098	0.58	0.864	0.22	0.32	0.10	3.436	0.568	2.316	4.499	0.09	39,390	
	248	1.73	0.108	0.55	0.444	0.38	0.27	0.09	3.054	0.574	2.476	4.885	0.08	41,380	235
	240	1.79	0.102	0.54	0.480	0.25	0.32	0.07	3.272	0.578	2.480	4.622	0.09	36,210	229
	216	1.95	0.087	0.56	0.432	0.22	0.38	0.08	2.998	0.792	2.276	4.461	0.10	37,600	241
	241	1.96	0.088	0.55	0.444	0.38	0.27	0.08	3.272	0.622	2.998	3.038	0.08	25,470	192
	191	2.00		0.60	0.452	0.15	0.32	0.09	3.490	0.492					
Phos., med. / T.C., high / Ni-Cr, none	13	1.00	0.123	0.66	0.492	0.04	0.00	0.14	3.599	1.096	2.503	3.715	0.17	30,040	204
	14	1.10	0.125	0.63	0.492	0.04	0.00	0.13	3.550	0.813	2.737	3.748	0.17	29,840	197
	15	1.38	0.131	0.60	0.478	0.06	0.00	0.13	3.648	0.867	2.781	2.841	0.15	25,470	187
	16	1.41	0.117	0.64	0.476	0.06	0.00	0.13	3.501	0.802	2.699	3.625	0.16	31,200	204
	17	1.48	0.126	0.59	0.500	0.06	0.00	0.13	3.545	0.900	2.645	3.500	0.16	28,000	197
	21	2.80	0.123	0.58	0.478	0.04	0.00	0.13	3.626	0.750	2.876	2.754	0.15	23,870	170

TABLE 2.—Results of Tests (Continued)

Division	Test No.	Si, Per Cent.	S, Per Cent.	Mn, Per Cent.	P, Per Cent.	Ni, Per Cent.	Cr, Per Cent.	Ti, Per Cent.	T.C., Per Cent.	C.C., Per Cent.	Graph., Per Cent.	Trans., Lb.	Defl., In.	Ten., Lb. Per Sq. In.	Brinell
Phos., med. / T.C., high. / Ni-Cr, low.	210	0.78	0.097	0.60	0.500	0.18	0.02	0.10	3.545	1.229	2.316	4,420	0.08	33,030	244
	132	2.25	0.099	0.63	0.442	0.14	0.01	0.09	3.890	0.946	2.944	4,094	0.10	29,450	197
Phos., med. / T.C., high. / Ni-Cr, med.	99	1.20	0.105	0.57	0.426	0.18	0.18	0.07	3.598	0.606	2.998	3,463	0.12	25,870	179
	184	1.34	0.100	0.59	0.432	0.11	0.33	0.08	3.598	0.436	3.162	3,443	0.09	24,180	187
	101	(a)1.43	0.132	0.56	0.434	0.14	0.16	0.09	3.741	0.743	2.998	2,860	0.09	22,860	164
	97	(d)1.45	0.121	0.54	0.480	0.25	0.15	0.10	3.545	0.655	2.890	3,665	0.12	23,600	190
	101	1.54	0.103	0.63	0.426	0.18	0.17	0.12	3.708	0.512	3.196	3,280	0.09	22,680	168
	130	(b)1.62	0.106	0.60	0.446	0.18	0.18	0.09	3.870	0.806	3.064	2,630	0.11	23,280	181
	131	1.70	0.099	0.65	0.442	0.18	0.21	0.10	3.572	0.710	2.862	3,255	0.10	21,880	187
	101	(a)1.79	0.094	0.64	0.440	0.18	0.24	0.09	3.654	1.038	2.616	3,034	0.09	19,900	179
	105	1.95	0.120	0.55	0.452	0.15	0.20	0.09	3.854	0.600	3.054	3,000	0.11	21,890	170
	106	1.96	0.115	0.67	0.460	0.09	0.17	0.07	3.544	0.381	3.163	3,207	0.08	23,870	164
	101	(c)1.98	0.102	0.62	0.410	0.18	0.16	0.13	3.616	0.562	3.054	2,927	0.10	19,900	178
	103	2.02	0.108	0.56	0.456	0.09	0.20	0.08	3.676	0.678	2.998	3,020	0.08	25,870	170
	188	2.08	0.117	0.65	0.520	0.12	0.16	0.13	3.599	0.709	2.890	3,352	0.11	19,900	174
	105	2.13	0.104	0.58	0.568	0.10	0.31	0.09	3.598	0.572	3.026	3,089	0.10	25,870	179
Phos., med. / T.C., high. / Ni-Cr, high.	104	1.04	0.125	0.60	0.470	0.30	0.20	0.14	3.544	0.464	3.080	3,215	0.05	37,800	183
	215	1.58	0.095	0.59	0.434	0.28	0.23	0.09	3.598	0.888	2.710	3,990	0.07	29,600	217
	186	1.60	0.102	0.61	0.414	0.17	0.34	0.08	3.762	0.948	3.190	3,154	0.08	21,885	210
	185	1.64	0.103	0.62	0.402	0.20	0.35	0.08	3.708	0.572	3.216	3,190	0.11	21,090	179
	187	1.94	0.099	0.62	0.420	0.18	0.28	0.08	3.654	0.492	3.190	3,396	0.11	21,490	185
	189	2.33	0.095	0.60	0.448	0.18	0.30	0.10	3.816	0.468	3.348	3,330	0.11	22,880	179
Phos., high. / T.C., low. / Ni-Cr, low.	90	1.25	0.144	0.54	0.834	0.15	0.00	0.07	2.836	0.820	2.016	4,082	0.08	39,790	244
	79	1.30	0.129	0.56	0.798	0.11	0.00	0.07	2.808	0.736	2.072	4,040	0.06	43,370	238
	84	1.80	0.123	0.58	0.816	0.13	0.00	0.08	2.836	0.546	2.290	4,163	0.08	39,390	241
	164	1.94	0.116	0.52	0.810	0.21	0.00	0.08	2.863	0.715	2.148	4,157	0.10	37,800	248
	82	2.09	0.126	0.56	0.784	0.15	0.00	0.07	2.726	0.546	2.180	4,215	0.09	39,390	235
	81	2.30	0.120	0.62	0.800	0.09	0.00	0.13	2.836	0.520	2.316	4,028	0.07	37,600	251
	83	2.50	0.129	0.54	0.824	0.09	0.01	0.12	2.846	0.392	2.454	4,398	0.07	39,380	245
	167	2.65	0.130	0.53	0.808	0.20	0.00	0.08	2.726	0.090	2.236	4,699	0.09	39,800	241
Phos., high. / T.C., low. / Ni-Cr, med.	163	1.79	0.127	0.53	0.688	0.14	0.17	0.09	2.840	0.768	2.072	4,587	0.07	45,560	255
	249	1.86	0.121	0.56	0.890	0.14	0.28	0.08	2.890	0.688	2.202	4,105	0.07	41,380	255
	168	2.66	0.135	0.58	0.850	0.21	0.14	0.09	2.672	0.382	2.290			31,440	248
Phos., high. / T.C., med. / Ni-Cr, none.	51	1.78	0.135	0.57	0.778	0.05	0.01	0.13	3.060	0.606	2.454	3,980	0.10	39,790	235
	23	1.86	0.151	0.60	0.810	0.03	0.00	0.14	3.437	0.492	2.945	2,880	0.11	25,070	199
	27	2.13	0.120	0.62	0.724	0.05	0.00	0.13	3.424	0.652	2.772	2,745	0.09	22,000	197
	53	2.13	0.134	0.58	0.776	0.03	0.00	0.13	3.081	0.545	2.536	3,943	0.11	33,820	241
	55	2.28	0.135	0.56	0.764	0.08	0.01	0.07	3.108	0.600	2.508	3,794	0.09	31,830	217

Label	No.														
Phos., high	24	2.29	0.150	0.59	0.814	0.03	0.00	0.14	3.365	0.453	2.912	2,760	0.13	24,870	197
	25	2.33	0.160	0.62	0.830	0.03	0.00	0.13	3.430	0.562	2.868	2,917	0.18	26,860	195
T.C., med.	54	2.43	0.147	0.57	0.798	0.12	0.00	0.08	2.998	0.490	2.508	3,852	0.09	33,820	217
	26	2.58	0.162	0.59	0.840	0.05	0.00	0.13	3.316	0.388	2.928	2,766	0.13	25,470	179
Ni-Cr, med.	56	2.78	0.132	0.59	0.816	0.09	0.00	0.08	3.108	0.546	2.562	3,225	0.10	27,460	207

Label	No.														
Phos., high	111	2.16	0.130	0.60	0.868	0.15	0.19	0.08	3.436	0.600	2.836	3,264	0.08	24,460	197
	138	2.27	0.131	0.58	0.854	0.15	0.18	0.09	3.272	0.584	2.688	3,699	0.07	28,250	217
	251	2.33	0.121	0.53	0.848	0.12	0.28	0.08	2.944	0.654	2.290	4,019	0.09	37,800	258
	139	2.35	0.111	0.57	0.880	0.10	0.18	0.08	2.272	0.736	2.536	3,642	0.09	31,040	217
T.C., med.	165	2.35	0.140	0.55	0.856	0.23	0.14	0.08	2.912	0.596	2.316	3,969	0.08	37,800	235
	219	1.45	0.128	0.55	0.864	0.25	0.27	0.07	3.326	0.790	2.536	4,176	0.06	37,800	241
	221	1.45	0.120	0.52	0.800	0.50	0.00	0.10	3.250	0.306	2.944	4,361	0.07	35,810	241
	247	1.53	0.121	0.56	0.992	0.19	0.35	0.08	3.036	0.528	2.508	4,213	0.09	36,410	229
	220	1.54	0.115	0.61	0.868	0.20	0.24	0.09	3.326	0.818	2.508	4,191	0.07	31,830	217
	135	1.60	0.125	0.57	0.828	0.28	0.25	0.08	3.436	0.710	2.726	3,785	0.09	27,060	199
	192	1.65	0.125	0.58	0.886	0.28	0.27	0.09	3.436	0.536	2.900	3,180	0.08	25,070	229
	223	1.69	0.102	0.55	0.720	0.33	0.23	0.09	3.272	0.874	2.398	4,400	0.10	37,200	232
	222	1.88	0.113	0.69	0.834	0.23	0.14	0.08	3.162	0.626	2.536	4,363	0.07	33,430	235
	166	2.01	0.131	0.55	0.854	0.14	0.89	0.08	2.944	0.708	2.236	3,866	0.10	33,030	255
NiCr, high	250	2.09	0.106	0.56	0.868	0.14	0.33	0.08	3.054	0.703	2.351	4,014	0.11	35,420	255
	252	2.28	0.120	0.56	0.860	0.14	0.30	0.08	2.974	0.902	2.072	4,336	0.10	38,280	255
	224	2.41	0.117	0.61	0.852	0.20	0.30	0.10	3.162	0.584	2.578	4,018	0.08	31,830	223

Label	No.														
Phos., high	190	1.42	0.107	0.56	0.876	0.12	0.28	0.09	3.680	0.556	3.124	3,126	0.10	21,890	185
	107	1.58	0.124	0.55	0.800	0.12	0.13	0.09	3.708	0.736	2.972	3,135	0.11	25,070	195
T.C., high	108	1.79	0.121	0.60	0.808	0.15	0.17	0.07	3.544	0.690	2.754	3,340	0.08	26,260	190
	110	1.87	0.128	0.58	0.800	0.15	0.15	0.07	3.572	0.682	2.890	3,040	0.08	23,780	185
Ni-Cr, med.	109	1.90	0.118	0.60	0.814	0.21	0.17	0.07	3.544	0.192	3.352	3,261	0.09	28,480	195
	112	2.11	0.121	0.62	0.760	0.12	0.16	0.08	3.598	1.020	2.578	3,148	0.09	25,070	192

Label	No.														
Phos., high	193	1.55	0.133	0.57	0.804	0.18	0.28	0.08	3.674	0.784	2.890	3,088	0.06	25,070	197
T.C., high	194	1.93	0.118	0.58	0.802	0.18	0.30	0.09	3.762	0.926	2.836	3,235	0.07	23,480	192
Ni-Cr, high	196	2.28	0.121	0.55	0.860	0.22	0.36	0.08	3.545	0.601	2.944	3,364	0.09	23,480	187
	195	2.28	0.113	0.61	0.860	0.25	0.32	0.10	3.654	0.650	3.004	3,063	0.09	25,270	197

TABLE 3.—Results of Compression Tests

Test No.	Compression, Lb. per Sq. In.	Tension, Lb. per Sq. In.	Transverse, Lb.	Brinell No.	Si, Per Cent.	S, Per Cent.	Mn, Per Cent.	P, Per Cent.	Ni, Per Cent.	Cr, Per Cent.	Ti, Per Cent.	T.C., Per Cent.	C.C., Per Cent.	Graph., Per Cent.
26	99,200	25,470	3,240	179	2.58	0.162	0.59	0.840	0.05	0.00	0.13	3.316	0.388	2.928
171	108,000	25,470	4,180	177	1.42	0.080	0.58	0.110	0.21	0.34	0.09	3.708	0.672	3.036
116	117,300	29,050	4,494	185	1.25	0.085	0.62	0.106	0.17	0.22	0.08	3.681	0.683	2.998
115	121,000	35,410	5,026	204	1.05	0.081	0.63	0.112	0.18	0.21	0.09	3.654	0.764	2.890
139	126,200	31,040	3,940	217	2.35	0.111	0.57	0.880	0.10	0.16	0.08	3.272	0.736	2.536
67	128,400	34,420	4,842	207	2.00	0.080	0.64	0.434	0.06	0.01	0.13	3.087	0.579	2.508

Except for a rather wide range in the case of medium total carbon, the compositions of the bulk of the results obtained run within reasonably serviceable limits, indicating that the physical characteristics in this series of tests can be safely judged in connection therewith. It is well known that in ordinary foundry practice it is not possible to predict strength from composition with sufficient exactness to allow specifying both in purchasing castings. The relative proportions of pig to remelted material, the conduct of the melting process, pouring temperature, and other things so affect the physical strength of the metal for a given analysis that the limits of chemical composition have to be made too wide to be of real use. It is quite a different situation with regard to materials of a homogeneous character, such as steel, where chemical analysis and physical strength go hand in hand, when heat treatment is constant. Hence, in the case of cast iron there must be a liberal allowance of physical strength as based on composition. This allowance must be the more liberal as the process of melting becomes less amenable to scientific control. Thus, the electric furnace, crucible, open-hearth furnace, air furnace and cupola would be the downward order of excellence in question.

It should not be inferred, however, that the results of these tests, as made in the crucible, are not capable of being duplicated in the cupola in sufficient measure to be perfectly reliable in daily melting practice. The fact that it was impossible to run five different mixtures through the cupola, one right after the other, because some of the upper charges melted along the cupola lining and ran into the lower ones, simply means that it is not wise to oper-

ate that way when a clear-cut separation of mixtures in molten iron is wanted. By running the same mixture for a considerable portion of the heat, however, this melting action will only affect the composition of the first tap or two, and after that the molten metal will be uniform in composition. The results from these 245 mixtures, which were melted in crucibles for expediency, may be safely applied to regular cupola melting with the probability of getting merely slightly lower strengths as the result of the oxidation inseparable from the latter method.

Results Shown by Transverse-strength Curves

1. The general tendency of almost all the curves for all percentages of total carbon is downward as the silicon increases. This is natural, as silicon promotes the formation of graphite and means softer weaker metal. There are some exceptional cases, however, where the curves rise at first but go downwards later; two of the curves go upwards throughout their length. It must be remembered that the lower silicon ranges make for white and mottled irons, which may not be compared with the gray varieties; also that a high phosphorus content has its effects.

2. In general, the iron is weaker as the total carbon increases. This accords perfectly with practice, for the more steel added (with rational melting) the lower the total carbon and the stronger the metal, and vice versa.

3. In general, the lower the phosphorus, the stronger the iron. This is, however, considerably affected by the percentages of the other elements present—particularly the total carbon and the silicon—for occasionally the medium-phosphorus curve ranges above the low phosphorus.

4. The curves show the irons with nickel-chromium to be stronger on transverse test than those where these elements are absent, thus attesting to the beneficial results of Mayari pig iron additions to the mixtures.

5. The low and medium nickel-chromium curves show stronger iron than where high ranges of these elements exist. This would indicate that the peculiarity of chromium in furthering the retention of carbon in the combined state yields irons comparable to the lower silicon ranges. As excellent practical results have been obtained when the percentage of Mayari pig iron used in mixtures of high silicon ran considerably beyond those of this series of tests, this point will be studied later in connection with some electric-furnace tests to be made supplemental to this series.

6. The best transverse strengths seem to have been obtained with the low to medium total carbon ranges, with 1.50 to 2.00 per cent. silicon, and low to medium phosphorus and nickel-chromium.

7. The least transverse strengths were obtained with a high total carbon, silicon above 2.00 per cent., and with nickel-chromium either zero or high. The phosphorus in these cases seems to have been negligible.

Results Shown by Deflection Curves

1. The general tendency of the deflection curves for all ranges of total carbon is to go upward as the silicon content rises. This is naturally the reverse of the transverse strength showing, and agrees with general foundry practice in that a soft iron bends more than a hard one. Some exceptions will be noted among the curves, but these might have looked differently if possible of extension each way through additional melting tests. Furthermore, the scale of the curves is rather large so that a few hundredths of an inch deflection up or down greatly affect the direction of the curves.

2. The deflection of the test pieces (when at the breaking point) is greater as the total carbon goes up, or again the reverse of the transverse results. The greatest deflections were observed in irons of high total carbon, no nickel-chromium and low phosphorus; i.e., in the higher silicon ranges, or when everything tended to make a strong but soft casting.

3. The deflection drops as the phosphorus content increases showing that the lower phosphorus ranges give better deflections, or more resilient iron than the high ones. This is in accord with recent developments in making piston rings, where the high-phosphorus and lower silicon iron is being replaced by low-phosphorus higher silicon iron.

4. The nickel-chromium content seems to run with the phosphorus in this series of deflection curves, for the greater deflections are with no, or but little nickel-chromium. The deflection goes down as the nickel-chromium goes up, seemingly paralleling the low-silicon effects noted in the transverse-strength curves, and plainly shown in the deflection results for all the carbon ranges.

5. The greatest deflections seem to have been obtained when the total carbon was medium, the silicon above 2 per cent., the phosphorus low, and the nickel-chromium either absent or low.

6. The least deflections are found in irons of low total carbon and silicon, with high phosphorus and nickel-chromium.

Results Shown by Tensile-strength Curves

1. As in the transverse tests, the trend of the curves for all the total carbon ranges is generally downward as the silicon increases. The few exceptions might have looked differently if possible of extension both ways.

2. As in the transverse tests, the tensile-strength ranges go downward as the total carbon goes up.

3. In general, with the low to medium total carbons, the low nickel-chromium with the low to medium phosphorus ranges give better tensile strengths than high phosphorus with either zero or high nickel-chromium.

In the case of the high total carbons, a high nickel-chromium and phosphorus content seems to have made the bars stiffer, so that while they did not bend very far, they were fairly strong on direct tension.

Results Shown by Brinell Test Curves

The harder the metal the higher is the Brinell number. In studying the individual tests in this large series, it was thought that a classification could be arranged based on the graphite-combined carbon relation. Thus, with an apparently very soft bar of high graphite content and with very low combined carbon (each the average of the entire section) a very low Brinell reading should have been obtained. As the proportion of graphite to combined carbon would go down, the readings should go up— the metal becoming stiffer, so that with a minimum of graphite and maximum of combined carbon, the hardest bar should have been obtained. This situation did not, however, eventuate, and all kinds of combinations of the elements in cast iron with the carbon divisions were tried. Finally, it was discovered that up to a certain point the Brinell hardness numbers ran directly with a decreasing total carbon content, no matter what the relation of graphite and combined carbon and the silicon content might be. The figures run very steady for the gray cast irons, but when the combined carbons run over 0.80 per cent., and the harder irons eventually become mottled and white appears, the results—though still running in the right direction—become more erratic and extreme.

CONCLUSIONS

An examination of the curves, arranged according to a regular progression of total carbon percentages, and grouped according to the phosphorus ranges with the silicon content ignored entirely, shows the following:

1. All the curves show that the higher the total carbon, the lower the Brinell numbers, or softer the metal; and vice versa.

2. That whether phosphorus is low, medium, or high, the relative positions of the curves are the same; that is, the element in question evidently has but little effect on the hardness of cast iron as compared with that of the total carbon content.

3. The nickel-chromium definitely increases hardness with rising percentages. The increase is not large, but it shows that the softer irons may be had with higher total carbon percentages, which in turn can be kept strong enough by holding the graphite down with the low silicon ranges.

4. There would appear a general showing that the conception of hardness in cast iron and strength must be kept distinctly apart, and that further investigations should be made to get more information on

this subject. The foundryman occasionally turning out gray-iron castings with chilled surfaces is not interested in strength as much as the maker of chilled rolls and car wheels, who wants reasonably soft, strong, interior metal with extremely hard chilled portions at the surface. He will be highly interested in developments to attain these physical characteristics in best measure.

The study of the photomicrographs taken from one bar in each of the sets of the test series has been left for future consideration, and any developments will be brought out in connection with a series of electric-furnace tests to be made on the higher nickel-chromium ranges for cast-iron products.

Oil Reserves of the United States

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THE submission of carefully prepared estimates of the oil reserves of the United States calls for no apology or explanation. In this country, petroleum is a rapidly wasting asset and an occasional appraisal of the amount remaining in the ground is a simple business procedure to safeguard the general welfare and the prosperity of the republic. The rumored danger of failure, in the near future, of the foreign supply on which we depend in constantly increasing amounts makes the immediate review of our available reserves the more imperative.

The Director of the U. S. Geological Survey, in March, 1921, requested the American Association of Petroleum Geologists to coöperate in the preparation of new estimates of the oil remaining in the ground. With the members of this coöperative committee, state geologists, regional geologists, consulting specialists, and geologists and engineers attached to many companies worked both independently and through subcommittees. The reports for districts, counties, fields, etc., after review by local or state committees, were discussed by the joint coöperative committee in conferences that, in some instances, were attended by the original compilers of estimates for the states or smaller areas. Finally the data were reviewed and the figures revised by the joint coöperative committee, so that the totals given in Table 1 represent the opinion of the committee, which was composed of F. W. DeWolf, W. E. Wrather, Roswell H. Johnson, Wallace A. Pratt, Alexander W. McCoy, Carl H. Beal, C. T. Lupton, G. C. Matson, K. C. Heald, W. T. Thom, Jr., A. E. Fath, Kirtley F. Mather, R. C. Moore, and David White, chairman.

Oil geologists and engineers understand well the speculative nature and the chances of error that must attend estimates of the petroleum in the ground when they cover not merely prospective territory, but possible territory. Neither the character nor the magnitude of the factors of uncertainty is to be disguised. Further, the inevitable differences of opinion between geologists of equal rank and experience must also be

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taken into account. Against these must be placed the personnel of the committee and the advantage of full discussion and conferences attended, in most cases, by geologists especially familiar with the regions in question; and the fact that the figures published represent the combined individual judgments of the members of the committee. The estimates given are undoubtedly based on more complete data compiled by better methods than those previously prepared for this country. No such appraisal has been prepared for any other country; for most countries, the development and the records of tests, exploration and production are not so long established nor so adequate for such calculations. These estimates, the preparation of which extended through the greater part of a year, may be regarded as the best now obtainable.

TABLE 1.—*Estimated Oil Reserves of the United States, by States or Regions, January 1, 1922, Millions of Barrels of 42 Gal.*

New York	100
Pennsylvania	260
West Virginia	200
Ohio	190
Indiana and Michigan	70
Illinois	440
Kentucky, Tennessee, northern Alabama, and northeastern Mississippi	175
Missouri, Iowa, North Dakota, Wisconsin, and Minnesota	40
Kansas	425
Oklahoma	1,340
Northern Louisiana and Arkansas	525
Texas, except Gulf coast	670
Gulf coast, Texas, and Louisiana	2,100
Colorado, New Mexico, and Arizona	50
Wyoming	525
Montana, Nebraska, and South Dakota	100
Utah, Nevada, Oregon, Washington, and Idaho	80
California	1,850
Eastern Gulf coastal plain and Atlantic coast states	10
	9,150

Of these 9,150,000,000 bbl. of oil over 4,000,000,000 bbl. are in the heavy-oil group, including fuel oils, and 5,000,000,000 bbl. in the light-oil classes. The Lima-Indiana field is credited with 40,000,000 bbl., and approximately 725,000,000 bbl. are believed to remain in the Appalachian states. These estimates relate to oil in the ground recoverable by present methods. It was the unanimous conclusion of the committee that, except for small areas in western New York and northern Pennsylvania where "water drive" has long been in operation, any attempt to forecast quantitatively the increased recovery, perhaps 40–80 per cent., to be

won in the different regions, fields, and pools by improved methods of recovery would introduce an element of uncertainty far greater than that entering into the present estimates.

The total exceeds the estimate last published by the writer by 2,000,-000,000 bbl. The anomaly of growing estimates simultaneous with exhausting resources is explained by the widespread wildcatting during and since the war; by the vastly greater volume of information touching all areas available for use by a larger and more diverse body of oil geologists in the committee; by the more thorough methods of calculating oil reserves and the experience developed in their use; and by the fact that the earlier estimates were far more conservative.

It is more than probable that, in most instances, the actual yields of the producing states will exceed the estimates, but it is equally probable that for the older producing states the excess will not be large compared to the whole amount. Greater uncertainty attends, of course, the states of present small production and less complete exploration and development.

It is not regarded advisable by the committee to make public the details of allotments for non-producing regions. Such publication would inevitably lead to abominable stock promotion as well as unnecessary losses by misguided but honest individuals and communities. It may, however, be of interest to note certain views of the committee:

1. The recovery of oil in commercial amounts cannot be expected in areas where the rocks are metamorphic, igneous, or too far altered, as indicated by the stage of carbonization of the organic debris in the strata.

2. Fresh-water deposits, such as some of the Tertiary formations of the Great Basin and the Northwest, are not wholly without oil possibilities, though the attitude of the oil geologists in regard to such deposits is far from optimistic.

3. The Triassic of the Atlantic states, on account of its generally strongly folded and sometimes shattered condition, its general composition, and the data available as to its stage of alteration, is given little value even in its deeper basins. For this reason as well as their widespread regional alteration, the New England states are practically excluded from the possible oil territory of the country, though it is conceded that conditions possibly favorable to oil may eventually be found in the Upper Paleozoic of a portion of northern Maine.

4. The absence, so far as at present known, of buckling of the Cretaceous and Tertiary formations of the Atlantic coastal plain into well-developed anticlines and domes, and the relative thinness of the aggregate post-Triassic deposits in most of this area offer little hope for appreciable amounts of oil in this great region.

·5. The relative alteration of the Paleozoic beds of the Upper Peninsula of Michigan and of Minnesota, the restrictions as to thickness of the

post-Paleozoic formations, and the apparent lack of folding in the latter exclude these regions from the possible oil territory of the country. On the other hand, the committee has considered and, in most cases, made allowances for possible occurrences of oil in older formations, including deeper sands than those now productive.

No estimate is submitted for Alaska, as to which recent geologic reports are in general very favorable.

It may be noted that the oil reserves of the United States, as estimated by the committee, comprise one-seventh of the oil reserves of the world, as estimated by Stebinger and White. From these figures, it would appear that this country, with an original content of about 14,000,000,-000 bbl., was much fuller of oil than other parts of the earth or, which is more probable, the estimates for some of the other countries are too low.

Contrary to the statement that the oil reserves of the United States will be exhausted in 20 years, a long time will be required for the exhaustion of our oil fields. Though the estimated reserves would not survive 20 years if they could be made available so quickly as to meet the demands, even at the present rate of consumption, some of the oil pools are not likely to be discovered within a generation and some will be producing 75 years from now. Therefore, if the oil cannot be so rapidly found and extracted from the ground, we must the sooner pass a point beyond which will follow a long, though naturally variable, decline.

As our domestic production has been increasing annually a little more than 6 per cent. during normal years, with occasional special jumps, a much larger increase in the amount of imported oil has been necessary to fill the gap between domestic production and domestic requirements. While the output of the fields in the United States in 1913 was 248,000,-000 bbl., an 11 per cent. increase over that of the preceding year; in 1918, 356,000,000 bbl., 6 per cent. increase; 1919, 378,000,000 bbl., 6 per cent. increase; 1920, 443,000,000 bbl., 17 per cent. increase; and 1921, 470,000,000 bbl., 6.1 per cent. increase; our importations of petroleum, mainly from Mexico, for the same years were, respectively, 18,000,000 bbl., 141 per cent. increase; 38,000,000 bbl., 25 per cent. increase; 53,000,000 bbl., 40 per cent. increase; 106,000,000 bbl., 101 per cent. increase; and 125,000,000 bbl., 18.2 per cent. increase. The situation is as alarming as it is striking.

We must have foreign oil. On the other hand, the importation of foreign oil may reach a point that, through oversupply, is harmful to the oil industry of one part of the United States, however profitable it may be to another. The effects of oversupply on our domestic production last year cannot be accurately appraised, for one cannot readily calculate the quantity nor the market prices of oil that would have been produced

in the United States in 1921 had there been a deficiency of oil for importation. Such a deficiency, it is reasonable to believe, will be experienced when Mexican production is forced to shift from a comparatively small number (say 300) of stupendously spectacular gushers to wells more nearly of the normal type. An idea of the magnitude of the task of developing 175,000,000 bbl. of annual production in Mexico from wells of the ordinary types and the years required may be drawn from the fact that in 1908, when the production of the United States first reached that amount, over 140,000 wells contributed the oil. Geologists may forecast the geologic possibilities of future Mexican production from salt domes, from anticlinal sands, or from cavernous limestones, but only the driller and the oil-field engineer can calculate approximately the great deductions to be made in this number of wells on account of necessary rapidity of development, freshness of production, etc. No one can forecast the efficiency of the oil companies and the measure of their success in insuring the oil-using industries against an hiatus in oil production in Mexico. The peak of production in the United States is likely to be passed during the early part of the period of overtaxing strain on the American oil fields consequent to the pronounced deficiency of imported oil, which in turn may be expected to result from the slump that must some day come in Mexican production. If sufficient time is given for the development in other regions of oil available to our people before Tamosopo gusher supplies are too far exhausted, and if this time is diligently utilized by American oil companies, the strain in the domestic oil market will be less acute and less disastrous.

Meanwhile a method should be devised by which through the purchase by some branch of the Government, such as the Navy, of oil, in relatively small quantities but in the most effective regions, in periods of oversupply like that of last year, oil prices in the United States may be stabilized and the oil producer saved from bankruptcy, while the Government will profit through the purchase for storage of oil in periods of low prices. In considering this question, which is one of sound economics and of conservation as well as of good business principles and management, the fact that the United States is dependent on foreign sources for petroleum with which to carry on its industrial and social programs must not be lost sight of, even momentarily. The dependence of the United States on foreign oil is a reality, and as time goes on the extent to which this dependence may become more fully absolute depends not on the discovery and substitution of some vegetable product, such as alcohol, but on the extent to which the production of liquid hydrocarbons as byproducts of the better use of our coals, and the successful distillation of American oil shales, can be made to fill the void. The prevention of waste of petroleum and its products and, especially, the development and application of methods for the larger recovery of the oil in the ground will go

far in deferring our period of greatest dependence on foreign oil, while contributing to our national efficiency and prosperity.   Finally, however, it would appear that not even the acquisition of foreign oil supplies which, combined with better extraction and better use of our own oils, may for many years postpone the evil day, can prevent the ultimate realization of the need for artificial petroleum or its substitutes.

Future Demands on Oil Industry of United States*

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IN 1920, 531 million barrels of crude petroleum were consumed in the United States. As imposing as this figure is, the fact that the domestic consumption of crude petroleum has increased at an average rate of 10 per cent. a year for the past decade is more striking. Will this rate of increase in demand continue? If so, the year 1930 will call for 1260 million barrels of petroleum—a staggering volume. On the other hand, may we expect the demand for petroleum to grow as rapidly as it did when the automotive industry was having its most phenomenal expansion? If not, to what extent will the growth of oil requirements decrease? When and to what degree will the petroleum industry become affected by the economic law of diminishing returns? Obviously, this matter requires careful analysis, especially as both national and industrial planning must be predicated upon present estimates of the requirements of the future. The purpose of this paper, accordingly, is to project, as closely as present limitations of knowledge permit, the course of the demand for petroleum in the United States over the next ten years.

At first, the problem of estimating the future of a demand as complex as that for petroleum might appear insoluble. Fortunately, however, there are several scientific methods of approaching the problem, centering around a study of rates of industrial growth, which lend considerable hope that the requirements of the immediate future in this field may be approximated with sufficient closeness to have practical value as a basis of action.

GROWTH OF PETROLEUM DEMAND

The demand for crude petroleum is the resultant of the demands for the products made jointly from this substance—gasoline, kerosene, gas and fuel oil, and lubricating oils. The demands for these have grown at

different rates, as shown in Fig. 1, which depicts on a semilogarithmic, or
ratio, scale the domestic consumption of the four major products for all
years for which there is a statistical record.[1]   On this chart, the slopes of
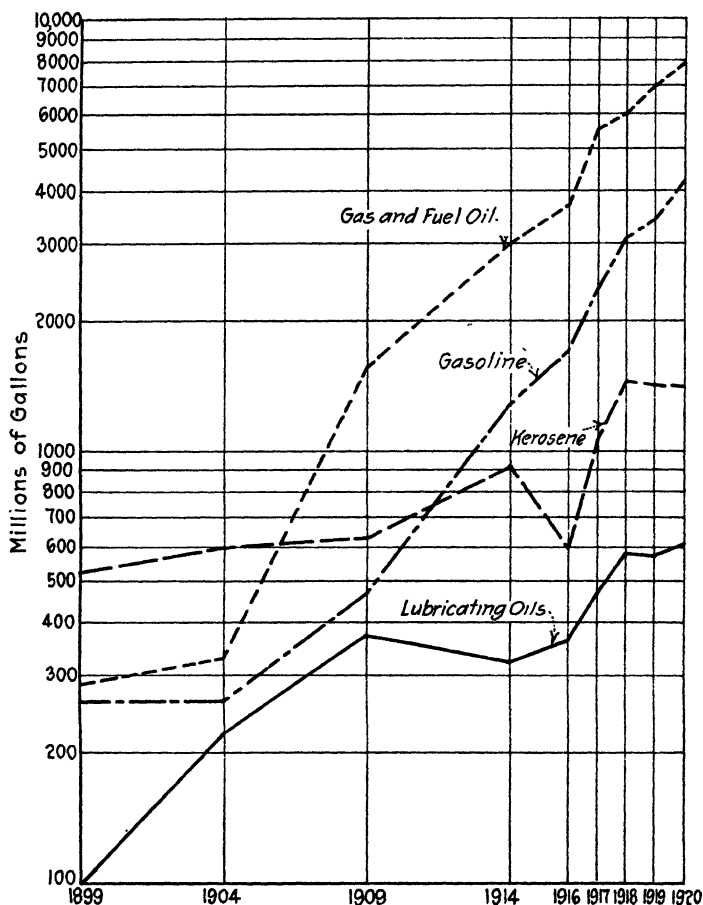
FIG. 1.—RELATIVE RATES OF GROWTH OF DOMESTIC CONSUMPTION OF PRINCIPAL
PETROLEUM PRODUCTS 1899-1920, SHOWING DOMINANT GROWTH OF DEMAND FOR
GASOLINE.

the curves are proportional to the rates of growth, and the demand for
crude petroleum is thus analyzed into its component parts.   The demand
for gasoline, of recent years, has grown more rapidly than that for fuel oil
and much more rapidly than the demands for kerosene and lubricating

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[1] Domestic consumption is taken as the index of demand, exports being eliminated
from consideration in the present problem as not being a fundamental demand upon
the American petroleum industry.

oils.[2] Gasoline has become the leading product, setting the pace for the quantity of crude petroleum needed.

The data on which Fig. 1 is based are presented in Table 1. In Table 2 is shown the relative consumption of the major petroleum prod-
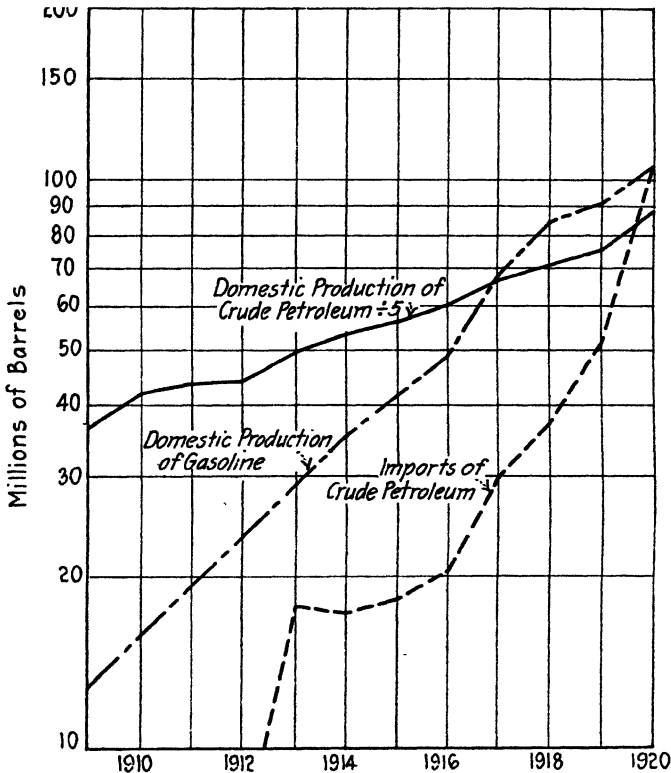
FIG. 2.—RELATIVE RATES OF GROWTH OF SUPPLY OF CRUDE PETROLEUM AND PRODUCTION OF GASOLINE 1909-1920, SHOWING DEGREE TO WHICH OUTPUT OF GASOLINE IS OUTDISTANCING DOMESTIC PRODUCTION OF CRUDE PETROLEUM.

ucts for the period 1914–1920, in percentages of the figures in 1914. Here, again, the extent to which the consumption of gasoline has forged ahead of the other products is unmistakably shown. Taking 1914 as a base, this product shows a total advance of 230 per cent., as compared

[2] The disparity in rates of growth is clearly apparent in Fig. 1, with the possible exception of the comparison between gasoline and fuel oil. By fitting a straight line by the method of least squares to the period 1914–1920, on each of these curves, it may be determined that the annual rate of increase in consumption during this time averaged 23.07 per cent. for gasoline and 18.41 per cent. for fuel oil   The equations representing the normal trend of consumption of gasoline and fuel oil for 1914–1920 are $y = 2328 \times 1.2307^x$ (origin at 1917) and $y = 4906 \times 1.1841^x$ (origin at 1917), respectively.

TABLE 1.—*Approximate Domestic Consumption of Principal Petroleum Products for All Years for Which Data are Available**

(In millions of gallons)

	Gasoline			Kerosene			Gas and Fuel Oil			Lubricating Oils		
	Pro-duction	Exports	Domestic Consumption	Pro-duction	Exports	Domestic Consumption	Pro-duction	Exports	Domestic Consumption	Pro-duction	Exports	Domestic Consumption
1899	281	18	263	1259	733	526	305	22	283	170	71	99
1904	291	24	267	1357	761	596	360	34	326	315	90	225
1909	540	69	471	1675	1046	629	1702	122	1580	537	162	375
1914	1500	205	1295	1935	1010	925	3734	704	3030	517	192	325
1916	2059	356	1706	1455	855	600	4664	964	3715	625	251	364
1917	2851	416	2360	1727	658	1069	6513	1124	5580	754	280	474
1918	3570	559	3139	1825	491	1452	7321	1201	6077	841	257	582
1919	3958	372	3445	2342	979	1404	7627	618	7002	847	275	574
1920	4883	635	4279	2320	862	1404	8661	847	7945	1047	411	612

* Exports from U. S. Bureau of Foreign and Domestic Commerce. Production for 1899, 1904, 1909, and 1914 from U. S. Census of Manufacturers. Production for 1916–1920 from U. S. Bureau of Mines. Domestic consumption for 1917–1920 is adjusted for imports and changes in stocks, and hence is more accurate than figures for prior years.

with 162 per cent. for fuel oil, 88 per cent. for lubricating oils, and 52 per cent. for kerosene.

RELATION OF GASOLINE TO CRUDE PETROLEUM

Having observed the degree to which the domestic consumption of gasoline is outdistancing the other petroleum products in point of growth, we may examine the relationship that obtains between gasoline and crude petroleum, keeping clearly in mind that we are concerned with relative rates of growth, not absolute magnitudes. For this purpose, it is desirable to separate the supply of crude petroleum into domestic production and imports, and to plot against these factors the domestic production of gasoline. The growths of these three important items are compared, in Fig. 2, on a semilogarithmic scale, and to facilitate comparison the figures representing domestic production of crude petroleum are divided by 5 before plotting. This procedure not only lowers the curve (without affecting its slope) to a convenient position on the chart, but makes it possible for the curve to depict both the output of crude petroleum and the approximate natural-gasoline content of this output.[3]

TABLE 2.—*Relative Consumption of Petroleum Products in United States, 1914–1920*

(Consumption in 1914 = 100)

	Gasoline	Kerosene	Gas and Fuel Oil	Lubricating Oils
1914	100	100	100	100
1915	...	...	...	...
1916	122	65	122	112
1917	182	116	184	145
1918	242	157	200	179
1919	266	152	231	176
1920	330	152	262	188

The rates of growth of the curves in Fig. 2 may be accurately determined by calculating, by the method of least squares, the normal trend of the data given in Table 3; the results are shown in Table 4. The point to be especially noted is that the domestic production of crude petroleum has been increasing at an average rate of 7.6 per cent. a year; the domestic production of gasoline, 21.87 per cent. a year; and the imports of crude petroleum, 72.3 per cent. a year. If this country should continue to import crude petroleum at this rate, the imports for 1930 would be 50.3

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[3] The ratio of gasoline production to crude run to stills was 19.8 per cent. in 1916, a year when the gasoline supply was only slightly affected by increments from cracking, casing-head gasoline, etc. This ratio grew to 21.5 per cent. in 1917, 26.1 per cent. in 1918, 24.9 per cent. in 1919, and 24.3 per cent. in 1920. The slight decline in 1919 and 1920 was due to the large influx of Mexican oils.

billion barrels! Such a statement, of course, is of value merely to emphasize the abnormality of the rate that has characterized the development of the Mexican fields over the past decade; such a rate cannot be maintained for any extractive industry.

Fig. 2 shows a second relationship of importance—the turning point in the economic status of gasoline that came in 1917, approximately. Prior to 1917, the quantity of crude petroleum produced in the United States contained more gasoline than was required. Since then, the natural gasoline-producing capacity of the domestic output has fallen behind motor-fuel requirements to the degree roughly expressed by the angle between the two curves in Fig. 2. The gap has been filled by imports and a factor that may be designated "new technology," by which is meant cracking, casing-head gasoline, change in end point, etc.

We have now established, and given measurements therefor, that the demand for gasoline has grown more rapidly than the demands for other products of petroleum, that the demand for petroleum is primarily determined by the demand for gasoline, and that, in spite of the remarkable growth of domestic crude-oil output, the mounting gasoline requirements have already called into action new technology and phenomenally increasing imports of raw material. Such, in outline, is the situation from which we must view the future. Our problem is to project these trends as accurately as may be, taking into account such modifying factors as may be measured and appraised.

TABLE 3.—*Production of Crude Petroleum and Gasoline, Together with Imports of Crude Petroleum, in the United States, by Years, 1909–1920*
(In millions of barrels)

	Domestic Production of Crude Petroleum[a]	Domestic Production of Gasoline[b]	Imports of Crude Petroleum[c]
1909	183	12.9	0.07
1910	210	[d]	0.56
1911	220	[d]	1.71
1912	223	[c]	7.38
1913	248	[a]	17.81
1914	266	35.8	17.25
1915	281	[d]	18.14
1916	301	49.0	20.57
1917	335	67.9	30.16
1918	356	85.0	37.74
1919	378	90.0	52.82
1920	443	105.4	106.18

[a] U. S. Geological Survey.
[b] U. S. Census of Manufactures (1909, 1914) and U. S. Bureau of Mines.
[c] U. S. Bureau of Foreign and Domestic Commerce.
[d] No data available.

TABLE 4.—*Normal Trend of Domestic Production of Crude Petroleum and Gasoline, and Imports of Crude Petroleum, 1909–1920*

Item	Unit	Period Covered	Equation to Normal Trend	Origin	Normal Annual Increase
Domestic production of crude petroleum...................	Million barrels	1909–1920	$y = 267 \times 1.076^x$	1914	7.6 per cent.
Imports of crude petroleum.......	Million barrels	1910–1920	$y = 14.3 \times 1.723^x$	1915	72.3 per cent.
Domestic production of gasoline..	Million barrels	1909, 1914, 1916–1920	$y = 51.9 \times 1.2187^x$	1916	21.87 per cent.

EMPIRIC METHODS OF FORECASTING DEMAND

The simplest method of estimating future demand is by projecting current trend into the future. Thus, consumption for a recent period, say the past ten years, may be plotted, an equation calculated that gives a good fit to the data, and the future consumption for any given date calculated therefrom. Such equations are empiric, determined wholly from the data in the past, and, as well known in the physical sciences, are unreliable for extrapolation beyond the limits of the observations. Empiric methods may be used for projecting economic data only with the utmost caution and then only for a short time ahead. Such methods, however, are those generally used in forecasting and are applied to this problem, even though the results may finally be rejected. Empiric methods have the prime advantage of eliminating the personal equation; the results are identical, irrespective of the investigator; but the method eliminates good judgment as well as bad.

Empiric Projection of Crude Consumption

The annual consumption of crude petroleum in the United States for
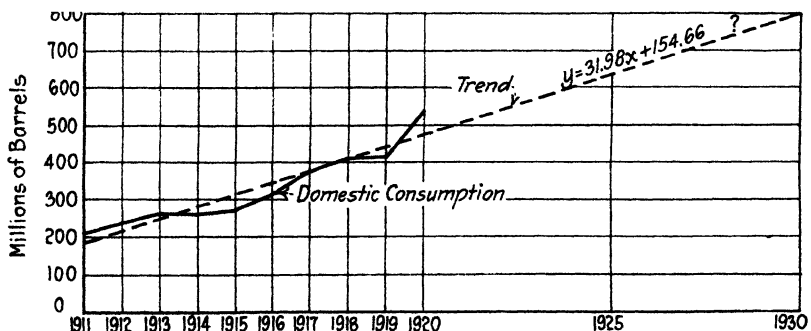
FIG. 3.—EMPIRIC PROJECTION OF DEMAND FOR CRUDE PETROLEUM IN UNITED STATES DETERMINED BY FITTING A STRAIGHT LINE TO DATA BY METHOD OF LEAST SQUARES.

the ten-year period, 1911–1920, is shown in Table 5, together with the

method of calculating consumption; the figures are taken from a report by
G. B. Richardson of the U. S. Geological Survey.   The increasing quantity
of crude petroleum consumed may be interpreted as an arithmetic growth
or else as a geometric progression of even or accelerating rate.   First a
straight line, then a compound-interest curve, and finally a parabola were
fitted to the data by the method of least squares, the equations of these
curves determined, and extrapolated values for 1925 and 1930 calculated.
The results are given in Table 6, while the straight-line and compound-
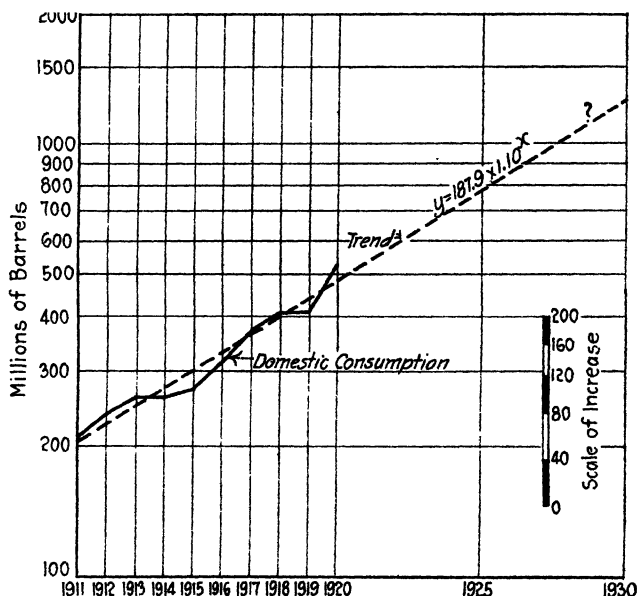
FIG. 4.—EMPIRIC PROJECTION OF DEMAND FOR CRUDE PETROLEUM IN UNITED
STATES DETERMINED BY FITTING A COMPOUND-INTEREST CURVE TO DATA BY METHOD
OF LEAST SQUARES.

interest curve are shown in Figs. 3 and 4.   It will be observed that the
extrapolated values are greater in the case of the compound-interest curve
and still greater for the parabola.

The method of least squares is such that an equation determined
thereby yields deviations the sum of whose squares is a minimum.   A
comparison of the deviations for the three curves is made in Table 7,
where it is apparent that the parabola, with a mean square deviation of
299, fits the data more closely than the compound-interest curve, with a
mean square deviation of 440, and that this, in turn, gives a closer fit
than the straight line, with a mean square deviation of 800.   But this
fact does not mean that either the parabola or the compound-interest
curve, if extended, will show more correctly than the straight line the

TABLE 5.—*Crude Petroleum Data for United States, by Years, 1911–1920*

(*In millions of barrels*)

	Domestic Production	Imports	Stocks, Dec. 31	Domestic Consumption[a]
1911	220	1.71	137	211
1912	223	7.38	123	240
1913	248	17.8	123	262
1914	266	17.2	142	261
1915	281	18.1	164	273
1916	301	20.6	162	319
1917	335	30.2	146	378
1918	356	37.7	122	413
1919[b]	378	52.8	128	418
1920[b]	443	106.2	134	531

[a] Domestic production plus imports minus exports plus decrease in stocks or minus increase in stocks.
[b] Preliminary figures.

future growth of crude petroleum consumption.[4] It is the writer's conclusion, in respect to the case in hand, that extrapolation with any empiric equation is subject to serious error, but that the exponential and

TABLE 6.—*Trend of Domestic Consumption of Crude Petroleum, by Years, 1911–1920, with Extrapolations for 1925 and 1930, According to Different Equations*

Calculated Consumption

Years	Observed Consumption	Straight-line $y = 31.98x + 154.66$ Origin at 1911	Compound-interest Curve $y = 187.9 \times 1.10^x$ Origin at 1911	Parabola $y = 322 + 35x + 3.066x^2$ Origin at 1916
1911	211	187	207	222
1912	240	219	228	230
1913	262	251	250	244
1914	261	283	275	264
1915	273	315	303	290
1916	319	347	333	322
1917	378	379	366	360
1918	413	411	403	404
1919	418	442	443	454
1920	531	474	487	510
1925	...	634	784	880
1930	...	794	1263	1412

[4] For an analogous conclusion in respect to pig iron, see Indices of Business Conditions, *Review of Economic Statistics* (Jan., 1919), 16.

TABLE 7.—*Comparison of Degree of Fit of a Straight-line, a Compound-interest Curve, and a Parabola to Consumption of Crude Petroleum, 1911-1920*

Year	Domestic Consumption of Crude Petroleum	Deviations of y from Straight Line		Deviations of y from Compound-interest Curve		Deviations of y from Parabola	
	y	d	d^2	d	d^2	d	d^2
1911	211	24	576	4	16	−11	121
1912	240	21	441	12	144	10	100
1913	262	11	121	12	144	18	324
1914	261	22	484	−14	196	− 3	9
1915	273	42	1764	−30	900	−17	289
1916	319	28	784	−14	196	− 3	9
1917	378	1	1	12	144	18	324
1918	413	2	4	10	100	9	81
1919	418	24	576	−25	625	−36	1296
1920	531	57	3249	44	1936	21	441
Sum..................		232	8000	177	4401	146	2994
Mean deviation.........		23.2	....	17.7	....	14.6	....
Mean-square deviation....		....	800	....	440	....	299
Standard deviation (σ)...		....	28.3	....	21.0	....	17.3

power functions are more unreliable than the straight line for extrapolating over a five- or ten-year period.

Empiric Projection of Gasoline Consumption

The domestic consumption of gasoline for the years for which accurate statistics are available is shown in Table 8, together with the method of calculating this item. By fitting a straight line to the data by the method of least squares, extrapolated values for 1925 and 1930 may be obtained as shown in Fig. 5 and in Table 8. The results are given for what they may be worth, but the writer again calls attention to the limitations of the empiric method.

Empiric Projection of Kerosene Consumption

The domestic consumption of kerosene (see Table 9 and Fig. 1) displays a recent rate of growth sharply at variance from that of the other petroleum products. For the period 1918-1921, the trend of this item has been downward. Whether this downward trend is temporary or permanent remains to be seen; good arguments in favor of either eventuality are available. Accordingly, a straight line has been fitted by the method of least squares, first, to the whole period 1916-1920, and second, to the last three years of that period when the trend was downward.

The shortness of a three-year period as a basis of projection is recognized, and the results are shown for what they may be worth without comment as to their probable correctness.
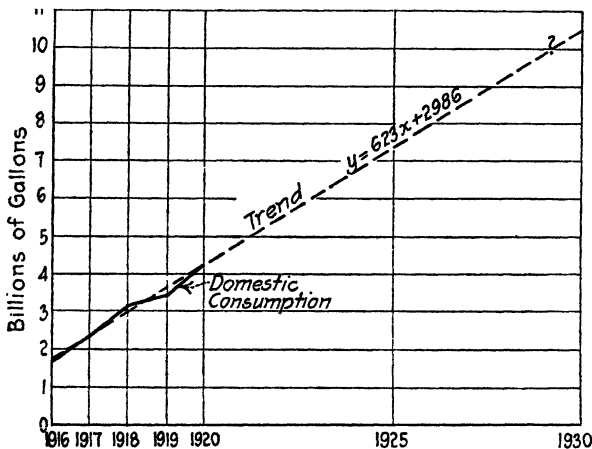
FIG. 5.—EMPIRIC PROJECTION OF DEMAND FOR GASOLINE IN UNITED STATES DETERMINED BY FITTING A STRAIGHT LINE TO DATA BY METHOD OF LEAST SQUARES.

Empiric Projection of Fuel-oil Consumption

The domestic consumption of gas and fuel oil for the period for which accurate statistics are available is shown in Table 10, together with extrapolated values for 1925 and 1930. It should be emphasized, however, that an empiric projection is of especially doubtful value for fuel oil, because of technical factors such as the salt-water invasion in Mexico and the increase in use of cracking methods in refining.

TABLE 8.—*Domestic Consumption of Gasoline, 1916–1920, Together with Extrapolated Values for 1925 and 1930, According to a Straight-line Projection ($y = 623x + 2986$, origin at 1918) Determined from Data by Method of Least Squares*

(*In millions of gallons*)

Year	Domestic Production[a]	Imports[b]	Exports[b]	Stocks[a] End of Period	Change in Stocks	Domestic Consumption, Actual	Domestic Consumption, Calculated from Equation
1916	2,059	3	356	327	.....	1,706[c]	1,740
1917	2,851	10	416	412	+ 85	2,360	2,363
1918	3,570	13	559	297	−115	3,139	2,986
1919	3,958	9	372	447	+150	3,445	3,609
1920	4,883	46	635	462	+ 15	4,279	4,232
1925	.....	..	...	...	.....	.....	7,347
1930	.....	..	...	...	.....	.....	10,462

[a] From U. S. Bureau of Mines.
[b] From U. S. Bureau of Foreign and Domestic Commerce.
[c] Slight error introduced because change in stocks is not known.

TABLE 9.—*Domestic Consumption of Kerosene for 1916–1920, with Extra-polated Values for 1925 and 1930, According to Straight-line Projection* ($y = 154x + 1186$ for 1916–20, and $y = -24x + 1420$ for 1918–20, origin at 1918) *Determined by Method of Least Squares*
(*In millions of gallons*)

Year	Domestic Production[a]	Exports[b]	Stocks[c]	Change in Stocks	Domestic Consumption. Actual	Domestic Consumption, Calculated from Equation	
						$y = 154x+1186$	$y = -24x+1420$
1916	1455	855	477	...	600[d]	878	
1917	1727	658	498	+ 21	1069[d]	1032	
1918	1825	491	380	−118	1452	1186	1444
1919	2342	979	339	− 41	1404	1340	1420
1920	2320	862	393	+ 54	1404	1494	1396
1925	....	...	...	.....	....	2264	1276
1930	....	...	...	.....	....	3034	1156

[a] From U. S. Bureau of Mines.
[b] From U. S. Bureau of Foreign and Domestic Commerce.
[c] Slight error introduced because of probable small quantity of imports.
[d] Slight error introduced because change in stocks is not known.

Empiric Projection of Lubricating-oil Consumption

The domestic consumption of lubricating oils for the period for which accurate statistics are available is shown in Table 11, together with extra-
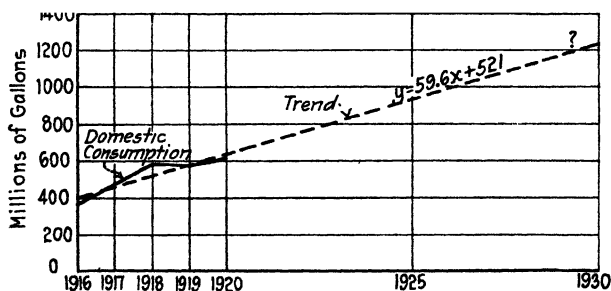
FIG. 6.—EMPIRIC PROJECTION OF DEMAND FOR LUBRICATING OILS IN UNITED STATES DETERMINED BY FITTING A STRAIGHT LINE TO DATA BY METHOD OF LEAST SQUARES.

polated values for 1925 and 1930 determined by fitting the equation of a straight line to the data by the method of least squares. The results are shown graphically in Fig. 6. As the consumption of lubricating oils is made of two components, industrial oils and motor oils, each growing at different rates, the empiric method of projection given is not apt to yield reliable results.

Criticism of Empiric Projections

The objection to empiric methods of projection is that, while the results may closely correspond to subsequent conditions, again they may not, and there is no way of telling in advance their reliability in any particular instance. Just as in the physical sciences, so with economic data, empiric equations serve admirably for interpolation but are unreliable for extrapolation. The problem of estimating future petroleum requirements has been attacked empirically in order to show what may be expected if present tendencies continue. This treatment of the matter does not, and cannot, discriminate as to whether present tendencies are likely to continue or to suffer modification; that question is discussed later in this paper.

It is the writer's opinion that for the problem in hand, the straight-line projection is more reliable than either the compound-interest curve or the parabola, but that results so obtained must be used with extreme caution, and that, on the whole, empiric methods do not offer a satisfactory solution of our problem.

TABLE 10.—*Domestic Consumption of Gas and Fuel Oil*, 1916–1920, *Together with Extrapolated Values for* 1925 *and* 1930 *According to Straight-line Projection* ($y = 988x + 6064$, *origin at* 1918) *Determined from Data by Method of Least Squares* (*In millions of gallons*)

Year	Domestic Production[a]	Imports[b]	Exports[b]	Stocks[a] End of Year	Change in Stocks	Domestic Consumption, Actual	Domestic Consumption, Calculated from Equation
1916	4,664	15	964	721	.....	3,715[c]	4,088
1917	6,513	48	1,124	578	−143	5,580	5,076
1918	7,321	38	1,201	659	+81	6,077	6,064
1919	7,627	48	618	714	+55	7,002	7,052
1920	8,861	64	847	847	+133	7,945	8,040
1925	.....	..	.....	...	.....	.....	12,980
1930	.....	..	.....	...	.....	.....	17,920

[a] From U. S. Bureau of Mines.
[b] From U. S. Bureau of Foreign and Domestic Commerce. The figures under imports are those classified as refined oils other than benzine, gasoline, and naphtha; they are assumed by author to represent dominantly fuel oil, but they probably also include a little kerosene.
[c] Slight error introduced because change in stocks is not known.

RATIONAL METHODS OF FORECASTING DEMAND

Although future requirements may be estimated by projecting observed rates of change, by appropriate mathematical methods, no allowance is made for modifying influences that may affect the projected rates of growth. Can such allowance be made scientifically? Can we utilize past data and at the same time introduce factors drawn from outside the

TABLE 11.—*Domestic Consumption of Lubricating Oils, 1916–1920, Together with Extrapolated Values for 1925 and 1930, According to Straight-line Projection ($y = 59.6x + 521$, origin at 1918) Determined from Data by Method of Least Squares*
(*In millions of gallons*)

Year	Domestic Production[a]	Exports[b]	Stocks[a] End of Year	Change in Stocks	Domestic Consumption, Actual	Domestic Consumption, Calculated from Equation
1916	625	261	...	....	364[c]	402
1917	754	280	137	....	474[c]	461
1918	841	257	139	+ 2	582	521
1919	847	275	137	− 2	574	581
1920	1047	411	161	+24	612	640
1925	....	...	...	....	...	938
1930	....	...	...	....	...	1236

[a] From U. S. Bureau of Mines.
[b] From U. S. Bureau of Foreign and Domestic Commerce.
[c] Slight error introduced because change in stocks is not known.

data, so as to arrive at sounder results than may be gained from the relatively simple methods thus far reviewed? Fortunately, there are rational methods that hold some promise in this connection.

In the rational approach to the problem, we must shift our attack from the rates of growth of the quantities of mineral oils consumed to the rates of growth of the industrial and transportation activities demanding these products. And, in the interest of brevity, we may simplify the problem by limiting our consideration to gasoline, on the assumption that all other petroleum requirements are, for the time being,[5] subordinate to the need for motor fuel.

Car Density Method of Projecting Gasoline Requirements

An estimate of the growth of motor-vehicle registrations to 1940 was recently worked out on the basis of an assumed car density for that year and the results plotted as a projected curve.[6] The projection was calculated by observing that in 1919 the car density in California was one car for every 6.07 persons, in Iowa one car for every 6.12 persons, and in Nebraska one car for every 6.54 persons; assuming that in 1940 there would be one motor vehicle for every 5 persons, the total number of vehicles being equally divided between passenger cars and trucks; using the projected curve of population worked out by the engineers

[5] Lubricants are probably universally recognized as fundamentally more important than motor fuel, but there is no immediate problem of lubricant supply so long as crude petroleum in excess of the requirements of lubrication is available.

[6] The investigation, made by the Commercial Research Department of the Franklin Automobile Company, was briefly described in *Automotive Industries* (Apr. 28, 1921), 894–896.

of the New York Telephone Co.; and plotting a smooth curve from the observed data of 1910–1919 to the year 1940, so as to reach the assumed ratio of vehicles to population on that date. The purpose of the investigation was not to forecast results for so remote a period as 1940, but to obtain workable figures for a few years ahead. The theory is that if the assumption of the ultimate ratio is made for a date far enough ahead, even a material error in that assumption would not seriously alter the forecast for the next few years.

According to this projection, there will be in use in 1925 approximately 10,000,000 passenger cars and 5,100,000 trucks; and in 1930, about 12,200,000 passenger cars and 9,200,000 trucks. If we multiply the number of cars by the average annual consumption of gasoline per unit and make a similar calculation for trucks,[7] we will arrive at an estimate of automotive-fuel requirements for 1925 and 1930, as shown in Table 12.

TABLE 12.—*Estimated Domestic Consumption of Gasoline, in 1925 and 1930, on Basis of Automotive Projections made by Commercial Research Department of Franklin Automobile Co. and Present Estimated Consumption Factors*
(*In millions of gallons*)

Year	Estimated Consumption by Passenger Cars	Estimated Consumption by Trucks	Estimated Consumption by Cars and Trucks
1925	3,350	5,100	8,480
1930	4,087	9,200	13,287

The calculation as given above, while having points of interest, is subject to criticism. In the first place, the assumption that the ratio of motor vehicles to population in 1940 will be 1 to 5 for the country as a whole is scarcely justified on the basis of a present attainment of 1 to 6 in certain highly individualistic states. In the second place, the assumption that the number of motor vehicles in 1940 will be evenly divided between passenger cars and trucks is open to question, as making both cars and trucks a like function of population, whereas the two are undoubtedly subject to different limitations. While recognizing the value of this projection for certain purposes, therefore, the writer does not accept the results as a satisfactory solution of the problem in hand.

Use of Prescott Formula in Projecting Gasoline Requirements

R. B. Prescott, of the Class Journal Co., has applied to economic data a formula devised some years ago by Gompertz and well known in ac-

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[7] The consumption factors selected are 335 gal. per year for passenger cars and 1000 per year for trucks. The determination of these factors is explained in Pogue, "The Economics of Petroleum," 123–124, 1921.

tuarial work, which involves the principle of growth at a diminishing rate; or in other words the principle of diminishing returns. The increase in population in any given country, for example, if plotted on rectangular coördinates, traces a curve of double flexure, in its earlier period concave upward and later on concave downward. Thus, the growth is at first slow, then more rapid, and later slows down as the country approaches mature and stable development.

The consumption of many commodities is a function of population and, as such, increases at first slowly, then rapidly, and then slowly, following the trace of the Gompertz curve. As a concrete example take the automobile; for a time, the growth in its use is slow as this new agency of transport passes through a stage of experimentation; then the automobile finds itself, so to speak, and an era of rapid expansion follows; subsequently the progress of expansion meets resistance as the number of automobiles approaches the maximum number that the population can use; and finally the rate of expansion approximates the rate of increase in population.[8]

The registrations of motor vehicles in the United States by years for the period 1912–20 are shown on a semilogarithmic scale in Fig. 7. To the registration figures is fitted a curve of the type, $y = ab^{e^x}$, which is projected to 1930. This curve, which is published by permission of Mr. Prescott, is thought to represent the trace of the normal growth of the automotive industry, and, if so, affords a rational basis for forecasting not only future demand for motor vehicles but the future requirements of motor fuel.[9] Curves of this type have been tested on pig iron, potatoes, population, wheat, sugar, steel, railroad mileage, and cotton; and it has been found that the equation, fitted to data up to 1900, when extended to 1920 gave extrapolated values for 1920 corresponding to the actual figures within errors of 2 to 10 per cent. While the application of this technique is still in its developmental stage, the method gives promise of becoming a useful tool for predetermining the normal growth of industrial activities and market requirements.[10]

The projection of motor-vehicle registrations, as shown in Fig. 7, indicates clearly that the further growth of automotive transportation will be at a decelerating rate. The projection of the registration figures in this manner gives 14,400,000 motor vehicles (cars and trucks) as the probable number for 1925 and 17,500,000 as the probable number for 1930.

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[8] In practice the matter is not so simple, as the growth at every stage is modified by fluctuations in buying power, price, etc., but variations from the normal may in the long run be expected largely to compensate one another.

[9] For a fuller exposition of the mathematics of the Prescott equation, see *Automotive Industries* (Nov. 17, 1921), 954.

[10] Mr. Prescott and the present writer are at work jointly on the further application of this and other methods to industrial data, and the outline of the theory given above is preliminary to a more detailed discussion to be published later.

These extrapolated values may be converted into gasoline by multiplying them by the average gasoline consumption per unit (assuming that the consumption factor will not change materially); or else by fitting the normal registration curve to the curve representing domestic gasoline
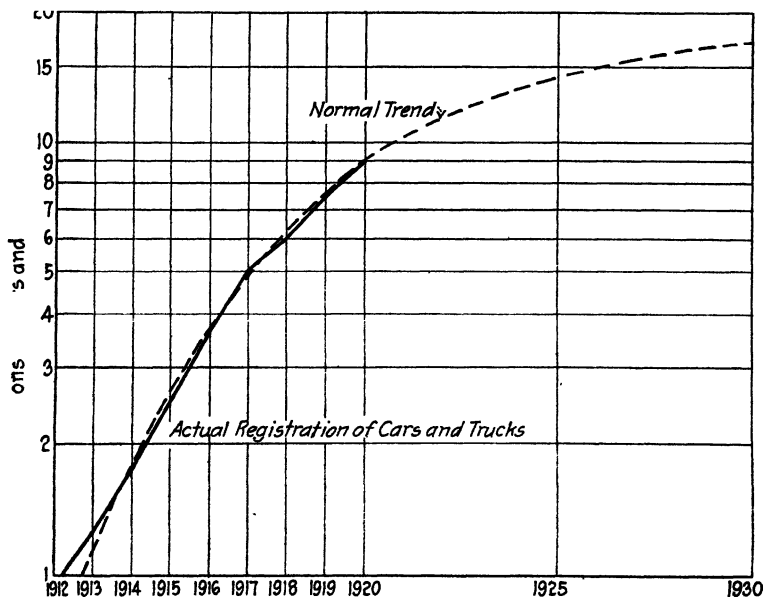
FIG. 7.—PROJECTION OF MOTOR-VEHICLE REGISTRATIONS IN UNITED STATES BY MEANS OF PRESCOTT FORMULA INVOLVING PRINCIPLE OF DIMINISHING RETURNS.

consumption and taking the trace of the former as the probable future course of the latter. The second method, shown in Fig. 8, projects the ratio between motor vehicles and domestic gasoline consumption and therefore yields figures representing total consumption, instead of the consumption of cars and trucks alone. The results of the two methods are given in Table 13.

TABLE 13.—*Estimated Consumption of Gasoline (and Other Motor Fuels) in United States in 1925 and 1930, Based on Application of Prescott Formula to Automotive Registrations*

1 Year	2 Number of Motor Vehicles, Millions of Units	3 Consumption of Gasoline[a] by Cars and Trucks, Column 2×400, Millions of Gallons	4 Consumption of Gasoline[a] by Cars, Trucks, Tractors, etc., Projection of Ratio, Millions of Gallons	5 Difference, Column 4 minus Column 3, to be Credited to Tractors and Miscellaneous Uses, Millions of Gallons
1925	14.4	5760	6800	1040
1930	17.5	7000	8300	1300

[a] Gasoline should be read in this table as gasoline and other motor fuels.

Several refinements, theoretically, are possible with this method. For example, passenger cars, trucks, and tractors might be advantageously projected as separate items, but accurate statistical data are lacking on these individual series. Also, the domestic consumption of gasoline could be projected directly from its own data; but here again accurate statistical data are available for only such a brief period as to render preferable the indirect graphic method shown above. Accordingly, it seems better to let the investigation rest for the time being with the generalized results given in Table 13.

Summary of the Several Methods of Forecasting Gasoline Demand

A comparison of the results of projecting the future gasoline requirements of the country by the three methods outlined is given in Table 14.

TABLE 14.—*Comparison of Estimated Future Demand for Gasoline in United States*

(In millions of gallons)

	1923	1925	1930
Straight-line projection	6,101	7,347	10,462
Vehicle to population ratio	5,748	8,480	13,287
Prescott formula	5,900	6,800	8,300

It will be noticed that while the methods yield widely different results for 1930 they are in reasonably close agreement for 1923. This concordance for the nearer date speaks for the practical value of mathematical methods of projection in general. Emphasis should be again laid on the fact that the need for accurate estimates is greater for the next few years than for a period five to ten years ahead, and all three methods meet this important criterion. The projection by the Prescott formula, however, is given as the most probable estimate of future demand and one that may be used as a working basis for long-range planning.

DISCUSSION OF CHANGES IN TECHNOLOGY

A rational estimate of the demand for motor fuel in 1925 and 1930 is 6800 million gallons and 8300 million gallons, respectively. These demands may be converted into corresponding demands for crude petroleum on the basis of the ratio between the consumption of crude petroleum and the production of gasoline in 1920, which ratio was 19.9 per cent. A conversion on this basis, however, is not valid, for it makes no allowance for a growing extraction of gasoline from its raw material, increasing efficiency in its utilization, and the employment of supplementary fuels. To forecast these changes in technology offers even greater difficulty than the problem of forecasting straight demand, and

the results accordingly are doubtless subject to a greater degree of error. The probable degree to which crude petroleum will be converted into motor-fuel service in 1925 and 1930 is indicated in Table 15, on the basis that the recent increase in the ratio between gasoline production and crude consumption will continue.
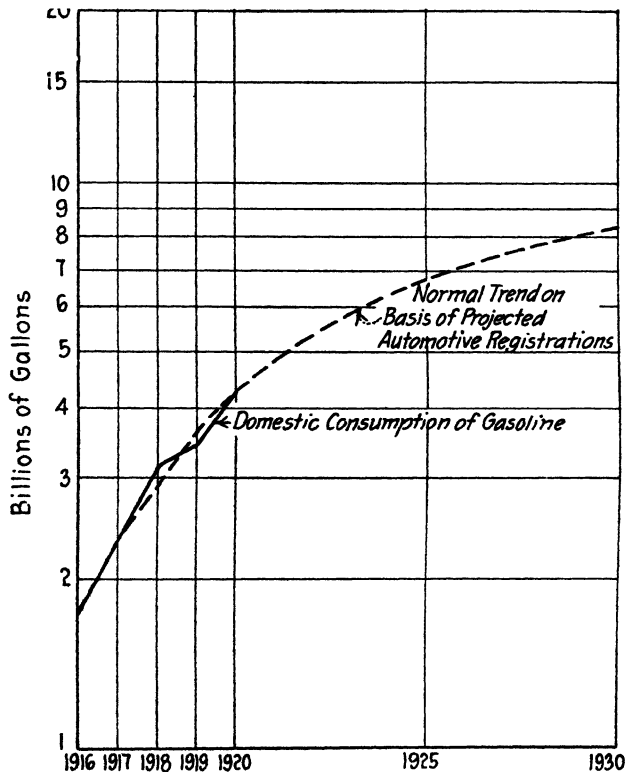
FIG. 8.—PROJECTION OF DEMAND FOR GASOLINE IN UNITED STATES DETERMINED BY FITTING CURVE REPRESENTING PROJECTION OF MOTOR-VEHICLE REGISTRATIONS (FIG. 7) TO DATA REPRESENTING DOMESTIC CONSUMPTION OF GASOLINE FOR 1916–20. RESULTS ARE GIVEN AS BEING MORE RELIABLE THAN THOSE OBTAINED BY EMPIRIC METHODS.

It appears, from Table 15 (see also Fig. 9), that in 1925 over a quarter of our total consumption of crude petroleum will be converted into motor-fuel service and that by 1930 this proportion will have increased to one-third. These factors, to repeat, are on the basis of a continuation of the increase in the motor-fuel crude-petroleum ratio that characterized the period 1916–1920. Should the change in technology be speeded up, the ratio will be proportionally higher.

These factors, however, may be tentatively used as they stand for

TABLE 15.—*Ratio of Production of Motor-fuel to Consumption of Crude Petroleum, 1916–1920, Together with Extrapolated Values for 1925 and 1930, According to a Straight-line Projection* ($y = 1.24x + 19.06$, *origin at* 1918) *Determined From Data by Method of Least Squares*

	1 Crude Consumed, Million Barrels	2 Gasoline Produced, Million Barrels	Ratio of Gasoline Produced to Crude Consumed	
			Calculated	Actual
1916	319	49	0.166	0.154
1917	378	68	0.178	0.180
1918	413	85	0.191	0.206
1919	418	90	0.203	0.214
1920	531	105	0.215	0.199
1925	...	...	0.277	
1930	...	...	0.339	

converting our best estimates of the future demand for motor fuel into a corresponding demand for crude petroleum, as shown in Table 16.

TABLE 16.—*Conversion of Estimated Demand for Gasoline, in 1925 and 1930, as Determined in Table 13, into Estimated Demand for Crude Petroleum, by Use of Ratios in Table 15*

Year	Demand for Gasoline, Million Gallons	Factors	Demand for Crude Petro- leum, Million Barrels
1925	6800	$\div 0.277 \times 42 =$	585
1930	8300	$\div 0.339 \times 42 =$	580

According to Table 16, and the line of analysis leading thereto, a crude petroleum demand of 585 million barrels in 1925 and 580 million barrels in 1930 is indicated. These amounts are much smaller than those arrived at by means of the projections summarized in Table 6. In addition, the figures show that the maximum demand is reached about 1925 and that this demand is maintained practically unchanged until 1930. While the results must be looked upon in a very critical spirit, the calculation is suggestive that a maximum volume of crude petroleum of approximately 600 million barrels, under the influence of continued progress in technology, will prove ample to maintain the expected expansion in automotive transportation over the next decade.

Should the volume of crude petroleum available for consumption fall below this estimated demand of 600 million barrels, the effect will be to stimulate a greater advance in technology as an offsetting factor. On the other hand, should a greater volume be brought upon the market, the

margin will doubtless be readily absorbed although with a complementary slowing down in technical advance.

CONCLUSIONS

1. The demand for crude petroleum for the next ten years will be determined by the growth of automotive transportation and the motor-fuel requirements resulting therefrom.
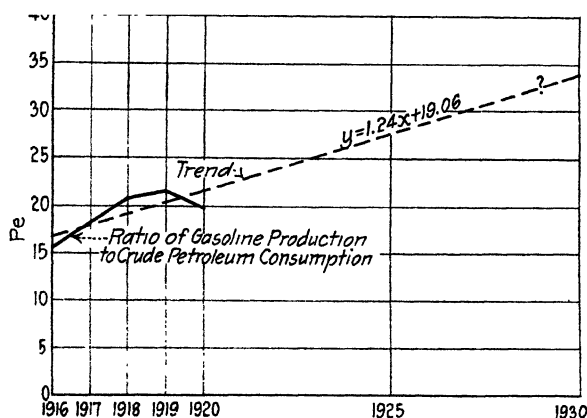
FIG. 9.—TENTATIVE PROJECTION OF IMPROVEMENT IN TECHNOLOGY AFFECTING MOTOR FUEL, DETERMINED BY PROJECTING RATIO BETWEEN PRODUCTION OF GASOLINE AND CONSUMPTION OF CRUDE PETROLEUM BY FITTING A STRAIGHT LINE TO DATA BY METHOD OF LEAST SQUARES.

2. Future consumption of crude petroleum and its products may be estimated by methods of empiric projection, and the results attained by such methods are given, but such methods are not regarded as satisfactory.

3. Two rational methods of forecasting the demand for gasoline are described and applied, and a method involving the principle of diminishing returns is advanced as the most satisfactory approach to the problem.

4. By means of the method involving the principle of diminishing returns, the domestic demand for gasoline is estimated to be 6800 million gallons in 1925 and 8300 million gallons in 1930.

5. The ratio between the quantity of crude petroleum consumed and the quantity of gasoline produced is empirically projected, by fitting a straight line to the data by the method of least squares, and according to this method 27.7 per cent. of our crude petroleum consumption will be converted into motor-fuel service in 1925 and 33.9 per cent. in 1930, as contrasted with a percentage of 19.9 for 1920. This projection is subject to a greater probable error than the estimate of gasoline requirements.

6. Combining the results mentioned in the two preceding paragraphs, the estimated demands for gasoline in 1925 and 1930 are converted into demands for crude petroleum. The results attained indicate that 585

million barrels of crude petroleum will be sufficient to support automotive transportation in 1925 and 580 million barrels will be sufficient for 1930, on the basis of a continuation of recent progress in technology.

7. The results attained are subject to more accurate determination on the further development of the methods followed, but the present results are advanced as having sufficient probability to be of practical value as a basis of present planning.

DISCUSSION

JAMES O. LEWIS, Marietta, Ohio (written discussion).—Petroleum engineers are indebted to the author for the novel methods for estimating future demands on the oil industry that he gives, particularly the curves and methods of Gompertz and Prescott; but the application of these methods and the conclusions derived are open to question. Future demand or future consumption of petroleum cannot safely be predicated solely on any estimates of the future demands for gasoline by automotive engines. Nor does the changing ratio of gasoline to other products correctly gage the possibilities of the markets for such products.

Fuel oil is more nearly the key to future demands than gasoline. It is quantitatively a greater product than gasoline, kerosene, or lubricants, even greater than their sum, and is the only product with a large competitive market always available. As a substitute for coal, fuel oil has practically the whole coal market as a competitive field and, by replacing coal, the fuel-oil market can be expanded quickly under any price stimulus, for fuel oil can replace coal on short notice with but minor changes in equipment. Gasoline, kerosene, and lubricants have, practically considered, inelastic and non-competitive markets. They have no quickly available substitutes nor can they replace in any considerable quantities substances in other fields. If there is a surplus of one or more of these products, this surplus must be stored or substituted, in as large part as is practicable, for the products in greater current demand.

The writer's meaning of the term "inelastic, non-competitive markets" can best be illustrated by considering lubricants. Lubricating oils are essential for every operating piece of machinery but the cost of lubricants is a small part of the cost of operating the machine. The demand for lubricants is in direct proportion to industrial activity and to general prosperity. If the factories, railroads, and automobiles are in active use, the market for lubricants is large; but it will not absorb more than its maximum current needs, together with reasonable stocks, no matter how cheap lubricants may be at the time. Nor will cheap lubricants materially enlarge the market in times of depression. The market enlarges permanently only as a result of industrial growth or growth in the number and use of automobiles or other new types of machines.

As only a part of the lubricating fractions are extracted from the crude oil, the only outlet for the surplus lubricating fractions is letting them remain in the fuel oil and be consumed in the same inferior uses. Lubricating-oil consumption expands and contracts almost independently of crude production, consumption, or prices. It does, however, follow closely the growth in population and wealth in accordance with the formulas of Gompertz and Prescott. Gasoline and kerosene, likewise, have inelastic non-competitive markets though not to such a limited degree. The gasoline market is practically limited to the number and activity of gasoline-using internal-combustion engines. When their demands have been supplied, the market can be increased comparatively little by price stimulation because gasoline is but one of the costs of operation. The market can be materially enlarged only by growth of the automotive industry, which is concerned more with other factors than with the price of gasoline.

The situation relating to the fuel-oil market is fundamentally different; there is always a large, quickly available, cheap market. As soon as the price goes down and reasonable assurance of supply can be given, fuel oil invades the coal market. In 1920, the fuel-oil market for ordinary heating or steaming raising purposes was equivalent to 70,000,000 tons of coal, or about a ninth of the coal consumed. The potential market for fuel oil is therefore enormous, even in times of depression, and the limits are economic rather than mechanical. The extent to which fuel oil can supplant coal is shown by the situations in the Southwest and on the Pacific coast.

The proportion of fuel oil produced can be varied greatly in accordance with the desires of the refiner. By cracking, fuel oil and lubricants can be made largely into gasoline and kerosene. Probably in a few years all fuel oil, even the heavy Mexican crude, can be processed into gasoline if there is the demand; but on the other hand almost all the crude oil, except the gasoline, can be used for fuel oil if necessary. But as the fuel-oil market is the cheapest, effort is always made to decrease the proportion of fuel-oil output. To the refiner, fuel oil is what is left after the demands for the more profitable products have been supplied or the limits in processing the heavier fractions into gasoline or kerosene have been reached. When the demand for these products is large, relative to the supply of crude, as much as possible of the oil is made into the more desirable products; when these markets are glutted, more of the crude must be dumped into the fuel-oil market to sell for what it will bring. The price of crude oil will depend largely on the proportion that must be sold in the fuel-oil market at the time. If all the crude oil must be sold as fuel oil, the price at the well must be less than the selling price of fuel oil; but if a large part of the crude oil can be sold as the higher price products, the price of the crude oil will be proportionately higher. In

1920, about 58 per cent.[11] of the crude oil produced and imported was consumed as fuel oil but 80 per cent. could have been so used had it been necessary and if marketed cheap enough to undersell coal.

The lack of assurance of supply rather than price has hindered the expansion of the fuel-oil market. In the fall of 1918, there was a fuel-oil famine; in the spring of 1919, a surplus; early in 1920, another famine; and in the fall of the same year, another surplus. Such rapid changes obviously would deter prospective consumers. The fallacy involved in basing estimates on the anticipated gasoline demands is brought out by considering the relation of the other products to the estimates of the demands for crude, as shown in the following tabulation which is adapted from the author's figures.

Year	Estimated Crude Demand	Estimated Gasoline Demand	Remainder	Per Cent. Gasoline to Crude
	In Millions of Barrels, of 42 Gal.			
1920	531	105	426	19.9
1925	585	162	423	27.7
1930	580	197	383	33.9

The remainder includes kerosene, gas and fuel oil, lubricants, miscellaneous products and refinery losses. It is to be noted that the author's estimates assume decreasing demand for these other products and that the remainder for 1930 is 43 million barrels less than for 1920; in fact the decrease is more, for the refinery losses by increased cracking will appreciably increase this figure. There is of course no reason other than high prices to anticipate decreased demands for other products than gasoline.

Nor has due allowance been made for the growth of uses now in the incipient stage, such as household heating and Diesel engine fuel, both of which seem destined to large increases in the next decade. Fuel oil for marine use should be given especial attention. A growing demand for fuel oils in ways that will command better prices can be looked forward to.

But now, fuel oil is the weakest point in the machinery for marketing petroleum products and the industry has so far shown itself incompetent to handle fuel oil satisfactorily. Because the fuel-oil market can be quickly expanded or contracted and because the proportion of fuel oil from the crude can be quickly expanded or contracted from, perhaps, less than 50 per cent. to, perhaps, 80 per cent. of the whole—a flexibility which will surely be greater in the future as cracking increases—fuel oil

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[11] Apparently the author's figures on gas and fuel oil include only refinery products and do not include crude oil consumed directly as fuel.

can act as a buffer to adjust under or overproduction of crude oil to the existing markets. Here is the key to the stabilization of the oil market and farsighted methods of marketing fuel oil, of storage, and of credits should be devised whereby the recurrent fluctuations inherent to oil production might be made less detrimental to both the producer and the consumer.

Because of the large potential market for petroleum as fuel oil, future demand is substantially the same as future production. But to a large extent production is the result of prevailing prices and costs. If the profits in sight are attractive, the production is likely to meet the existing demands, though there may be a lag of one to two years between production and price stimulus or deterrent. However, at present apparently we cannot meet from domestic production all demands at the prices of recent years; but future demands will be governed largely by future prices. If such prices are not attractive, imports will be automatically restricted. The quantity of imports will depend on the cost of producing and importing the foreign oil and on the rapidity with which it can be found, developed, and brought to this market. If foreign oil is obtainable in large amount and at low cost, it means low prices and large demand here, but if the foreign oil cannot be obtained or is costly to produce, oil prices will be correspondingly high here and demand restricted accordingly to the higher uses.

The Oklahoma producer was not perturbed on account of Mexican oil in 1919 when he was getting $3.50 for his own production, because, for lack of tankers, Mexican oil could not be brought in fast enough to flood the market. Nor is the oil man perturbed now by the vast shale-oil reserves in the west as he knows that shale oil will become an important competitor only when the price of oil will be quite satisfactory to him. No matter how vast may be the oil reserves abroad, they will not glut our markets unless they can be developed with comparative rapidity and at comparatively low cost. If the Mexican light-oil fields prove to be exceptional and the other reserves in Latin America are of the nature of Panuco with its smaller wells, or of the Tehuantepec district with its salt domes, or like Trinidad with its loose-sand troubles, or with the oil inland and difficult of access or under political restrictions, the menace of foreign competition can be viewed with equanimity.

The point is that one cannot say there will be such and such a demand for petroleum without also stipulating something as to the price. The author's methods, if applied to kerosene, lubricating oil and other important products, also taking into account incipient uses and growing foreign competition, should give reasonable estimates for the products in the non-competitive markets where growth relates more closely to growth in population and wealth, which is the basis of Prescott's curves. But the method fails when applied to fuel oil without considering price

because the future demands of fuel oil relate more to underselling coal in an existing market than on growth in population or wealth. The tendency would seem to be for higher prices for oil and for demands to be restricted accordingly, but unless the obstacles to foreign exploitation prove greater than they now appear, it is likely that considerably more oil will be marketed in 1925 and in 1930 than the author has estimated. Should his calculations prove correct, it will mean highly satisfactory prices for those fortunate enough to own production, for there will be only enough oil available for the higher priced markets.

Canadian Oil Reserves

BY RALPH ARNOLD AND WALTER A. ENGLISH, NEW YORK, N. Y.

THOUGH production began in Canada only a short time after the discovery of oil in the United States, it has never attained large proportions, and if we were to judge entirely by the past the reserves of Canada would be put at a very low figure. There are only two areas from which oil is marketed at the present time; one, the old district around Petrolia in southern Ontario, and the other is the small Sheep Creek field near Calgary in Alberta. The former district has produced between 200,000 and 800,000 bbl. annually for the past 40 years, recent figures being near the minimum amount. Sheep Creek has produced about 12,000 bbl. annually for the past five years. The known reserves in proved territory in each of these fields are exceedingly small, when considered in terms of world reserves.

The chief hope of Canada for reserves of considerable magnitude lies in the western provinces. Sedimentary rocks similar to those in the productive fields of the United States occupy thousands of square miles in the western provinces, so that it seems likely that somewhere within this great area the peculiar conditions necessary for the presence of oil pools will be found. Prospecting is being carried on energetically by several large companies, as well as by many independents. The Dominion Government, which controls most of the public lands, is pursuing a fairly liberal policy toward the prospector, which should stimulate wildcatting, although the province of British Columbia, which controls public lands within that province, is at present unwilling to give adequate encouragement to make wildcatting attractive.

As there is really no assurance that any productive pools will be found in western Canada, the estimates of reserves are subject to a wide possibility of error. In making estimates, the various areas within which geologic conditions are more or less uniform are discussed separately; the estimates given for each area are not given with the idea that there is a reasonable probability of their eventually proving to be correct. It is hoped that an average of these estimates, taken together with those made for other undeveloped countries, will give a correct total of reserves

for all areas considered, even though the estimates are much too large in some cases and too small in others.

The estimates are made by comparing the Canadian areas to geologically similar districts in the United States where development has been sufficient to prove fairly definitely what the ultimate resources will be. It is assumed that the current estimates for total resources of each of the main producing areas of the United States are not greatly in error and, with this as a basis, an attempt is made to arrive at a figure that represents the probable maximum production of each of the Canadian areas, provided its production proves to be as great as the most similar area in the United States. A second factor is then introduced, to give the authors' view as to the probability of such a result being achieved. This figure is based both on an analysis of the conditions peculiar to the Canadian area being considered, and on the proportion of the total area with roughly similar conditions in the United States which has proved to be part of a productive district.

The estimates are for the following geographic divisions: Eastern Canada, southeastern and central plains of Alberta and southwestern Saskatchewan, northern plains of Alberta, Rocky Mountain front of Alberta and British Columbia, Mackenzie River district.

EASTERN CANADA

The greater part of eastern Canada is occupied by the shield of pre-Cambrian rocks in which there is no chance of finding oil. In southern Ontario and Quebec are areas of nearly flat-lying Palaeozic rocks from which oil has been produced on the southwestern tip of Ontario, in the Petrolia district. The rest of this area appears to be about as favorable geologically, except for the absence of seepages, which characterize the Petrolia area. Prospecting, although extending over a long period of time has shown discouraging results, except in the Petrolia district, where a few small pools are discovered from time to time. An estimate of 300,000 bbl. annually for 20 years seems fair for the Petrolia district, and 4,000,000 bbl. for the rest of the province, giving 10,000,000 bbl. as the reserves of eastern Canada.

SOUTHERN AND CENTRAL PLAINS OF ALBERTA

The prospective oil fields of the plains region of central and southeastern Alberta and southwestern Saskatchewan derive their supply from the basal Cretaceous beds, the southern extension of the big tar sands of the Athabasca on the north, and the northern extension of the Montana oil-bearing beds on the south. The oil-bearing formations in the belt from Lower Slave Lake to southwestern Saskatchewan lie on the eastern flank of the great Edmonton syncline and are affected by several cross

structures, including anticlines and domes, favorable for oil accumulation. The depth to the oil horizon varies from 2500 to 4000 ft. (760 to 1200 m.). Three important commercial gas fields have been developed in the region in sands overlying the oil horizon.   The areas of the possible and probable oil land, if oil is found at all in commercial quantities, is as follows:

NAME	POSSIBLE AREA, Sq. Mi.	PROBABLE AREA, Sq. Mi.
Birch Lake–Sounding Creek.............	1500	75
Southeastern Alberta....................	360	18
Saskatchewan..........................	720	36
	2580	129

Northern Plains of Alberta

This region includes the basin of low-dripping Cretaceous rocks from the latitude of Edmonton northward.   Outcrops are scarce, being practically confined to the banks of the principal rivers, and consequently the details of structure are not known.   Dips vary from 5 to 20 ft. per mile and no are asof steeper local structure have been found.   Thus far, most of the interest has centered around the tar sands of the Fort McMurray district.   At this point, basal Cretaceous sands, which directly overlie a petroliferous Devonian limestone, are saturated with tar for a maximum thickness of 200 ft. (61 m.) and a linear extent along the Athabasca River of about 50 mi. (80 km.).   This is the most extensive area of exposed tar sands in the world.   Prospect wells have been drilled to reach these sands away from the outcrop but the same quality of tar has been encountered in all wells drilled to date.   Wells directly south of McMurray do not get any tar in the sand, but at points 50 mi. southeast and 200 mi. directly west tar is obtained.   In between these points are large areas that have not been prospected, and in which productive fields may be found.   Conditions may be compared to those in Illinois and possibly Ohio.   The estimated reserve area is about the same as for the last mentioned region, that is, 2600 sq. mi. of possible territory and 130 sq. mi. of probable territory.

Rocky Mountain Front of Alberta and British Columbia

This region comprises a long strip along the edge of the Rocky Mountains, commonly called the disturbed belt of the Cretaceous, and varies in width from 5 to 20 miles.   The rocks are similar lithologically to the Cretaceous of Wyoming, but the structural conditions are different. There are no separate ranges of pre-Cretaceous rocks with intervening Cretaceous basins, around the edges of which dome structures are found, as in Wyoming. The Canadian mountain front is a continuous, nearly straight line, and the folds in the Cretaceous are almost exactly parallel to the mountain front.   The anticlines are long and narrow and

have steep beds along the crest, which is usually faulted. The Sheep Creek field, near Calgary, is one of the best appearing structures, but tests have proved only a very small production. The only oil seepages known along the mountain front are in the Sheep Creek field. In British Columbia, south of Peace River, apparently more favorable structures have been discovered, but they have not been thoroughly investigated because of their remoteness, and the unsympathetic attitude of the British Columbia government. The possible area is 60 sq. miles.

MACKENZIE RIVER DISTRICT

Much prominence has been given recently to the far north of western Canada because of the completion near Fort Norman of a flowing well. This well, which was drilled by the Imperial Oil Co., is said to have flowed at the rate of more than 1000 bbl. per day for periods of less than an hour. After only a small amount of oil had been produced, the capacity of the well greatly declined. The well is located on the flank of a large anticline where the dip is about 10°. The productive horizon is eroded from the crest of the fold and outcrops less than a mile from the well. Oil seepages are numerous over a large area and there is known to be a considerable area of mountains in which similar structures are developed. The oil in the well comes from a Devonian shale which is underlain by bituminous limestone. This district may be compared to West Virginia and part of Pennsylvania, and there are adjacent areas of flat-lying Paleozoics similar to those of Pennsylvania and Ohio. A possible reserve area for this region is about 1200 sq. mi. and a probable area is about 60 sq. mi. At the present time, this reserve is not commercially available but it may become commercially feasible to bring the oil from this district to the world's markets.

SUMMARY

	POSSIBLE AREA	PROBABLE AREA
Eastern Canada.....................	Probable future production 10,000,000 bbl.	
Southeastern and central plains of Alberta and S. W. Saskatchewan.....	2580 sq. mi.	129 sq. mi.
Northern plains of Alberta...........	2600 sq. mi.	130 sq. mi.
Rocky Mountain front of Alberta and British Columbia..................	60 sq. mi.	
Mackenzie River district.............	1200 sq. mi.	60 sq. mi.
	6440 sq. mi.	319 sq. mi.

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Present Conditions in Mexican Oil Fields and an Outlook Into the Future

BY VALENTIN R. GARFIAS,* NEW YORK, N. Y.

THE various phases of the Mexican oil industry have received so much publicity that there is little to add to the discussion of present and future production, extent and importance of prospective fields, and status of petroleum legislation, to mention but a few of the angles of the Mexican oil problem. However, out of this wealth of information, which is generally of contradictory nature, the average petroleum man is at a loss to know something definite regarding the various questions involved; so, keeping the foregoing in mind, it has been the writer's aim to record in a concise form the most reliable and up-to-date information regarding the salient features of the Mexican oil situation.

PRESENT STATUS OF MEXICAN OIL INDUSTRY

Production During 1921

The Mexican oil fields, during 1921, produced in round numbers 203,000,000 bbl. of which 176,000,000 bbl., or 86 per cent., were exported, the bulk of these exports, or about 73.3 per cent., going to the United States. The Mexican exports were divided by grades as follows: Light crude, 47.0 per cent.; heavy crude, 22.0 per cent.; tops, 3.5 per cent.; fuel oil from light crude, 24.0 per cent. The balance, or $3\frac{1}{2}$ per cent., includes the bunker fuel shipments and the coastwise trade in Mexico, which is not classified as to grades. The figures show that, in importance, the light-oil fields form about 75 per cent. of the Mexican petroleum industry.

The production of the Panuco field has followed the general history of the American fields, and its gradual decline, which is at present foreshadowed, may be estimated as well as its ultimate economic life. On the other hand, the behavior of the light-oil fields to the south is unlike any in the world so far discovered, and as a result we have concentrated overproductions followed by sudden droughts, both sometimes equally unexpected. The coming in of two or three gushers within adequate pipe-line facilities will flood the market with 100,000 bbl. or more a day, while the flooding by water of a particular pool will, in a few weeks, remove from the market one-half the total production of the Mexican fields.

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Total Production To Date

The production of the light-oil fields from their inception to January, 1922, has been estimated as follows:

	BARRELS
Tepetate and Casiano	135,000,000
Chinampa and Naranjos	155,000,000
Southern Amatlan	42,000,000
Zacamixtle	8,000,000
Toteco	21,000,000
Cerro Azul	75,000,000
Potrero	100,000,000
Alamo	37,000,000
Miscellaneous	1,000,000
	574,000,000

Present Production

The year 1921 saw the flooding of the southern fields, except Cerro Azul and Toteco, and the bulk of the light-oil production confined to this limited area; at the present time it is estimated that the daily production of these fields, including the oil that is being taken out of the flooded areas, is as follows:

	BARRELS
Tepetate-Upper Chinampa (stripping)	20,000
Naranjos, Amatlan, Zacamixtle (stripping)	100,000
Toteco-Cerro Azul	460,000
Alamo	10,000
Miscellaneous	20,000
	610,000

It will be seen that the present importance of these fields depends on the productivity of the Toteco-Cerro Azul area, which yields over two-thirds of the total.

Estimates of Future Production

It is therefore of the greatest importance to analyze the possibilities of the Toteco-Cerro Azul pools in view of their production to date and the possibilities of future yield, considering the time factor involved before a gradual decline is manifested. Figs. 1 and 2, prepared by R. V. Whetsel, represent two interpretations of the underground structure of these pools and give estimates of future yield based on the assumptions, first, that the Toteco pool is separated from Cerro Azul by a low saddle where oil has not accumulated; second, that the two fields really represent one pool separated only by a low flexure above the water horizon. The estimates show a minimum of 71,000,000 bbl., which, at the present rate of production, would show about 2 months' available supply before water appears, as explained in Fig. 2, and a maximum
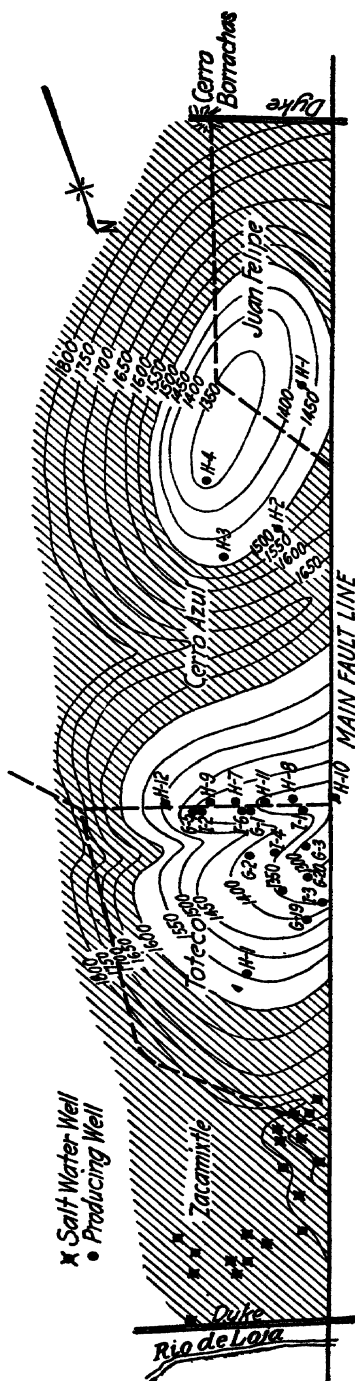
FIG. 1.—Assuming cutoff between North and South Cerro Azul. Contour lines are depth of oil horizon below sea level. Wells marked G, Mexican Gulf Co.; H, Huasteca; I, International.

Status of fields north of Toteco-Cerro Azul fields: Los Naranjos-Chinampa, flooded and producing 48,000 bbl. a day from stripping. Amatlan, flooded and producing 46,000 bbl. a day from stripping. Status of fields south of Toteco-Cerro Azul fields: Potrero-Cerro Viejo, salt water and stripping about 10,000 bbl. a day; Tierra Blanca, new field, drilling one well producing 10,000 bbl. a day; Alamo, salt water and stripping, about 11,000 bbl. a day.

Minimum estimate, 71,000,000 bbl. If assumed saddle between Toteco-North Cerro Azul exist, the South Cerro Azul pool may be ignored in estimating length of time flush product will last in southern fields as it is producing only at rate of 46,000 bbl. a day. Northern Toteco-Cerro Azul is flowing at rate of 450,000 bbl. a day; therefore, considering a reserve of 50,000,000 bbl. and subtracting 10,000,000 bbl. obtained from stripping, 40,000,000 bbl. must be flowed before salt water appears in higher wells, which at present rate of flow of 12,500,000 bbl. a month will be about May 1, 1922. (Water appeared on June 27.)
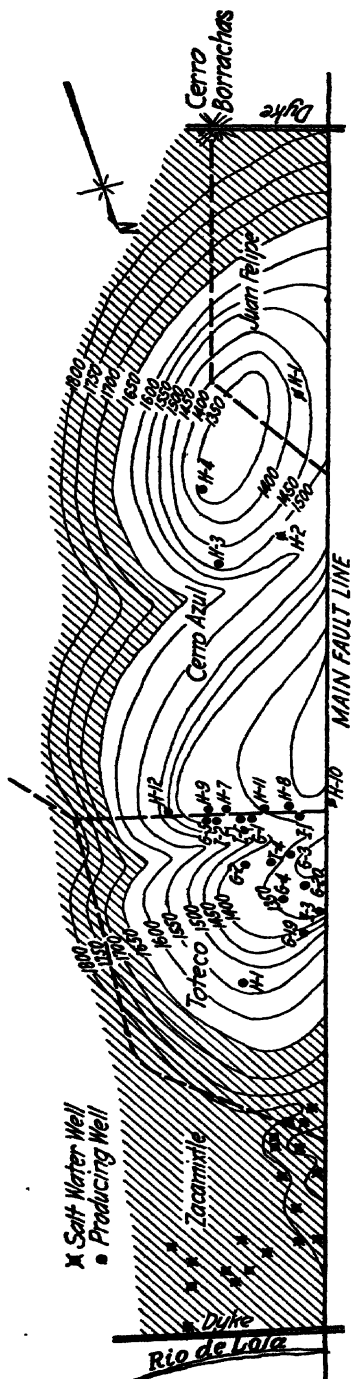
FIG. 2.—ASSUMING TOTECO AND CERRO AZUL AS ONE POOL. CONTOUR LINES ARE DEPTH OF OIL HORIZON BELOW SEA LEVEL. WELLS MARKED G, MEXICAN GULF CO.; H, HUASTECA; I, INTERNATIONAL.

Maximum estimate, 122,000,000 bbl. Subtracting 22,000,000 bbl. obtained from stripping leaves 100,000,000 bbl. at rate of 500,000 bbl. a day or 200 days before salt water appears in highest wells, or July, 1922. Mean estimate, about 95,000,000 bbl. remaining in Toteco-Cerro Azul pool; subtracting 20,000,000 bbl. for stripping leaves 75,000,000 bbl. at rate of 500,000 bbl. a day, or June, 1922, before salt water appears in highest wells. (Water appeared on June 27.)

of 122,000,000 bbl., giving approximately 5 months' supply before the encroachment of salt water; a mean estimate of, say, 100,000,000 bbl. would give these fields an economic life of about 3 months prior to the intrusion of the water.

It is needless to state that estimates of this nature are based, of necessity, on many conjectural points and represent at best more or less justified estimates of the future behavior of the fields in view of past performances of similar pools in Mexico; it is also needless to say that these estimates may prove incorrect because of factors that are not taken into account. For this reason errors of over or under estimation not only are to be expected but very likely have been incurred in the figures presented.

Assuming, however, that the estimates reflect actual conditions, we should expect, before the middle of 1922, a sharp decline in the production of the Toteco and Cerro Azul pools, which at present yield the bulk of the Mexican production.

EXTENSIONS OF PRODUCING FIELDS

The Knife Edge

The fields producing light oil, as is well known, are located along the narrow strip running from Dos Bocas, on the Tamiahua lagoon, to Alamo, on the Tuxpan river, and possibly to Furbero 30 mi. south, thus forming a slightly curved belt about 1 mi. wide and 76 mi. long. Excluding the present producing areas, there are about 30 mi. that have been drained of oil and there are about 30 mi. between Alamo and Furbero where the possibilities of extensions are questionable; 7 mi. between Alamo and Potrero, controlled by two companies, which have obtained production although its real importance remains to be proved; about 4 mi. between Potrero and Cerro Azul, likewise controlled by two companies, where the possibilities of encountering oil are not considered very promising; and an extension of 5 mi. between Tepetate and Dos Bocas, in which important development is going on at the present time and where there are strong probabilities of encountering oil although probably of a heavier grade (14° B.) than the typical product of the region. (Wells drilled after this paper was presented have shown that the writer was too optimistic regarding this region.)

Although it is admitted that there are possibilities of encountering oil along the entire belt between Furbero and Dos Bocas, in considering the availability of this supply for the purpose of counteracting the expected sharp decline of the present producing fields about the middle of this year, we can only consider the region between Dos Bocas and Tepetate, which so far has shown a heavier oil than to the south, and the region between Alamo and Cerro Viejo, in which producing wells have been encountered, although reports about this development have been of a contradictory nature.

Wildcat Wells

In the wildcat fields, outside of the so-called knife edge the following wells are being drilled at present:

	No. of Wells	Depth, Feet		No. of Wells	Depth, Feet
Chocoy	1	2560	Palo Blanco	2	Rig
Tamimas (Tempoal Valley)	1	2753	Jardin Cicuque	1	Rig
Comales	1	2400	El Dorado	1	1200
Aguada	3	2600	Castillo Teayo	1	Rig
Tamantao	1	3418	San Javier	1	1180
Idolo Island	1	2433	Agua Nacida	2	Rig
Palma Sola	1	1847	Chinton (Panuco)	1	Rig
Tlacolula	1	1490	Chila Cortaza	1	Rig
San Isidro	1	2390	Santiago de la Peña	1	Rig
Alamo No. 12	1	2615	Salinas (Abasolo)	1	40

The twenty-four wells that are being drilled with a view of locating entirely new productive regions are not near pipe-line facilities and even if some of them should prove successful their production will not compensate for the expected decline in the present fields.

Of late, considerable publicity has been given the results of drilling in the Tehuantepec region and reports of a highly productive flowing well have been circulated. However, no definite information has been obtained regarding this development other than that the well has been ruined and its production become unavailable. Although it is probable that the Tehuantepec region will again be commercially productive, the record of its wells and the small amount of prospecting being carried on lead us to conclude that the production of this region will not increase soon enough to counteract the expected declines during the present year.

Summarizing the foregoing, we may state that there is a possibility, should an important producing field be developed between Tepetate and Dos Bocas in the next few months, that the intensive development of this area will, to some extent, make up for the decline of the Toteco-Cerro Azul pools and so maintain the Mexican production near its present high level; and that the successful intensive development of the region between Alamo and Potrero may serve the same purpose, but that in this case, owing to the large size of the holdings of two companies, the production would be more intelligently and economically handled and very likely the total exports from the Mexican fields would decline considerably below its present high rate.

CONTROLLING INFLUENCE OF THE MEXICAN OIL INDUSTRY

American and Mexican Petroleum Statistics

The petroleum statistics for the last 3 years are shown in the accompanying table, which gives the official figures of production, imports,

exports, stocks in Mexico and the United States, and oil prices in this country. The figures were compiled with the object of showing to what degree the American petroleum industry is dependent on Mexican production and to what extent, also, the security of the world's petroleum supply hinges on the productivity of the American and Mexican fields, which combined represent over 90 per cent. of the world's total.

An analysis of the table shows that the production of the United States during the last 3 years has gradually increased, although during the latter part of 1921 it was lower than during the same period for the previous years; and that the imports into the United States during the last 3 years have materially increased, with the exception of three months in 1921, when they were artificially curtailed. The exports from the United States have increased somewhat, although the over-all figures have not materially changed in the last 3 years, the monthly exports aggregating between 5,000,000 and 6,000,000 bbl. The stocks have increased monthly during the last 3 years, with the exception of ten months; in 1919 the net movements of stocks showed only a total increase of about 3,000,000 bbl., in 1920 an increase of 31,000,000 bbl., and in 1921 an increase of 66,000,000 bbl. There was a decrease in stocks during three months of 1921. On the other hand, domestic consumption has almost doubled during the last 3 years. In January, 1919, the consumption was about 24,000,000 bbl., in December, 1921, 45,000,000 bbl.; and the total demand represented by domestic consumption plus exports increased from 30,000,000 to 50,000,000 bbl. during the same period.

The Storage-consumption Coefficient

In trying to visualize the American industry as a whole and to analyze its present position, as compared with previous years, the writer has been confronted by the usual difficulties in interpreting statistics. If one studies production records exclusively, the conclusion might be drawn that there is too much oil available, which conclusion would be further strengthened if the figures of imports are taken into account. On the other hand, the figures of consumption at home and abroad show that there has been an increased demand for American and Mexican oils and an exclusive analysis of these records would force over optimistic conclusions. It is evident, therefore, that to ascertain whether the relative position of the industry was more sound at the end of 1921 than in previous years a monthly comparison of the total output and input of oil and of the safety factor represented by storage is necessary.

With the foregoing in mind, the writer has tried to analyze this relation and has endeavored to estimate the factor of safety represented by the oil above ground or in storage, the available supply above ground so

to speak, in terms of months of consumption. This factor, which is dependent on the consumption and storage and is represented by the ratio of the total domestic consumption plus exports and the corresponding total stocks of crude and refined oil for any one month, has for convenience been called the *storage-consumption coefficient*, or the months' supply in storage. It represents the oil on hand, which is the factor of safety on which the stability of the American oil industry depends. This monthly coefficient has been computed for the last 3 years and the results plotted are worth noting.

During 1917 and part of 1918, there was in storage enough oil to supply the demands of the industry for about 6 months. From November, 1918, to April, 1919, the factor of safety increased; in other words, the available storage could take care of the total output for 7 months without additional production. From April to December there was a gradual decline and by the end of 1919 there was in storage only enough oil to take care of 4½ months of supply, which factor was kept practically unchanged to January, 1921. The conditions of supply and storage during the past year have been erratic, with a sharp increase during February and fairly uniform for the balance of the year; there was in storage in December, 1921, enough oil to meet the demands of about 6 months at the rate of consumption and exports prevailing in December.

The fluctuations of this storage-consumption coefficient from August, 1917, to the end of 1921 show that there was on an average close to 5½ months of available supply in storage, the fluctuations ranging between 7½ months for February, 1921, to 4 months in August, 1920. At no time during the past 4½ years has there been too much safety nor too much oil in sight if the magnitude of the American oil industry and its preponderance over the world's supply is taken into account. It is also evident that conditions during 1921 were by no means exceptional; in fact, when the increased consumption and needs for oil are taken into account, the additional home production and foreign imports have not more than met the increased demands and maintained the corresponding average factor of safety represented by oil in storage.

It should be further borne in mind that 1921 was a year of business depression and that even under these unfavorable conditions there was enough demand to absorb the increased production and maintain the factor of safety of the industry within average figures.

Relation between Storage-consumption Coefficient and Price of Oil

It is interesting to compare this storage-consumption coefficient with the average oil price for the same period; therefore the monthly average of posted prices for Mid-Continent, California, and Gulf Coast crudes

were plotted against the oil-supply-in-store figures.   Fig. 3 shows that the average oil price was fairly constant during 1917, 1918, and part of 1919, but at the end of 1919 there was a gradual increase in prices corresponding to the decrease of safety afforded by the reduced stocks, and by April, 1920, the average oil price had reached $2.73 per bbl., which price prevailed with little fluctuation until October. Coincident with the sudden increase of oil in storage, as compared with consumption at the beginning of 1921, there was a rapid drop in the price of oil which continued to August, when the average price of oil was $1, which is the lowest figure recorded during the last 5 years. The readjustment of the storage-consumption coefficient since February, 1921, was not reflected in the average oil prices until November of that year, when the average price was raised to $1.43 per barrel.

The curves show the close relationship between the storage-consumption coefficient and the oil price; in fact, they show that the prices of oil depend, to a great extent, on the stability of the industry, which in turn is represented by the ratio between the oil in sight—the available supply represented by the oil in storage above ground—and the total consumption for the corresponding period. They also show that in comparison to the previous years, the sharp decrease in price during 1921 was not justified, or at least that the radical drop to an average price of $1 was not warranted in view of the comparative low safety afforded by the proportionately small amount of oil in storage; also that the present oil prices if anything, are low in comparison to the storage-consumption coefficient.

Controlling Importance of Mexican Production

In Table 1, columns H, I, and J show the dependence of the American oil industry on the importation of Mexican oil. Column H shows the difference between American consumption plus exports and domestic production, in other words, the amount of oil that is needed
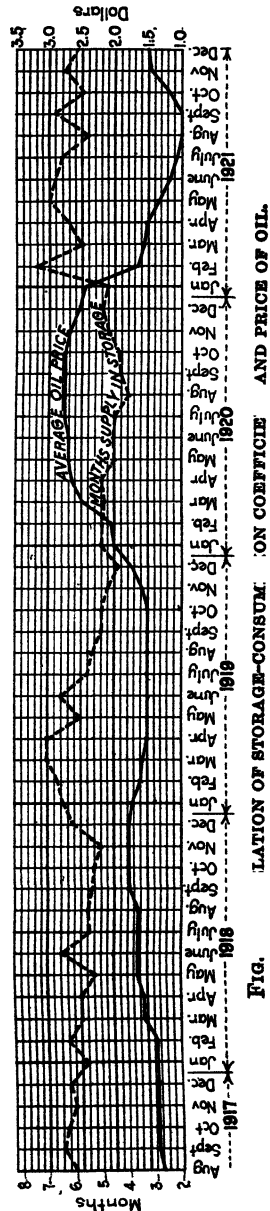
TABLE 1.—*Petroleum Statistics of Last Three Years*

In Millions of Barrels

	1	2	3	4	5	6	7	8	9
	U. S. A. Production, U. S. G. S.	U. S. A. Imports, All Products, U. S. Dept. Commerce	Decrease in Stocks, All Products	Total Income, All Products 1+2+3	U. S. A. Exports, All Products, U. S. Dept. Commerce	Increase in Stocks, All Products	Total Out Going 5+6	Domestic Consumption 4−7	Domestic Consumption Plus Exports 5+8
1919									
January......	30.20	4.03		34.23	5.67	3.80	9.46	24.77	30.43
February....	26.91	3.87		30.77	4.07	1.30	5.38	25.40	29.47
March.	30.23	3.55		33.79	3.92	5.41	9 33	24 46	28.38
April........	29.39	3.98		33.37	5.31	4.08	9.40	23.97	29.28
May........	29.99	4.86	0.39	35.23	4.47		4.47	30.76	35.24
June.	31.64	4.78		36.42	6.17	4.48	10.65	25.78	31.94
July........	33.89	4.63		38.53	4.47	0.36	4.83	33.70	38.17
August	33.86	4.14	1.34	39.35	4.56		4.56	34.79	39.35
September...	33.67	4.49	1.89	40.05	5.10		5.10	34.94	40.05
October......	33 32	6.12	2.20	41.64	6.65		6.65	34.99	41.64
November....	32.11	5.19	4.94	42.24	6.31		6.30	35.93	42.24
December....	32.51	4.52	6.22	43.24	6.16		6.16	37.08	43.24
Total......	377.72	54.16	16.98	448.86	62.86	19.43	82.29	366.57	429.43
1920									
January......	33.77	6.37		40.14	5.81	1.40	7.22	32.92	38.74
February. ...	32.73	5.19	1.14	39.05	5.68		5.68	33.37	39.05
March.......	35.83	6.54		42.37	6.90	1.75	8.65	33.72	40.62
April........	35.58	6.44		42.03	6.42	3.60	10.02	32.01	38.42
May........	36.50	7.07		43.57	6.60	0.11	6.71	36.86	43.46
June........	36.95	8.56		45.51	6.40	2.27	8.67	36.84	43.24
July........	38.20	6.85		45.06	6.88	0.26	7.14	37.92	44.80
August......	39.06	11.01		50.07	6.31	1.34	7.65	42.42	48.73
September..	37.53	11.96		49.49	5.55	2.33	7.87	41.62	47 16
October......	39.59	11.51		51.10	7.53	3.81	11 34	39.76	47.29
November....	38.70	14.14		52.83	6.11	7.87	13.98	38.85	44.97
December....	38.96	13.12		52.08	7.92	7.26	15.18	36.90	44.82
Total......	443.40	108.76	1.14	553.30	78.11	32.00	110.11	443.19	521.30
1921									
January.....	37.85	13.55		51.40	7.43	4.34	11.77	39.63	47.06
February....	35.35	11.64		46.99	6.13	15.04	21.17	25.82	31.95
March.......	40.97	12 70		53.67	5.65	9.31	14.96	38.71	44.36
April........	40.06	10.21		50.27	5.60	8.15	13.75	36.52	42.12
May........	42.04	9.38		51.42	4.67	11.94	16.61	34.81	39.48
June........	40.41	10.48		50.89	4.87	8.81	13.68	37.21	42.08
July........	40.33	8.37		48.70	4.36	4.97	9.33	39.37	43.73
August	40.97	8.61	7.17	51.75	5.64		5.64	46.11	51.75
September...	36.62	9.35	2.12	48.09	5.60		5.60	42.49	48.09
October......	35.62	11.71	0.78	48.11	6.04		6.04	42.07	48.11
November....	37.78	13.43		51.21	5.40	6.66	12.06	39.20	44.60
December....	41.96	14.31		56.27	5.46	7.10	12.56	43.71	49.17
Total......	469.96	128.74	10.07	608.77	66.85	76.82	143 17	465.65	532.50

TABLE 1.—*Petroleum Statistics of Last Three Years.—Continued*

Stocks

	A		B	C	D	E	F	G
	Crude in Fields		Crude at Refineries, U. S. B. M.	Mexican Crude in U. S. A., U. S. G. S.	Total Crude $A+B+C$	Refined Products, U. S. B.M.	Grand Total $D+E$	Storage-consumption Coefficient $F \div 9$
	Net, U. S. G. S.	Gross, U. S. G. S.						
1919								
January......		127.78	16.47		144.25	49.09	193.34	6.4
February.....		126.98	16.00		142.98	51.66	194.64	6.6
March..		129.21	16.23		145.44	54.61	200.05	7.0
April........		130.73	16.17		146.89	57.24	204.13	7.0
May........		130.32	17.15		147.47	56.27	203.74	5.8
June........		134.00	18.05		152.04	56.18	208.22	6.5
July........		140.09	16.34		156.43	52.15	208.58	5.5
August.......		136.47	16.06		152.53	54.70	207.24	5.3
September....		137.13	15.10		152.23	53.11	205.34	5.1
October.....		135.46	16.06		151.52	51.62	203.14	4.9
November...		131.60	15.10		146.70	51.50	198.20	4.7
December....		127.87	14.28		142.16	49.83	191.98	4.4
1920								
January......		127.16	14.17	2.57	143.90	49 49	193.39	5.0
February.....		126.34	14.36	2.38	143.08	49.17	192.25	4.9
March... ...		125.60	15.15	2.16	142.90	51.05	194.00	4.8
April.......		125.00	15.90	2.29	143.18	54.42	197.60	5.1
May........		124.69	16.17	2.50	143.36	54.35	197.71	4.5
June		126.76	17.20	3.16	147.12	52.86	199.98	4.6
July.........		128.17	17.99	2.65	148.81	51.43	200.24	4.5
August.......		129.04	18.98	3.94	151.97	49.61	201.58	4.1
September ...		128 79	19.90	4.19	152.87	51.03	203.90	4.3
October......		129.45	20.38	5.55	155.38	52.33	207.71	4.4
November....		131.33	22.39	6.60	160.32	55.19	215.51	4.8
December....		133.69	22.38	7.44	163.51	59.26	222.77	5.0
1921								
January......	117.10	130.32	22.36	8.13	160.81	66.30	227.11	4.8
February.....	121.71	135.09	23.83	10.75	169.67	72.48	242.15	7.6
March.......	128.75	142.08	22.69	11.97	176.74	74.72	251.46	5.7
April........	134.72	147.86	22.42	10.30	180.58	79.03	259.61	6.2
May........	143.61	156.56	21.93	10.20	188.69	82.86	271.55	6.9
June........	151.34	164.79	21.81	9.71	196.31	84.05	280.36	6.7
July.........	159.03	172.49	22.80	8.32	203.61	81.72	285.33	6.5
August.......	162.81	176.35	19.98	5.29	201.62	76.54	278.56	5.4
September....	164.08	177.38	17.25	7.29	201.92	74.12	276.04	6.7
October.......	163.33	176.25	18.83	9.27	204.35	70.91	275.26	5.7
November....	163.81	176.44	19.54	12.20	208.18	73.74	281.92	6.4
December....	170.35	182.48	19.00*	13.54	215.02	74.09*	289.02	5.9

* Estimated.

TABLE 1.—*Petroleum Statistics of Last Three Years.*—*Continued*

	H	I	J	K	L	M	N
					Oil Prices		
	9−1	Mexican Imports into U. S. A., Dept. Commerce	Difference between H & I	Mid. Cont.	Calif. 21° B.	Gulf Coast	Average
1919							
January.........	0.23	3.89	3.67	$2.25	$1.29	$1.62	$1.72
February........	2.56	3.66	1.09	2.25	1.29	1.25	1.59
March..........	R 1.15	3.49	5.34	2.25	1.29	1.15	1.56
April...........	R 0.09	3.97	4.07	2.25	1.29	1.00	1.51
May............	5.25	4.74	R 0.51	2.25	1.29	1.00	1.51
June.	0.30	4.70	4.40	2.25	1.29	1.00	1.51
July............	4.27	4.40	0.22	2.25	1.29	1.00	1.51
August.........	5.49	4.14	R 1.34	2.25	1.29	1.00	1.51
September	6.38	4.44	R 1.94	2.25	1.29	1.00	1.51
October........	8.32	5.89	R 2.43	2.25	1.29	1.00	1.51
November.	10.12	4.94	R 5.19	2.33	1.29	1.00	1.54
December	10.73	4.35	R 6.39	2.58	1.29	1.31	1.73
Total.........		52.74					
1920							
January.........	4.97	6.29	1.32	2.97	1.29	1.70	1.99
February........	6.33	4.94	R 1.39	3.00	1.33	2.84	2.06
March..........	4.79	6.50	1.71	3.50	1.33	2.50	2.44
April...........	2.84	6.19	3.35	3.50	1.58	2.80	2.63
May............	6.96	6.96		3.50	1.58	3.00	2.69
June............	6.29	8.12	1.83	3.50	1.58	3.00	2.69
July............	6.59	6.77	0.17	3.50	1.70	3.00	2.73
August.........	9.67	10.72	1.07	3.50	1.70	3.00	2.73
September.......	9.63	11.65	2.02	3.50	1.70	3.00	2.73
October........	7.70	11.36	3.67	3.50	1.70	3.00	2.73
November.......	6.27	13.75	7.48	3.50	1.70	2.86	2.69
December.......	5.86	12.84	6.98	3.50	1.70	2.50	2.56
Total.........	77.90	106.11					
1921							
January.........	9.21	13.19	3.98	3.40	1.70	2.32	2.47
February........	R 3.40	11.38	14.78	1.87	1.70	1.34	1.64
March..........	3.39	12.30	8.91	1.75	1.70	1.25	1.57
April..........	2.06	10.04	7.98	1.75	1.70	1.10	1.52
May............	R 2.56	9.15	11.71	1.50	1.45	1.00	1.32
June............	1.67	10.21	8.54	1.22	1.45	0.89	1.19
July............	3.40	8.05	4.65	1.00	1.45	0.80	1.08
August..........	10.78	8.35	R 7.43	1.00	1.20	0.80	1.00
September.......	11.47	9.09	R 2.38	1.00	1.20	0.80	1.00
October........	12.49	11.58	R 0.91	1.42	1.20	0.95	1.19
November.......	6.82	12.99	6.17	1.90	1.20	1.18	1.43
December.......	7.21	14.50	7.29	2.00	1.20	1.25	1.48
Total.........		125.83					

from foreign fields to break even and keep the American industry going. Only during the months of March and April, 1919, was more oil produced in the United States than was actually utilized and exported. The deficiency has been gradually increasing from 1,000,000 or 2,000,000 bbl. at the beginning of 1917 to 7,210,000 bbl. at the end of 1921. Column I shows Mexican imports into the United States, and column J shows the excess of imports over deficiency; it further indicates that during 10 months in the last 3 years, even with the Mexican imports, there was not enough oil available to supply American consumption and exports; this was the case during August, September, and October, 1921. An analysis of these figures will demonstrate the absolute dependence of the American oil industry on Mexican oil imports, which are needed not only to take care of foreign business but to meet the domestic needs of the United States.

MEXICAN OIL TAXES

The Ad Valorem Tax

In a paper presented to the Institute a year ago, the writer recommended that Mexican oil taxes be figured on a more reasonable basis, and that volume rather than weight relations be employed; that the mooted question of the value of Mexican oils in Mexico as a basis for taxation be substituted by a relation to the average oil prices in the United States; and that other minor changes be made to conform as much as possible with the current American practice. As a result of this work and in accordance with independent studies made by the Mexican Government, the ad valorem tax in operation a year ago was abrogated and in its place a tax was put into effect as per decree of May 24, in which practically all the points above noted were incorporated. This tax has since been collected in full by the Mexican Government from month to month, and its amount has fluctuated with the average American oil prices.

Infalsificable Tax

The paper redemption tax, or Infalsificable, was abrogated and has not been collected since Aug. 1, 1921.

The Export Tax

For the special purpose of taking care of the foreign debt, according to arrangements to be made at a later date, the Mexican Government instituted as per decree of June 7, 1921, a fixed tax, which was created by additions to the export tariffs of the country; this tax went into effect on July 1 and is in effect today.

In September, a committee representing the Association of American Producers in Mexico, headed by Walter Teagle, held a series of conferences in Mexico City with Secretary of the Treasury de la Huerta with a view of revamping this export tax. Although the results of these conferences have not been made public, enough has been given to the press to warrant, with a degree of assurance, the statement that the general bases of the agreement are as follows:

1. The Mexican Government advised the Committee that it was willing to receive in payment for the export tax Mexican bonds at their face value; in other words, the Government would receive $1000 Mexican Bond for $1000 worth of export taxes as per decree of June 7.

2. The Committee agreed to negotiate with the committee of bankers, headed by Thomas Lamont, with the object of obtaining its coöperation and the necessary bonds for carrying out the scheme; in other words, the Committee was to reach an agreement with the bankers, before being able to pay the tax with bonds.

3. Dec. 25, 1921, was set as the time limit for the Committee to complete the arrangements, it being understood that if by that time no agreement was reached with the bankers, the Mexican Government was to receive in cash 40 per cent. of the taxes due from July 1 to Nov. 30, inclusive, representing the average cash price at which Mexican bonds might be procured within those dates.

4. No agreement was reached prior to Dec. 25 between the Teagle and Lamont Committees so the Mexican Government extended the time for an indefinite period, with the understanding that the term could expire with a 10 days' notice from the Mexican Government. It is further understood that no agreement having been reached between the Committees, the Mexican Government a short time ago notified the Teagle Committee that the taxes from July 1 to Dec. 1, 1921, inclusive, would be paid in cash at 40 per cent. of their face value and that the taxes from Dec. 1 on would be paid either full value in cash, or in bonds which could be furnished by the Mexican Government in case the Teagle Committee is unable to procure bonds from the committee of bankers.

5. The Mexican Government has repeatedly made the statement that the money levied from this tax would not be used for any other purpose than reducing the amount of its foreign obligations and that the money so collected would be turned over to some bank which would handle said funds for the specified purpose.

SUMMARY

The information herein recorded may be summarized as follows:

1. Production of the Panuco field will remain stationary or gradually decline during 1922. Production from the fields yielding light oil should

show a sharp decline during the year, beginning sometime during the summer.

2. The decline of Mexican production may be partly neutralized if the development work now being started between Tepetate and Dos Bocas proves successful; also, if the result of an intensive drilling campaign between Alamo and Potrero meets the optimistic view held by some operators.

3. Development of wildcat drilling in northeastern Mexico and the Isthmus of Tehuantepec, although of great importance as to the future supply, should not offset the sharp decline anticipated during 1922. Wildcatting is not being carried on in Mexico as intensively as the importance of the industry warrants.

4. Notwithstanding all that has been written, the conditions of the American oil industry did not warrant the sharp decline in prices during the first half of 1921 and present conditions, as regards factor of safety and stability, do not warrant any further curtailment in prices; in fact, the American oil industry, as regards supply and demand, is now under normal conditions and the oil prices should correspond to such conditions.

5. The absolute dependence of the American oil industry, in fact the world's oil industry, on Mexican production, is self-evident and a disastrous effect in the industry from a rapid curtailment of Mexican production will be the logical consequence. It follows that the intelligent development of the Mexican oil fields is a first necessity to the stability of the American oil industry.

6. Mexican taxation is now levied on a reasonable basis, which will do much toward stabilizing the world's petroleum industry, which is absolutely dependent on Mexican and American production. The new, or export, tax levied for the sole purpose of reducing its foreign obligations, has been in effect since July 1, 1921, and must be paid by exporters of Mexican oil either in cash or in Mexican bonds.

7. Perhaps the outstanding feature of the Mexican operations during the last year has been the friendly undertone that has prevailed in all matters in which American operators and the Mexican officials have come in contact, this being due primarily to the radical departure from the old practice prevailing for many years when both the Mexican Government and American operators were dealing through intermediaries, mostly attorneys, whereas all dealings during the last year have been transacted between Secretary de la Huerta of the Obregon Cabinet and the presidents of five of the leading American oil companies.

Petroleum Reserves of Central America*

By ARTHUR H. REDFIELD, WASHINGTON, D. C.

(New York Meeting, February, 1922)

IN ESTIMATING the unmined petroleum reserves of Central America, it is not feasible to employ the methods that have been worked out in the oil fields of the United States. No producing wells have been brought in and no drilling sections have been made public. It is accordingly necessary to fall back on a crude "barrels per square mile" estimate. The method chosen is a comparison of the structure of the foreign field of unknown reserves with that of some supposedly analogous North American field of which the reserves have been estimated. This method makes no pretense at scientific accuracy, but furnishes a working basis for an estimate.

The areal geology of Central America is illustrated in the map prepared by Sapper and more recently in the geologic map of North America accompanying U. S. Geological Survey *Professional Paper* No. 71, "Index to the stratigraphy of North America." Disregarding for the present the narrow Quaternary coastal plains and alluvial deposits of the river valleys, three chief zones may be distinguished. These are: A zone of late eruptive and effusive rocks; a zone of highly folded pre-Cambrian(?) or early Paleozoic crystalline schists and slates, considerably intruded by pre-Tertiary plutonic rocks; and a zone of more or less folded sediments, chiefly of Cretaceous or Tertiary age.

The zone of late eruptive and effusive rocks of Central America begins in southeastern Chiapas, Mexico, and extends, with increasing breadth, across southern Guatemala, across practically the whole of El Salvador, and across southern Honduras to the valley of Rio Goascorán, where the igneous zone attains a width of about 100 mi. (160 km.) East of the Goascorán Valley late eruptives and effusives play a subordinate part, being confined largely to the Departments of Valle and Choluteca. Dikes and sills of igneous rocks intrude the sedimentary formations of central and northern Honduras.

Late igneous rocks cover about four-fifths of the area of Nicaragua. In Costa Rica, late igneous rocks form the diagonal ranges of the Sierra de Guanacaste and the Cordillera Central, and play a part in the upbuild-

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ing of the Sierra de Talamanca. Eruptive and effusive rocks form the backbone of the Isthmus of Panama from the Costa Rican frontier to the Colombian border.

The crystalline complex attains its greatest development in central Guatemala and northern and central Honduras. In central Guatemala, the crystalline complex forms three parallel mountain arcs, separated by structural valleys filled with Tertiary and Quaternary sediments. One of these arcs forms the boundary range between Guatemala and Honduras and is continued in the Bay Islands off the coast of Honduras. East of the transverse Ulua Valley, which is apparently a structural depression, the crystalline complex extends in a series of mountain arcs of flatter curvature than those of central Guatemala. In Costa Rica and Panama, old crystalline schists do not appear. Pre-Tertiary intrusives form, however, the core of the Sierra de Talamanca. This range is continued in Panama. Exposures of plutonic rocks occur at intervals in the complex of late eruptives and effusives that form the backbone of the isthmus.

The sedimentary zone is greater on the Caribbean side of Central America than on the Pacific side. It comprises formations ranging in age from the Carboniferous to the recent. The greater part of the sediments of northern Central America, however, are not older than the Cretaceous; and in southern Central America no deposits older than the Tertiary are known. An arc of Carboniferous sediments, largely limestones, extends in central Guatemala from western Huehuetenango to Rio Cobán. A horst of Lower Carboniferous sediments, intruded by granites and quartz porphyries, occupies the central portion of British Honduras and extends into the Department of El Petén, Guatemala. These are the only known occurrences of unaltered Paleozoic sediments in Central America.

Limited outcrops of Triassic sediments occur, notably the Todos Santos strata of Huehuetenango, Guatemala, and the Tegucigalpa strata of Honduras. Generally, however, no Mesozoic strata older than the Lower Cretaceous are exposed in Central America.

In Guatemala, Lower and Middle Cretaceous limestones and dolomites occupy the greater part of the Departments of Huehuetenango, El Quiché, Alta Verapaz, and Izabal, as well as important areas in the center and southeast of El Petén. From southeastern Petén the Cretaceous zone continues into southern British Honduras. In the north and center of Spanish Honduras, highly folded Cretaceous sediments form valleys and ranges between the mountain arcs of the crystalline complex. The occurrence of Cretaceous sediments in the southern half of Central America is open to doubt. In Costa Rica, the San Miguel limestone, referred by Hill to the Cretaceous, is considered by Romanes to be no older than Tertiary. In Panama, no sediments older than the Eocene have been determined.

The largest area of uninterrupted Tertiary sediments occurs in the Department of El Petén, Guatemala, and in the northern third of British Honduras. Outcrops of Tertiary sediments do not occur in El Salvador; in Spanish Honduras and in Nicaragua they are isolated and of limited extent. The second largest area of Tertiary sediments extends with varying breadth in southern Central America, from Rio San Juan, Costa Rica, to the Colombian border of Panama. On the Pacific coast of southern Central America, Tertiary sediments occupy the narrow neck of land between Lake Nicaragua and the Pacific Ocean, and form the coastal ranges and cordilleran foot hills from the Gulf of Nicoya to Mala Bay and continue beyond Rio Diquís to the Panama frontier. They continue in Panama along the Pacific coast of the isthmus and attain their greatest breadth on the Azuero Peninsula and southeast of San Miguel Bay.

<center>TECTONICS</center>

All of the sedimentary formations of Central America older than the Miocene have evidently undergone considerable dislocation by folds and flexures, faults, and combinations thereof. The older rocks generally show stronger folding and more marked dislocation. The highest degree of diastrophism exists in the crystalline schists, many of which probably represent altered Paleozoic sediments. In contrast, the Pliocene and Quaternary deposits of Central America are generally horizontal, or show only slight, very flat folds and elevations of the strata. On the other hand, many faults with vertical displacement occur.

The present disturbed state of the formations of Central America gives evidence of four principal periods of diastrophism: At or before the close of the pre-Cambrian; at the close of the Paleozoic; late in or at the close of the Miocene; and late in or after the Pliocene. The pre-Cambrian (?) and Paleozoic folding developed the cordilleran axes of central Guatemala and central and northern Honduras. The Miocene movements, less intense than the Paleozoic, developed the mountain ranges in the Mesozoic beds of central Guatemala and central Honduras, and initiated the cycle of erosion in which the dissection of the present mountains was accomplished. The late Pliocene or post-Pliocene movements, which were evidently both vertical and tangential, folded and faulted the late Tertiary sediments from Yucatan southward through the Caribbean coastal plain of Honduras.

As a result of these movements the tectonic lines of Central America have a characteristic east-west trend to which Sapper gives the name of "Central American strike." This trend is not everywhere locally apparent, as the lines of strike form two principal series of arcs, which convex toward the south. These are represented in the physiography of Central America by two arc-shaped systems of mountains, the arcs of

which meet at a sharp angle somewhere under the igneous flows of Nicaragua.

The northern, or Guatemalan, system, to employ Sapper's terminology, has greater length and breadth than the southern, or Costa Rican, system. It consists of a series of pre-Cambrian (?) or early Paleozoic and Carboniferous mountain ranges, extending in a flat arc, which is convex toward the south from Chiapas across central Guatemala and western Honduras to the Humuya-Ulua Valley, which interrupts the trend of the ranges and appears to be a structural depression, perhaps a graben between parallel faults transverse to the mountain arcs. East of this so-called Honduras Depression, the pre-Cambrian(?) and Paleozoic ranges continue, but in a much flatter arc. The trend of these older sediments, with their accompanying intrusive rocks, is closely paralleled by the Cretaceous sediments of central Guatemala and central Honduras and by the Tertiary sediments of El Petén and northern British Honduras.

The southern, or Costa Rican, system consists principally of two mountain arcs of parallel curvature, with short parallel neighboring chains. Its trend is parallel to that of the isthmus. These mountains are believed by Sapper to have resulted from the Miocene movements. In the geologic map of North America accompanying U. S. Geological Survey Professional Paper No. 71, however, the Sierra de Talamanca of southern Costa Rica is mapped as pre-Tertiary.

The intersection of the two mountain systems of Central America in Nicaragua is not apparent. The greater part of the surface of this republic is covered by late eruptives and tuffs, which effectively conceal the underlying structure.

The Miocene and Pliocene folding movements, to which the present structure of the Cretaceous and Tertiary sediments are due, were in general parallel to the preceding tectonic movements. The folded Cretaceous and Tertiary sediments of central and northern Guatemala, central Honduras, and British Honduras show the same curved lines of strike, convex to the south, as do the older mountain ranges with which they are associated. The Tertiary sediments which flank the cordilleras of Costa Rica and Panama trend parallel to the S-shaped curve of the isthmus.

VULCANISM

The four principal periods of diastrophism in Central America may have been also the chief periods of vulcanism. Volcanic action in the present central portion of Central America undoubtedly began before the Miocene deformation, as the earlier volcanic rocks are steeply folded. Pre-Tertiary plutonic rocks cut the crystalline complex in northern Central America and similar plutonic rocks form some of the sierras of eastern Honduras. In the period of volcanic activity that began in the

late Miocene large areas of Guatemala, Honduras, Salvador, and Nicaragua were covered with igneous flows and tuff deposits, the Cretaceous sediments of northern and central Honduras were intruded, and the cordilleras of northern Costa Rica and of Panama were built up. The late Pliocene or post-Pliocene movements were accompanied by vulcanism, which resulted in a series of volcanoes along the Pacific coast, some of which are dormant today.

The alignment and order of the late volcanoes that border the Pacific coast of northern Central America, indicate the existence of lines of dislocation of a special character, unrelated to the older mountain arcs. The most important volcanoes of northern Central America (Guatemala to Nicaragua) are arranged in a few converging lines, which trend generally west-northwest by east-southeast. These longitudinal lines of vulcanism are intersected by several short, transverse lines. The main lines appear to indicate the course of fault lines that arose, as suggested by Powers, from the subsidence of the Pacific Ocean basin.

In southern Central America, the majority of the volcanic cones occur along the crest of the north Costa Rican mountain arc. A single volcanic peak, that of Chiriquí, rises from the south Costa Rican cordillera. The lines of volcanic activity in southern Central America do not seem to have moved to any great extent since greater activity in Tertiary times.

AGE OF POSSIBLE OIL-BEARING FORMATIONS

No indications of petroleum, asphalt, or other hydrocarbons have been reported in Mexico or Central America in rocks older than the Cretaceous. The Tamasopo limestone of the Tampico-Tuxpam fields of Mexico belongs to the Middle Cretaceous, according to the Geological Institute of Mexico. The principal oil-bearing horizon of the Tehuantepec and Tabasco-Chiapas fields of Mexico is a Cretaceous dolomitic limestone. The reported bitumen seepages in Guatemala occur in the Cobán limestone of probable Lower Cretaceous age. The seepages reported in Honduras may be traced to the Metapán formation in which Lower Cretaceous fossils have been found. In Costa Rica and Panama, the oil-bearing formation belongs apparently to the Oligocene (or Miocene).

POSSIBLE OIL-BEARING AREAS

On the basis of the preceding summary of the geology of Central America, the discussion of the probable oil reserves may be confined to three major areas: Central and northern Guatemala and portions of southern and northern British Honduras; certain limited zones in northern and central Honduras; both the Caribbean and Pacific coastal zones of Costa Rica and Panama.

Salvador is excluded because of the predominance of late eruptive and

effusive rocks in its geologic structure, and Nicaragua, because of the preponderance of late igneous rocks over three-fourths of its area. In Nicaragua, the known sedimentary rocks are intruded in several places. No well-authenticated seepages occur in the known sediments, which are predominantly shaly in character; and few beds of a carbonaceous character occur. The only area that appears to offer any possibilities of containing sedimentary formations of any considerable extent with favorable structures is the Mosquito Coast (Department of Zelaya); and practically nothing is known about the geology of this area. It is not impossible that limited petroleum reserves exist in Nicaragua, but too little is known of the sedimentary formations of the republic to hazard an estimate as to their extent.

GUATEMALA AND BRITISH HONDURAS

Both the petrographic character and the structure of the Cretaceous and Tertiary sediments of northern Guatemala are favorable to the occurrence of petroleum in commercial quantities. In these sediments two distinctly bituminous horizons occur, the Cobán limestone in the Lower Cretaceous and bituminous limestones in the Upper Tertiary of El Petén. The numerous beds of relatively impervious marl and limestone would appear to insure a sufficient cover to retain the hydrocarbons that may have accumulated in the sandstones, conglomerates, or other porous beds.

The Lower and Middle Cretaceous sediments that occur in Guatemala between latitudes 15° 30′ and 16° are not only strongly folded, but considerably faulted and fractured. In Alta Verapaz, the Cretaceous plateau drops in a series of step faults down to the plain of El Petén.

The Upper Tertiary marls and sandstones of southern Petén and southeastern British Honduras show slight disturbance. The Cretaceous and Tertiary sediments of the northern part of El Petén, though likewise relatively flat lying, give evidence of some folding. The relative positions of the Cretaceous and Tertiary in this Department appear to indicate a series of broad, gentle folds curved in broad arcs convex to the south. The existence of such folds is reported by Sapper and is evidenced in his geologic profiles.

There is accordingly in northern Guatemala and certain adjacent portions of British Honduras a broad zone of moderately disturbed Cretaceous and Tertiary sediments. Bituminous beds occur, associated with a due succession of porous and impervious strata. These are folded into broad anticlines and synclines with gentle dips. Faults are few and of slight displacement. No intrusions of late igneous rocks are known to occur. On the basis of our present knowledge of Central American geology, this appears to be the most favorable area for oil prospecting in Central America.

The American field to which the Guatemala-British Honduras area presents the closest resemblance is the Cretaceous and Tertiary belt of central Texas, exclusive of the Gulf Coast oil fields. This comparison is based on certain analogies of stratigraphy and structure and on the geographic relations between the two areas. It must be accepted at the start, however, that the resemblances noted are by no means absolute.

The oldest stratified rocks of northern and central Guatemala consist of Mississippian and Pennsylvanian, against which Triassic and Cretaceous sediments are faulted down. The Tertiary series succeeds in apparent conformity. In central Texas, the Pennsylvanian sediments are overlain unconformably on the east by Cretaceous, succeeded by Tertiary. In Guatemala, a fault separates the Lower and the Middle Cretaceous; in central Texas, the Balcones fault runs through the Middle and Lower Cretaceous formations. In both areas a series of step-like faults lead to the more or less gently dipping Tertiary sediments. In both areas, the Cretaceous and Tertiary sediments are affected by small folds, which diminish in intensity toward the sea. In Texas, the folded Lower Tertiary gives way to the salt-dome oil fields in the Upper Tertiary of the Gulf Coastal Plain. Oil fields occur on salt domes in the Upper Tertiary beds of the isthmian zone of Mexico in which the geologic formations of Guatemala are continued to the west.

The area of the Department of El Petén is about 13,200 sq. mi. To this may be added 1000 sq. mi. for parts of British Honduras, and 2300 sq. mi. for the less disturbed parts of the Departments of Huehuetenango, El Quiché, Alta Verapaz, and Izabal.

By ratio of areas, assuming the same relative oil content for the Cretaceous and Tertiary of Guatemala and British Honduras as for central Texas, we may estimate the unmined petroleum reserves of the Guatemala-British Honduras area at a maximum of 38,000,000 bbl. Certain reservations would be wise, however. The Cretaceous of Texas has been demonstrated to be productive, notably at Corsicana, Thrall, and at Mexia; the Cretaceous and Tertiary of Guatemala have yet to be proved. The geology of central Texas is known from numerous reports of the U. S. Geological Survey and of the State Geological Survey as well as through oil-company records; that of northern Guatemala and British Honduras is known almost entirely from the explorations of a single geologist, Dr. Karl Sapper. Only at one point is the thickness of the Cretaceous of Guatemala known; the thickness of the Tertiary and of the possible oil-bearing horizons in either Cretaceous or Tertiary are entirely unknown. The continuity of the possible oil-bearing horizons is by no means established. The reported oil seepages may be derived from local pools of limited extent. Moreover, about one-third of this area is faulted to a marked degree. Accordingly, the estimate for the Guatemala-British Honduras area may be reduced to

about 25,000,000 bbl. This again is purely an arithmetical calculation by ratio. It is smaller than the U. S. Geological Survey estimates for any region of the United States except the Atlantic and eastern Gulf Coastal Plains.

HONDURAS

The only well-authenticated seepages in Honduras issue from the Metapán strata. These consist chiefly of clays, red marls, sandstones, and conglomerates, with interbedded limestones. Fossils found in these rocks place their age as Lower Cretaceous. They are conformably overlain by a moderate thickness of Middle Cretaceous limestones.

Good evidence exists for assuming the petroliferous character of the Metapán strata. Well-authenticated seepages occur in this formation in the Guare Mountains of the Department of Cortés and gilsonite exposures in the vicinity of Juticalpa.

The zones of Cretaceous sediments in northern and central Honduras are narrow, broken, and of limited extent. The close folding, faulting, and igneous intrusion to which they have been subjected makes it unlikely that oil pools of any considerable extent should occur. Anticlinal structures have been noted in only a few places on the geologic profiles made by Sapper. Oil pools of limited extent may indeed occur on fault-sealed monoclines or in shattered zones; but sufficient knowledge of local geology is not available to predicate any estimate on this basis. No effort will be made to estimate the oil reserves of Honduras, which from all the evidence presented are of negligible extent in comparison with the unmined reserves in any one of the minor regions of the United States.

COSTA RICA AND PANAMA

The Tertiary sediments of Costa Rica and Panama belong to the Oligocene and Miocene. Eocene sediments have been noted in only cne restricted area of the Azuero Peninsula of Panama. Pliocene formations are restricted chiefly to the upraised coral reefs which occur at intervals along the coast. The Tertiary of Costa Rica and Panama consists of shales, many of which are carbonaceous, interbedded with sandstones and conglomerates, with a few limestone beds. Outside of the Canal Zone, the correlation of the various members is not sufficiently definite to permit at this time any more definite subdivision. The petroliferous character of these Tertiary sediments is fairly well established. Well-authenticated oil seepages occur in the basin of Sixaola River and asphalt seepages occur along Mosquito Gulf in Panama. Other seepages are reported to occur locally along the Pacific coast of both Costa Rica and Panama.

In general, the zones of Tertiary sediments on both the Caribbean

and the Pacific coasts of Costa Rica and Panama are relatively narrow in extent. From their close proximity to the granite ranges, which apparently underlie the late igneous flows of the central cordillera, one may postulate the close folding and steep dips, which have been observed in the Sixaola basin and in Bocas del Toro Province, Panama. Considerable faulting has occurred and igneous intrusions have been noted.

Three areas of the coastal Tertiary of Costa Rica and Panama may be selected for discussion as the most favorable. The first includes the basin of Rio Sixaola in the so-called Talamanca district of southeastern Costa Rica, and the coastal Tertiary zone of the adjacent Province of Bocas del Toro, Panama. The second, or South Darien district, comprises the basins of the rivers which empty into the North and South Bays of the Gulf of San Miguel, on the Pacific coast of the Province of Panama. The third lies in the Azuero Peninsula in the Provinces of Herrera, Los Santos, and Veraguas.

The Talamanca-Bocas del Toro district occupies an approximate area of 3300 sq. mi. Oil seeps along fault lines in shattered dark shales which form the lowest member of the Tertiary. These shales are overlain unconformably by interbedded sandstones and shales, and these in turn unconformably by conglomerates.

The late Miocene thrusts were exerted in lines parallel to the present mountain trends of the isthmus, and these have resulted in the formation of folds striking about N 55°–60° W in southeastern Costa Rica, and northwest-southeast in Bocas del Toro Province, Panama. The districts nearest to the central cordillera show steeply dipping strata, closely folded, highly shattered and faulted, with numerous dikes and other igneous intrusions. The intensity of the folding diminishes toward the Caribbean coast.

The South Darien district comprises a geosynclinal basin of Tertiary sediments in the Province of Panama in the drainage basins of Rio Tuyra and Rio Savannah, and similar Tertiary basins south of San Miguel Bay and in the lower basin of Rio Chepo. These geosynclines are wrinkled into minor folds that strike parallel to the trend of the central cordillera. The formations exposed in these basins are Lower Tertiary sandstones, marls, and limestones. Well-attested seepages occur about San Miguel Gulf. The structure shows anticlinal folds that trend generally northwest to southeast. In spite of the proximity of mountain ranges of late eruptive effusive rocks, no intrusions of igneous rocks have been noted. The area of the South Darien district may be roughly estimated at 1200 square miles.

On the Azuero Peninsula, Lower Tertiary sediments occupy a zone about 30 miles in width and approximately 1500 sq. mi. in area, from east to west across the northern half of the peninsula and portions of the

adjacent mainland. These consist chiefly of sandstones, shales, clays, breccias, and conglomerates with intercalated lignite seams. These sediments are folded into a series of anticlines and synclines that strike about N 85° E. Local faults may occur. Seepages apparently from these formations are said to occur around Montijo Bay.

No oil field of the United States offers a close analogy to the geologic conditions of the Costa Rica-Panama area. In the marked unconformities, close folding, and considerable faulting, the oil fields of southern California offer a resemblance. Late igneous intrusions, however, play no part in the southern California oil fields as in the coastal areas of Costa Rica and Panama. The California fields probably do not occur in such close proximity to a cordillera of late igneous rocks. The sediments of the California valleys occupy a wider area and are more continuous than those of the coastal plains and foot hills of Costa Rica and Panama. Accordingly, no satisfactory basis exists for an estimate of the oil reserves of the Costa Rica-Panama area. There are, in all probability, reserves of unmined petroleum in this territory. But on the basis of the known geology of the district, it is fairly safe to assume that they will hardly be comparable with those of any oil districts of the United States.

Colombian Oil Fields

By L. G. Huntley and Shirley Mason, Pittsburgh, Pa.

(San Francisco Meeting, September, 1922)

The Colombian highlands consist of three parallel mountain ranges (Fig. 1) called respectively the eastern, central, and western cordillera of the Andes. A segment of the range that forms the backbone of the Isthmus of Panama also extends into Colombia, along the Pacific coast, as far southward as the mouth of the San Juan River.

These four mountain ranges furnish the key to the varying climatic conditions and vegetation of the country. The river valleys are hot and covered with dense tropical growth. This is also generally true of the coastal plain, although in the northern portion there are some fairly open stretches where cattle are raised and even small districts where semi-arid conditions prevail. The population of this heavily wooded wet lowland depends almost entirely on river transportation for all intercourse.

The elevated mountainous plateaus of the three ranges are more open and more generally cultivated; all traffic is by means of horses, mules, and the burro. The canoe and river-steamer transportation systems of the valleys are connected with the highland system of trails and roads by short cross trails and the few short railroads of the country. As these cross trails must pass through the intermediate foot-hill region, where the heavy rainfall reaches its maximum, they are exceedingly bad and the life of pack animals is short. The oil development of the Magdalena River valley is in this intermediate region, between the river and the sierras, north of Bogotá.

Geology

Stratigraphy

The predominant sedimentary formations in Colombia are those of Tertiary age. They form the floor of the Magdalena River valley and the coastal plain that borders the Caribbean Sea: this is also true of the southern part of the narrow coastal belt of the Pacific coast.

They are usually of great thickness, as shown in their exposures along the flanks of the eastern cordillera, and along the folds of the more

FIG. 1.—AREA GEOLOGY OF COLOMBIA. HEAVY DASH LINES INDICATE DIRECTION OF FOLDS AND FAULTS.

recent coastal uplift from Barranquilla to the Gulf of Urabá. These are the two regions of greatest promise for oil production in the next generation.

FIG. 2.—SECTION ALONG LINE *AB*, FIG. 1.

B
Lake Maracaibo
RECENT AND QUATERNARY
TERTIARY
TERTIARY
MESOZOIC
MESOZOIC
QUATERNARY IGNEOUS AND
MESOZOIC
Bucaramanga
TERTIARY
MESOZOIC
RECENT AND QUATERNARY
TERTIARY
Magdalena River
TERTIARY
PALEOZOIC
METAMORPHIC IGNEOUS AND
RECENT AND QUATERNARY
Cauca River
METAMORPHIC IGNEOUS AND
PALEOZOIC
TERTIARY
San Juan River
RECENT AND QUATERNARY
Pacific Ocean

15,000' 8000' 0

The lower portion of these Tertiary sediments is of the greatest value as a source of oil; this is usually called the coal series and includes numerous beds of coal, which sometimes are 30 ft. thick. This is probably the equivalent of the Pavjí series of the Lake Maracaibo district. The coal in this series seems to thin out rapidly and disappear to the westward, as does the coal in the Pavjí series in crossing Lake Maracaibo to the eastward. Only a few thin streaks of coal are found west of the Magdalena River.

This coal series contains petroliferous shales and slates, and at the outcrop presents a number of oil-saturated sands. It is underlain by black slates and limestones of Cretaceous age (Fig. 3), which are also petroliferous at the outcrop. The lower portion of this latter series has marine fossils, which may be of Jurassic age. The lower Tertiary coal series and the underlying Cretaceous represent the probable source of all the oil that will be found in the Magdalena valley. All of the seepages found in this region are exudations from these sediments or from those closely in contact with them.

The smaller seepages in the coastal region, in some instances, come from upper Tertiary formations, which in themselves present no further evidence of containing material from which these seepages might have been derived. It is probable that here also the origin of the oil has been from underlying formations of the same age as those of the eastern cordillera and the Maracaibo basin. Along the coastal uplift, these oil shales and sands usually lie at considerably greater depths than they have been found on the folds in the Magdalena valley. The overlying sediments in the coastal region are of such thickness that it is not likely that the underlying oil sands can be encountered by the drill except on a few of the highest folds, where erosion has removed the upper beds. No past drilling has furnished conclusive evidence of the presence or absence of oil in this coastal uplift, as all wells have been either badly located, from a

structural standpoint, or were not carried to a sufficient depth to test their respective localities.

The Tertiary formations of the west coast are not favorable for oil, being largely composed of sandy clays and clay shales, with little or no organic matter. No seepages are known or could be found west of the central cordillera and south of the Atrato River. Paleozoic rocks are encountered on the Pacific slope of the western cordillera, and also on the

Magdalena formation. Red and yellow sands and coarse conglomerates, probably of lacustrine origin. Unconsolidated. 0–1000 ft. thick.

Oponcito formation. Soft sandstones and clay shales. Reddish and bright-colored grading into gray sands at the bottom. 8000 ft. thick or more.

Fossils infrequent.

Oil production from near bottom.

La Paz sandstone series. Massive hard quartzitic sandstones. Cliff forming. 1000 ft. thick.

Umir coal series. Gray shales and coals, thinning to the west. Interbedded sandstone. 2000 ft. thick. Oil seeps from outcrop.

Churcurri slates and shales. Carbonaceous and oily at the outcrop. 1000 ft. thick.

Palmira limestone. Cherty limestone, weathering white in cliffs. Carbonaceous at times on fresh outcrop. Fossiliferous. Many large ammonites. 500 ft. plus.

Fossiliferous black slate and limestone. Petroliferous at the outcrop.

FIG. 3.—GENERALIZED STRATIGRAPHIC SECTION IN EASTERN MAGDALENA VALLEY BETWEEN CARARE AND SOGAMOSA RIVERS.

west side of the Magdalena valley. They consist largely of altered slates and are of no importance from an oil standpoint.

Structure

Each of the three main ranges of the Andes represents a geanticlinal uplift, of which the backbone is composed of granites and other igneous rocks, and altered igneous and metamorphic sedimentaries. The latest

Tertiary formations show little folding at their contact with these great folds. A number of late cross stresses have set up other folds in a direction roughly parallel to the present northern coast line. The last of these is still in process of adjustment, and is marked by a series of mud volcanoes and seepages along the coast. Coral limestone reefs of recent origin are found along the strike of this fold, in places several hundred feet above the present sea level.
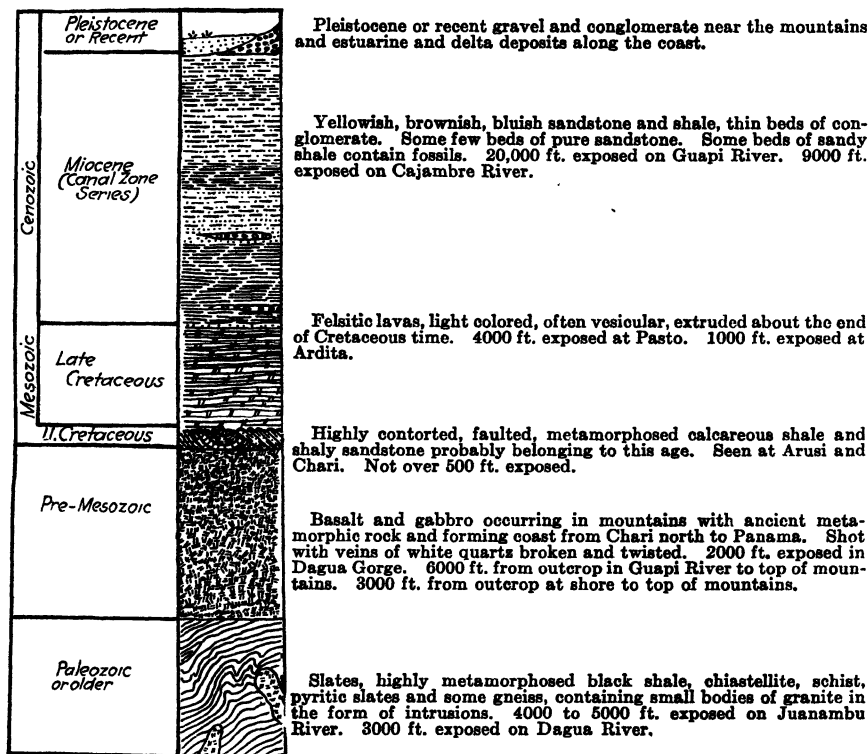
FIG. 4.—GENERALIZED STRATIGRAPHIC SECTION IN SOUTHWESTERN COLOMBIA AND ALONG PACIFIC COASTAL SLOPE, AFTER BOSSLER.

The folds flanking the eastern cordillera on the west may be considered promising, also the cross folds mentioned, when not too badly faulted. Such folds cross the Magdalena River southward from Puerto Berrio, in the vicinity of Mompos and Banco and also along the coastal zone mentioned previously.

It must be remembered that while some of these folds are very steep, and often faulted and difficult to map accurately, in many ways they seem to offer parallel conditions to those of the California fields. The authors believe that with a stratigraphic column at least 3000 ft. thick

composed of oil shales and sands, such as is found along the eastern side of the Magdalena valley, a number of areas of great productivity will be found. With such a great thickness of rich sediments, experience and common sense indicate that not nearly the same extent of lateral gathering ground is necessary to produce commercial accumulations on folds, as when the source beds are thin. Steepness of dip and faulting have been no handicap in California, the oil-bearing formations of which are of the same age as those of Colombia. The geology in Colombia will be much harder to map because of the heavy mantle of vegetation.

LAND LAWS

Four classes of lands, in Colombia, carry oil rights under different conditions, as follows:[1]

1. Private properties and government sales of public lands, the titles of which were issued previous to Oct. 28, 1873.

2. Lands sold as uncultivated government lands after Oct. 28, 1873.

3. Lands on which titles to "oil mines" were issued by the government in 1912 and 1913, up to April 1.

4. Oil claims acquired under the present law, passed on Dec. 30, 1919.

The taxes and conditions imposed by the present law on lands under all groups except the third are so onerous as to make operations unprofitable at present. A realization of this will no doubt cause the law to be modified, either by decree or application, so as to encourage development; otherwise the high cost of production under the physical conditions existing in tropical countries plus the high royalties and taxation will retard oil development in Colombia. The terms of the present law were the result of a more or less superficial study of the Mexican field, without realizing that under the different geological conditions existing in Colombia, a duplication of the phenomenally large wells of Mexico was not to be expected. With smaller wells, or wells of sharp decline curve, high royalties and taxation merely penalize all development, without helping any one. In the third group are several concessions the terms of which were less difficult, and in one case at least not impossible of fulfillment.

For administration purposes, and also for the levying of a development tax, the law divides the republic into three zones, as follows (Fig. 5):

Zone 1, lands lying at a distance of 200 km. or less from the sea coast.

Zone 2, lands lying 200 to 400 km. from the sea coast.

Zone 3, lands lying more than 400 km. from the sea coast.

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[1] Johnson, Huntley, and Somers: "The Business of Oil Production." New York, 1922. John Wiley & Sons.

Taxes within these various zones vary from 6 to 10 per cent. on gross production. The law makes exceptions for a region bordering the Gulf Uraba (Zone A), which for the time being is eliminated from development, and also a strip 20 km. wide along the west coast (Zone B), which is of no interest for oil whatever.
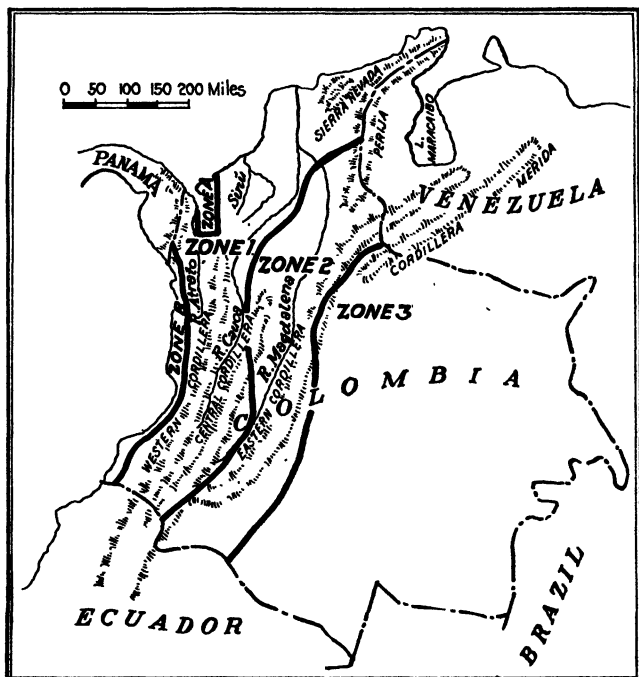
FIG. 5.—PETROLEUM ZONES IN COLOMBIA.

FUTURE DEVELOPMENT

The hope of success in prospecting for oil in Colombia will be restricted to four general regions, which may be listed as follows, in their order of promise:

1. Magdalena valley, particularly the eastern side.
2. Northern coastal plain, from Barranquilla to the Gulf of Uraba.
3. Headwaters of the Catatumbo and Rio Oro on the Venezuelan border, on the eastern flank of the eastern cordillera of the Andes.
4. Territory drained by the headwaters of the Meta and Caquetá, on the eastern flank of the Andes.

Magdalena Valley

The Magdalena valley north of Honda, and particularly the part drained by its eastern tributaries, presents the following favorable conditions for oil development:

1. Commercial production of a high-grade oil already developed on the Colorado River by the Tropical Oil Co.

2. Large active seepages in greater number and extent than in any other region in Colombia.

3. A greater thickness of source beds exposed in the mountain outcrops (3000 ft. or more).

4. Good structural folding in places.

5. Navigable river transportation to the coast, as well as a feasible low-grade pipe-line route to deep water.

Northern Coastal Plain

Seepages are small and less frequent in this northern region than in the Magdalena valley, and the probable source beds are not exposed. Drilling will, therefore, be deeper and exploration carried on with less certainty as to depth and results. The fact that both good folding and seepages exist over wide areas make this region promising.

Of all geologically favorable regions in Colombia, this northern coastal plain is the only one that has the advantage of being near good sea harbors, and close to the Panama Canal. Greater chances can be taken, and more money can be spent legitimately in such a coastal region than would be justified elsewhere. None of the drilling up to the present time has furnished any conclusive evidence as to the existence of commercial oil deposits.[2]

Atlantic Slope of the Andes

The eastern flank of the eastern cordillera contains several known areas of seepages similar to those of the Magdalena valley. So far as this region has been examined, it shows equally as good promise for oil from the standpoint of Tertiary stratigraphy and structures as the latter. However, development of the former region will be postponed for the following reasons:

1. All known promising structures are from 100 to 500 mi. from the nearest navigable water, which is still from 1000 to 3000 mi. from the seacoast, and still farther from a market; or, the price of oil for some time will not justify the cost of pumping oil over a 5000-ft. mountain divide, to a point that is several hundred miles inland, and still farther from the nearest market in Panama.

2. Portions of this eastern slope are in territory disputed by several governments.

3. High costs of operations.

. [2] Chester W. Washburne and K. D. White: Oil Possibilities of Colombia. See page 1023.

Maracaibo Basin

That part of the Maracaibo basin that lies along the Venezuelan border has a limited amount of territory in which are found both good structure and good source beds, with oil indicated both by seepages and actual wells, although the latter are of relatively small size. The difficulties of transportation to markets are of the same character as those of the Atlantic slope, although less in degree. With the main range of the Andes to the west and an international boundary on the east, the matter of producing oil and marketing it successfully is still to be solved.

The Pacific Coast

This region offers no oil possibilities for reasons given previously, and the southern Andean plateau, including most of the Cauca and Patia valleys, is composed of igneous and metamorphic rocks, with some thin barren inliers of sedimentaries. Hence this entire region can be disregarded.

The entire question for the immediate future resolves itself in the oil possibilities of the lower Magdalena valley and the northern coastal plain. The writers believe that there is in these two districts a large oil reservoir which can be exploited profitably, although this can probably be done to best advantage by adapting California methods rather than those of the Mid-Continent fields. The proximity of the Panama Canal makes northern Colombia a strategic point of oil production.

Oil Possibilities of Colombia

By Chester W. Washburne and K. D. White, New York, N. Y.

Colombia has an almost ideal situation with respect to the world's markets, being only a short distance from the Panama Canal and the West Indies. The sailing distance from its Caribbean ports to New York is less than that from Tampico.

Geographically, Colombia consists of three systems of broad mountain ranges separated by two long narrow valleys. The Cauca River valley separates the Western or Coastal range from the Central range. On the west side of the Coastal range are the Atrato and San Juan Rivers. The Central and Eastern ranges are separated by the Magdalena River valley. In the department of Santander the Eastern range divides, one branch continuing northward as the Cordillera of Perija, the other turning eastward across Venezuela as the Cordillera of Merida. Between these ranges is the great basin occupied by Lake Maracaibo.

These valleys consist mainly of long narrow reëntrants or tongues of Tertiary sediments between the older rocks of the mountain ranges. The late Miocene and younger sediments seem to have been deposited in separated basins, but the Cretaceous and possibly some earlier Tertiary strata were laid down more or less continuously over a great part of Colombia and Venezuela. These strata were subsequently folded into the present ranges. The deposition of the Cretaceous and of some of the Tertiary was in a great sea, the main land mass being in Brazil, with land probably along some of the cores of the present mountain systems. The sediments, except the Lower Cretaceous, vary markedly in character and thickness. An illustration is seen in the massive series of non-marine conglomerates and sandstones of the upper Magdalena River valley, near Honda, which are almost wholly wanting in the coastal sections where marine sediments prevail.

The oldest rocks bearing on the search for oil are the black carbonaceous and bituminous shales, limestones, and cherts of the upper part of the Lower Cretaceous. These probably are the main source of oil in both Colombia and Venezuela. They include thick bodies of true "oil shale." Above the carbonaceous and bituminous beds, lies a series of many thousand feet of clastic sediments. The most noticeable and easily recognizable formation among these clastic sediments is the "coal-bearing series," which may be either upper Cretaceous or Eocene. The
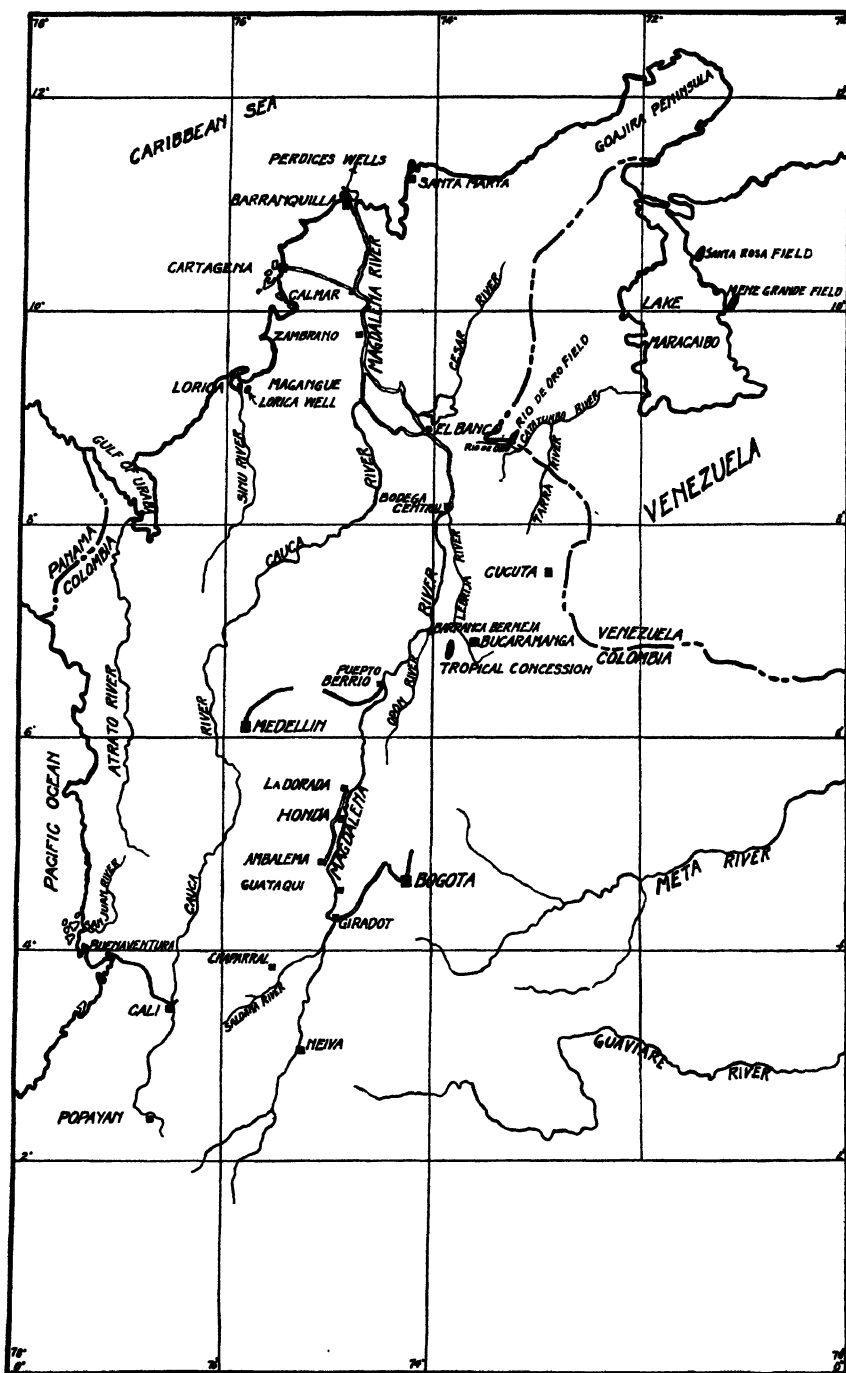
FIG. 1.

horizons that have been proved to be petroleum producers in Colombia occur in this series and in the next overlying formation.

The several stratigraphic sections available for Colombia are platted in Fig. 2, with tentative correlation lines drawn between formations which are thought to be nearly contemporaneous. Special thanks are due A. Faison Dixon for his aid in the correlations.

For comparison, Colombia may be roughly subdivided into the following areas or districts:

1. The Santa Marta area and the Goajira peninsula.
2. The Sinu River valley.
3. The Cartegena-Barranquilla coastal region.
4. The lower Magdalena River valley.
5. The upper Magdalena River valley.
6. The Amazon River drainage, including the Meta River basin.
7. The Colombian-Venezuelan boundary region.
8. The Pacific coastal plain.
9. The Atrato River valley.

SINU RIVER VALLEY

The sediments of this valley are of Miocene and Pliocene age, in large part marine. They have been sharply folded and in some localities have been faulted. Sharp anticlines with limbs dipping 45° or more near the crest are the rule.

Elfred Beck[1] describes several structures which he located east and north of the Sinu River, the most important being the Sincelejo anticline, which he traced for more than 50 mi. (80 km.) a little east of north. The eastern flank of the anticline dips from 5° to 30° while the western limb dips at 45°. In Beck's cross-section the folding becomes sharper westward and considerable faulting is shown.

West of the Sinu River, between Monteria and the sea, three anticlines have been recognized and each traced about 10 mi. (16 km.). The two more eastern folds are separated by a syncline about 10 mi. wide between anticlinal axes. The anticlines are closely folded, the limbs towards the synclines having an average dip of about 45° for a mile from their crests. The other limbs are nearly vertical. The petroleum evidences on both anticlines are similar; large mud volcanoes and seepages of dark green oils come from their steeply dipping crests. The anticlines, the crests of which are in Miocene rocks, expose 7000 or 8000 ft. (2134 to 2438 m.) of strata. The third anticline is situated on the sea coast, probably in Pliocene rocks. It also shows oil seepages.

The petroleum evidence of this district consists of seepages of heavy

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[1] Elfred Beck: Geology and Oil Resources of Colombia. *Economic Geology* (Nov., 1919).
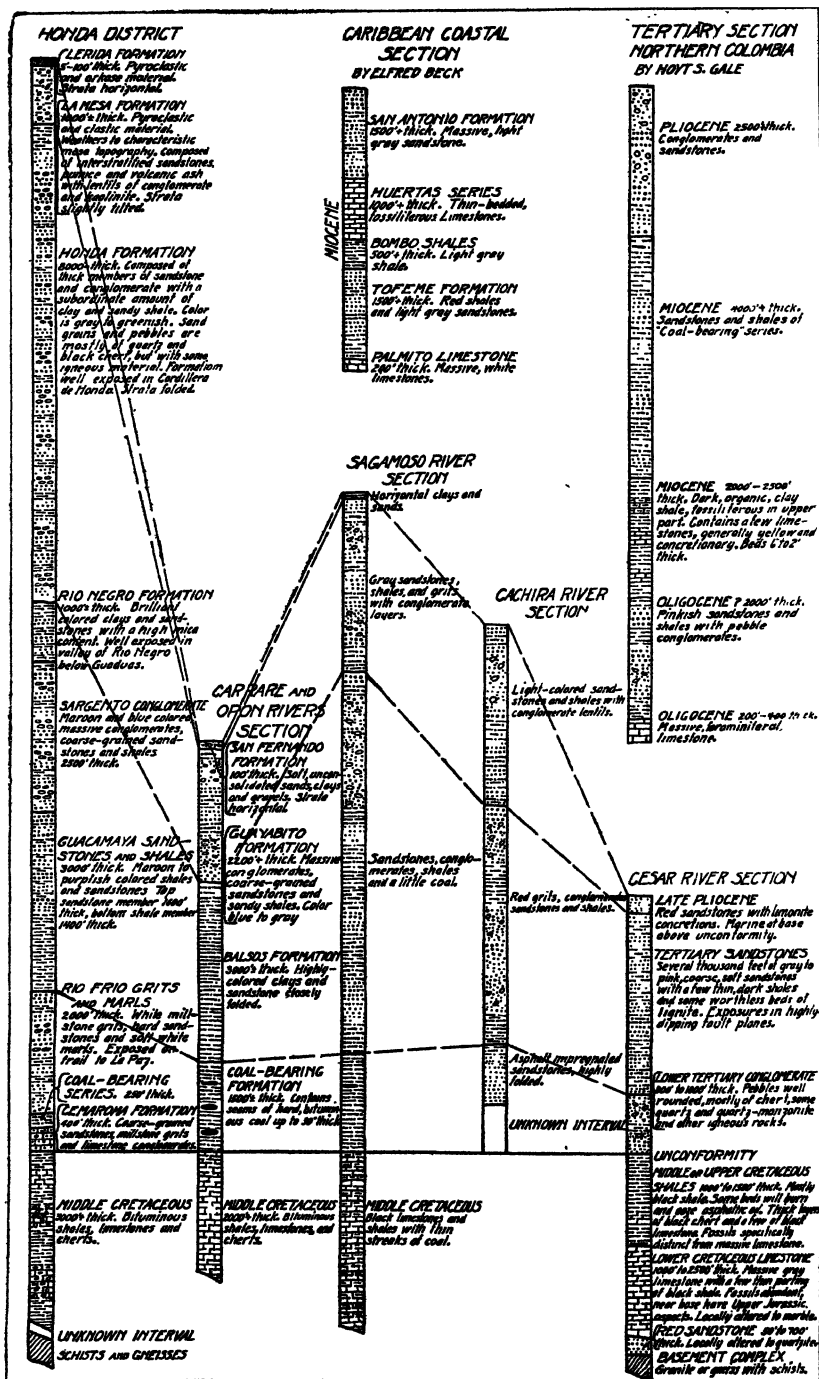
FIG. 2.—CORRELATED STRATIGRAPHIC SECTIONS OF COLOMBIA.

tarry oils, dark green oils, gas springs, and mud volcanoes.   The authors believe that mud volcanoes have little significance.   Practically all oil and gas seepages are located either in the fractured core of closely folded anticlines or along fault planes.

Four wells have been drilled in the Sinu River valley and another is now being drilled by the South American Gulf Oil Co., at San Andres. The first well was drilled by a Dutch company in 1910 to 1912, near San Sebastian, east of the Sinu River.   It reached a depth of about 400 ft. (122 m.) and encountered a dark green oil of about 36° Bé. gravity.   It was reported to have been a 50-bbl. well.   In 1915 it was badly mudded and would pump only a few barrels.   The Gulf Oil Company's well is said to be located on good structure.

In 1915 and 1916 The Standard Oil Co. of New York, drilled three wells to depths of about 2000 ft. (610 m.)   The wells were located, respectively, 4 mi. (6.4 km.) east, 8 mi. northeast, and 12 mi. northwest of Lorica.   These wells are all reported to have struck small shows of oil and gas, but nothing has been definitely proved by them.   All except the shallow well were drilled quite far down the flank of the structures on which they were located.

The entire region contains thick deposits of Tertiary sandstone and shale which are closely folded and much faulted.   Any structure in this district deserves testing where the equivalent of the "coal-bearing" series of the Magdalena River valley, or strata close above it, may be reached by the drill, where the rocks are not too badly shattered, and where the drainage area is not too limited.   With the thick shale section of the later Tertiary and with the presence of numerous petroleum seepages, well-developed folds in the Miocene and Pliocene rocks are also worthy of test, but it should be recognized that no oil horizons have been proved in these strata and that drilling to upper Tertiary horizons is much more speculative than drilling to lower Tertiary and Upper Cretaceous horizons.

CARTEGENA-BARRANQUILLA COASTAL REGION

This area roughly embraces the region between the Caribbean sea coast, the Magdalena River, and the Cartegena-Calmar railroad.   In this territory about as many wells have been drilled as in all the rest of Colombia.

About 6 mi. (10 km.) southwest of Puerto Colombia, at the village of Perdices, seven wells had been drilled up to the beginning of 1918, all within 1000 ft. (305 m.) of each other.   An eighth well is located some 3 mi. east of this locality.   According to Alfred Beck, the drilling was commenced by a Canadian company in 1907, and was continued in 1912 or 1913 by the Kelly Oil Co.

. Of these eight wells, the deepest, No. 6, Perdices, was drilled to 3030 ft. (923 m.).   This well encountered showings of oil and gas at

various horizons, and at 2443 to 2446 ft. (745 m.) in a sandstone, 3,000,000 cu. ft. (85,000 cu. m.) of gas was reported. Well No. 1, Perdices, had the best showing; salt water and oil, with a gravity of 32° Bé., were reported at 135 ft. (41 m.), while in a broken sand and shale formation at 745–754 ft. (230 m.), oil of 43.4° Bé., which flowed naturally and on the pump made 140 bbl. in 7 hr., was reported. In 1917 the well was badly mudded but pumped about ½ bbl. a day. The other wells, which were drilled to depths of 1600 ft. (488 m.) or less, encountered shows of oil and gas.

The Three Seas Oil Co. is now drilling a deep test in the Perdices region at no great distance from the other wells. Beck states that the drilling has been on a steep, narrow, poorly defined fold.

At Turbaco, a station on the Cartagena-Calmar railroad, the Martinez Oil Co. drilled five wells, beginning in 1908. The first four wells were drilled with a Keystone machine to a depth less than 500 ft. (152 m.). They were all located close together in the immediate vicinity of numerous and active mud volcanoes, which had been given prominence by their description by Von Humboldt. In 1912, another well was drilled about a mile south of the mud volcanoes, by the Martinez Oil Co., with a standard rig. It went to a depth of about 2200 ft. (671 m.), its log showing remarkable uniformity in rock strata. This is readily explained by an excavation 500 ft. east of the location, where strata striking N. 40° dip westward from 75° to vertical.

The surface evidences of petroleum in this district are large, active mud volcanoes, gas springs, and seepages of dark green oil. Unfortunately, none of the drilling seems to have been on large, well-formed structures. The drilling on suitable structures in this region seems to be amply justified.

LOWER MAGDALENA RIVER VALLEY

This is one of the two areas that have been proved to contain commercial oil deposits. The field is about 300 mi. (483 km.) from the mouth of the Magdelena River, and 20 to 25 miles eastward, up the Colorado River. The discovery well was drilled in April, 1918, by the Tropical Oil Co., after abandoning the first two wells, at about 600 ft., owing to great difficulty with caving. Of the three producing wells, No. 1, drilled 2300 ft. (700 m.) was good for 3500 bbl.; No. 2, drilled 700 ft. (213 m.), 700 bbl.; No. 3, drilled 1500 ft. (457 m.), 1500 bbl. The gravity of the oil was between 42° and 45° Bé. The company was purchased in 1920 by the Standard Oil Co. of New Jersey, which has built a railroad from Barranca Bermejo to the field and has shipped in large supplies of pipe, casing, tools, etc., preparatory to extensive developments.

North of the Tropical Oil Company's property, on the Lebrija River, the Colombian Syndicate has drilled several wells, all of which had oil showings, though none produced appreciable amounts.

At Zambrano, which is about 100 miles (160 km.) from the mouth of the Magdalena River, two wells have been drilled, one by the Shell interests, the other by the Transcontinental Oil Co. The Transcontinental well, at between 700 and 800 ft. (213–244 m.), struck a big flow of salt water in a sand 150 ft. (45 m.) thick. The well is located on the plunging end of an anticline.

The Standard Oil Co. of California is drilling two wells, about 8 mi. (12.8 km.) apart, near El Carmen and San Jacinto, about 60 km. southeast of Cartagena, on the large concession of the Latin-American Petroleum Corpn. The San Jacinto well had a good show of oil and gas at 756 ft. (230 m.). The wells are in good structure.

In this region there are large deposits of asphalt or brea, active seepages of dark green oil, gas springs, and veins of devolatilized asphaltum varying from gilsonite to grahamite. Here the large deposits of asphalt and hydrocarbon veins are restricted to the east side of the valley.

In crossing the Lebrija River region eastward from the Magdalena River, the first encountered rocks dip west; on the west side of the Magdalena River the rocks dip eastward. The syncline is coincident with the Magdalena Valley as far as El Banco, from which place it narrows and continues with more complicated structure up the valley of the Rio Cesar.

Since the occurrence of oil in commercial quantity is now proved, one should test any structure that is not too contorted, under which the drill can reach the coal-bearing series or strata just above it.

UPPER MAGDALENA RIVER VALLEY

This refers to the region above La Dorada, the head of navigation of the lower Magdalena. It includes the Honda district, where recently there has been considerable leasing and some drilling.

The region has numerous small anticlines. Its major structure is a series of great north-south block-faults, the throw of some of these being many thousands of feet. There are two main lines of seepages. One is along the east side of the Magdalena valley, or the west foot of the second range of the Eastern Andes, locally called the Cordillera de Sargento; the other follows the west side of the valley, or the east foot of the Central Andes. Most of the seepages below Guataqui consist of heavy asphaltic oil or brea, while most of those above Guataqui consist of dark green oil, but exceptions occur.

An overlap of Tertiary strata conceals the structure in the older sedimentary rocks in many places along the east base of the Central Andes. There are exceptions to this, as at Chaparral, where there is an anticline with a large deposit of brea. The seepages of the Eastern Andes occur either on small folds or along fault planes. They are located

either near the top of the Lower Cretaceous black shales and limestones or in beds a few hundred feet higher.

Three wells have been drilled. In 1918 the Carib Syndicate drilled a 700-ft. (213 m.) hole about 3 mi. (4.8 km.) southwest of the San Felipe railroad station. A little tarry oil was found in the bottom of the hole, which stopped in schist.

In 1921 the Transcontinental Oil Co. drilled a well about 15 mi. (24 km.) southeast of Honda, to a depth reported at 3300 ft. (1005 m.) stopping in black limestone. From 800 to 2000 ft. (244–610 m.), several shows of oil and gas were encountered. Near Guataqui, the same company is reported to have drilled 1170 ft. (357 m.) and to have encountered small showings of oil.

In the upper Magdalena River valley there are two long lines of seepages, and many small anticlines which give promise of producing oil. Whether the quantity can be sufficient to justify the cost of transportation to the coast is doubtful. The problem of this region is one of transportation and quantity of oil, rather than the actual presence of oil, which can hardly be questioned.

ESTIMATED AREA OF POSSIBLE AND PROBABLE OIL LANDS

The three wells producing in the Tropical Oil Company's concession, and the one on the Barco concession, indicate a certain amount of commercially proved and highly probable oil land, but the present stage of development at these two places is not sufficient to enable us to estimate either the proved acreage or the probable production from proved land in Colombia. All we can say is that, on geological evidence, we think there is roughly 5 sq. mi. (13 sq. km.) of "probable" oil land in the two fields mentioned. On account of the small number of wells drilled, the rest of the attractive acreage must be thrown into the "possible" class.

In the Magdalena Valley above El Banco, there may be 20 to 30 sq. mi. (52–78 sq. km.) of possible oil land in addition to the probable oil land above mentioned.

Below El Banco it is necessary to consider the Magdalena River valley in connection with that of the Rio Sinu, with which it has close relationship in many ways. Although we have worked in several parts of this region we do not feel competent to estimate the possible oil areas within it. The anticlines generally are narrow and sharp, and on most of them it is not possible to reach the base of the Tertiary. In other words, most of them may never produce oil. The district, however, is highly attractive because of the possibility that oil may have migrated long distances upward into the attainable Tertiary strata, and because the Marine Tertiary of this region offers attractive possibilities as an additional source of oil. Our best guess is that there may be 30 sq. mi. of possible oil lands in this district.

Concerning the Atrato River valley, neither of us knows anything. The region is very unhealthy. We understand that geologists have reported favorably on parts of this territory, but we cannot make any estimate concerning it.

The west coast of Colombia, so far as we can ascertain, has no known potential oil lands. There generally is only a narrow strip of unaltered strata west of the metamorphic and igneous rocks of the mountain mass, except north of the Rio San Juan, where the old rocks disappear; and also near the south end of Colombia, where there is said to be an embayment of Tertiary in the older rocks.

Our information is inadequate for much comment on the three other districts mentioned in the subdivision, namely, the Santa Marta areas and the Goajira peninsula, the Amazon River drainage (Meta River basin), and the Colombian-Venezuelan boundary area.

In the Colombian-Venezuelan boundary region the Barco concession in Colombia and the Colon Development Company's property in Venezuela have been partly tested. The Colon Development Co., since 1914, has drilled four productive wells on the north side of the Rio de Oro and on the Rio Tarra. One of the Rio de Oro wells had a good flow of 27° oil at about 1000 ft. (305 m.) and another well had 32° oil of good refining quality at about 2000 ft. A shallower well, reported at about 800 ft. (244 m.), had oil of about 23° Bé. The exact size of these wells is unknown but they indicate the presence of an oil field in that region.

In 1920 the Colombian Petroleum Co. drilled on the south side of the Rio de Oro on the Barco concession. The well is reported 800 ft. deep and good for 500 bbl., with a gravity of about 27° Bé. The company is preparing to drill on the Rio Tarra.

Oil seepages are reported in places along the entire eastern front of the Cordilleras, including the Meta River basin; the geology is known to be attractive, but the transportation problems seem forbidding.

Oil Resources of Ecuador

By V. F. Marsters, Kansas City, Mo.

Seepages of oil in Ecuador have been known for many years. The locality first to receive attention, and still worked in a modest way, lies on the north shore of the Santa Elena peninsula, between La Puntilla and Salinas. Operations were first conducted largely by local capital. Later, foreign companies acquired large blocks of land, where they have done considerable drilling. For details concerning present drilling activities in Ecuador, we are indebted to Leon Price, an American resident engineer of Ecuador. The information solicited came to the author through the office of the American Consulate,[1] with permission to use it.

The coastal plain bordering the Province of Tumbes, Peru, continues to the northeast, but narrows on passing the Zarumilla River. From this point northward it widens toward Santa Rosa. In fact, the pampas, as shown on the official Wolf map, correspond fairly well to the areal extent of the coastal plain as far as Guayaquil. This is shown in a general way on the accompanying map. The author understands that the Guayaquil embayment is bordered on all sides by the coastal plain, as recognized at Tumbes and farther southwest, and that it can be followed without a break as far north as Palmar. This view is supported by the mapped area regarded by Mr. Price as favorable or possible oil territory. Moreover the islands in the Gulf of Guayaquil are the same geologically as the coastal edge, being composed of Tertiary sediments. The author understands also that Tertiary formations occur north of Palmar. According to a report by Mr. Cardon, of Guayaquil, Tertiary sediments are exposed in the region of San Francisco and extend to and beyond Pta. Calera. There is also reason to believe that at least the lower part of the valley of the Santiago River is occupied by Tertiary sediments, which probably border the coast line and extend into the River Mira basin, and into Colombia.

On the south coast there are many seepages, most of them near the inner edge of the coastal plain, between Zarumilla and Santa Rosa and also between Santa Rosa and Machala. In the latter region are

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[1] F. W. Goding, Consul General, Guayaquil, Ecuador.

many low places or basins, usually partly filled with water; at times, the pools in these lowlands show oil films. The author is informed also
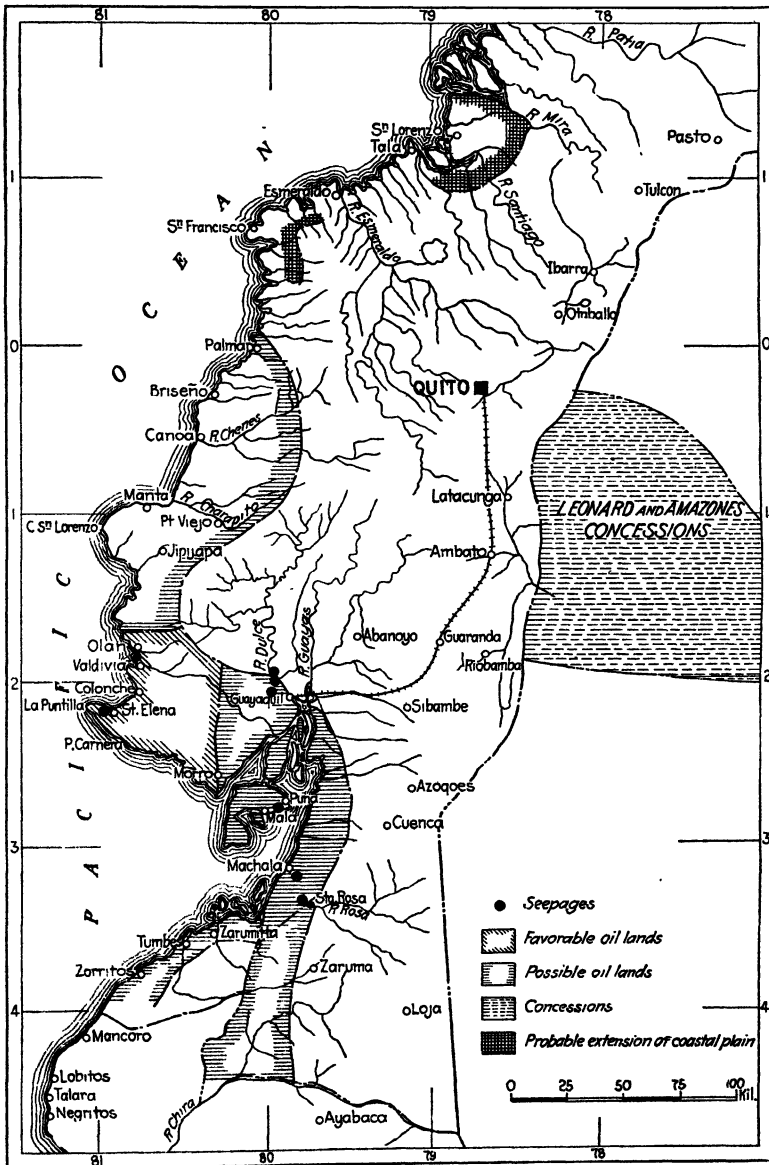
FIG. 1.—PETROLEUM REGIONS OF ECUADOR.

that a few seepages are known between Machala and Nevasa. It is reported that seepages occur on Puno Island, which, as well as many other islands, is thought to be composed mainly of Tertiary sediments.

In reality, it represents a partly drowned section of the coastal plain. The Machala-Santa Rosa seepages are vouched for by Mr. Cardon, and have been mentioned also by other engineers. North of Guayaquil in the basin of the Guayas (or Daule) River, at Portrillo and near Daule, also to the west of Guayaquil near Chongon, seepages are mentioned by Price and others.

In the interior, east of Quito, seepages occur near Baños. Seepages on the coast line, at or near Manglar Alto, are mentioned by Price, Cardon, and others.

Concerning present operations in Ecuador, I quote the statements of Mr. Price.

Anglo-Ecuatoriano Oil Fields Co., Ltd., or "Ancon Oil Co."—This company was organized in 1919 to take over the holdings of the Lobitos Oil Co. in Ecuador, and in fact is a subsidiary of that company. Its property includes approximately 129,000 acres (52,204 ha.) in the region of Ayangue, about 30 km. northeast of Santa Elena. The company drilled one test well 1600 ft. (488 m.). Drilling activities stopped about July, 1921.

It holds some 35,000 acres (14.160 ha.) in the vicinity of Ancon, about 12 km. south of Santa Elena. A well was drilled about 2 km. west of the village, but is now practically abandoned, although it is capable of producing a few barrels of oil per day. Well No. 2 of this company, 1 m. (1.6 km.) east of No. 1, was drilled to 2800 ft. (853 m.) and produced about 40 bbl. per day. Well No. 3 is being drilled about 1 mi. northeast of No. 3.

British Controlled Oilfields Co., Ltd.— This company has optional holdings on approximately 60,000 acres (24,280 ha.) located about 15 km. inland from Punta Centinella. The company drilled three wells, two of which are located near the shore and were drilled to depths of 700 and 2000 ft. (213 and 610 m.), both showing oil. A third well was drilled near Volcancito, about 25 km. east from Santa Elena, the depth of which is unknown. The company closed its operation in Ecuador in December, 1921, and concentrated its activities in Venezuela.

International Petroleum Co., Ltd. (of Canada).—This is a subsidiary of the International Oil Co. of Canada. It is drilling a well near Manta, which in August, 1921, had reached a depth of 1500 ft. (457 m.), with no indications of oil. Under the old law, this company denounced a block of 1000 acres (405 ha.) and also bought a number of claims scattered over the Santa Elena and Ancon regions. It is producing no oil.

Sackay & Harmsworth Exploration Co.—This company controls about 20,000 acres in the extreme western part of the Santa Elena peninsula, the producing property lying southwest of Santa Elena town. The company has three wells, each about 700 ft. deep (213 m.). It has also a number of open pits which produce considerable seepage oil. The oil from the wells averages about 45° Bé. but that from the pits

ranges from 20° to 40°. The company purchased a small refinery at Saltivo. The production is sold at Guayaquil and is sufficient to interfere with the imports of that commodity.

Local Petroleum Interests.—A number of hand-dug pits lie south of Salinas, about 40,000 bbl. being the annual output. This is refined at the Carolina refinery. Another Guayaquil syndicate is drilling a well on its property near Ancon.

Leonard Exploration Company of New York.—This company has been merged recently into the Andes Corporation of New York. It has been granted a concession in El Oriente of 20,000 sq. km., divided into two zones, one around the village of Canelas, and the other farther north along the Cofanes and San Miguel Rivers. Under its contract, the company paid a deposit of 500,000 sucres and must build roads through the cordilleras to their properties. A telephone line is being constructed.

Amazon Corporation of Canada.—This company has been granted a large concession in El Oriente, also divided into two zones. The total acreage is not known, but it is near the concession granted to the Leonard Exploration Co. The company agrees to enter into the construction of railway on a large scale.

SUMMARY OF POSSIBLE TO FAVORABLE OIL-BEARING TERRITORY

The author has no reliable information upon which to make any calculations on the possible, probable, or favorable oil-bearing territory at the east foot of the Andes. There is no doubt that oil seepages occur near Yurimaguas, on the Haullaga River, close to the boundary between Peru and Ecuador.[2] Cretaceous and probably Tertiary sediments are thought to occur in this eastern province. Geological conditions are similar to those in the coastal province, and duplicate in part what is reported in the Argentine and Bolivia. Hence one may expect to find oil deposits of commercial importance in the oriental provinces of Ecuador and Peru.

A summary of possible to favorable oil territory in the coastal province of Ecuador is given in the following tabulation.

	Sq. Km.	Acres
Favorable territory:		
Part of Santa Elena peninsula..............	4,704	1,181,888
Possible to probable territory:		
Olan to Palmar (coastal plain).............	5,152	1,271,541
San Francisco area (coastal plain)..........	1,400	345,800
Mira-Santiago area (coastal plain)..........	2,800	691,600
Guayaquil (coastal plain)...................	2,464	608,608
Islands, Gulf of Guayaquil.................	1,232	304,304
Guayaquil to Peru border (coastal plain).....	3,360	830,020
Interior belt, Santa Rosa to Peru border.....	4,088	1,009,736
Total possible to probable................	20,496	5,061,609

[2] Publications of Ricardo A. Deustua, Lima, Peru.

This tabulation is based on the map submitted by Mr. Price; the following remarks may help elucidate them. Mr. Price indicates on his map that an interior oil belt extends south from Santa Rosa to the Chira River. He considers the belt as possible oil-bearing territory.

Investigations by the writer east of Tumbes, believed to touch the area shown on the map, did not reveal any of the oil-bearing formations of the coastal region. Moreover, the older sediments observed had undergone complicated folding and metamorphism. Therefore, the author suspects that similar conditions exist to the south and throughout much of this area, as shown on the map. The possible producing area of this part of Ecuador seems to be confined to the coastal region.

Following this view, the grand total of possible producing territory on the west border of Ecuador would measure approximately 5,091,609 acres (2,060,574 ha.). Deducting the interior area from the above total, leaves about 4,081,873 acres to represent a more conservative view of the total. From the stratigraphical data obtained from various sources, it seems probable that not more than 20 per cent. of this area may be placed in the "probable" list. Hence, in this case we would have:

	ACRES	HECTARES
Possible producing territory, coastal area of Ecuador.....	4,081,873	1,651,934
Probable producing territory, coastal area of Ecuador....	1,009,736	408,640
Total..	5,091,609	2,060,574

PRODUCTION

Ecuador possesses but one deep well now "producing." This well is on the property of the Anglo-Ecuatoriana (or Ancon Co.), near Ancon, on the south coast of the Santa Elena Peninsula, and was drilled to 2800 ft. (853 m.). It has produced from 37 to 40 bbl. per day. The three wells drilled by the Sackay & Harmsworth Co., west of Santa Elena, are reported to have been drilled to about 700 ft. These are reported to have produced oil, but the author understands that they are temporarily abandoned.

The main production of Ecuador is obtained from pits near Santa Elena. Mr. Price reports that this amounts to approximately 40,000 bbl. per annum, most of which is refined at the "Carolina" refinery, the product being sold in Guayaquil. From these figures, it seems that the Ancon well might produce about 13,000 bbl. per year; adding this to the pit production gives a total of 53,000 bbl. per annum.

Assuming that the extent of favorable, probable, and possible territory corresponds in a general way to the area of the coastal plain province, as recognized on the border of Peru, Ecuador may possess more oil-bearing territory than Peru.

There is not now enough information to justify a definite statement concerning the possibilities of the interior provinces. The surface indications are favorable in the interior of both Peru and Ecuador.

The fact that Ecuador does not now possess any developed oil land makes it impossible to establish a basis for calculating the future total production of the republic. By comparing the Zorritos field of Peru with the possibilities of the Ecuadorian coast, which is the northward continuation of the same geological province, the author estimates that promising territory on the coast of Ecuador should produce from 100,-000,000 to 150,000,000 bbl. Data at hand, however, lead one to believe that the coastal-plain province of the Santa Elena area may continue to the Colombia border. Parts of this province to the north, as at San Francisco, also are expected to show favorable conditions for the occurrence of oil.

Oil Resources of Peru

By V. F. MARSTERS, KANSAS CITY, Mo.

(New York Meeting, February, 1922)

PERU has produced petroleum since the early seventies, the first work being in the Zorritos field, in the Province of Tumbes, adjoining Ecuador.

In the early nineties, the Negritos field, in the Department of Piura, came into prominence, and by 1905 had attained an annual production of 335,160 bbl. The early development of the field was under the general direction of the London & Pacific Petroleum Co., but it is now handled by the International Petroleum Co.

In 1900 and 1901, the Lobitos field became the scene of marked activity. In 1905, its total production amounted to 75,000 bbl.; it has continued to increase, as has Negritos.

From 1897 to 1902, there was prospecting in the region of La Brieta (or Fernandez), on the Mancora River, about 25 to 30 km. from the coast. Small production of a heavy oil was obtained from three different sands. So far as the author knows, no work has been done in that locality since 1902. There are thus four distinct fields in the north end of the Peruvian coastal plain.

The remaining productive locality of Peru is the Pirin or Pusi field, at the northwest end of Lake Titicaca, Department of Puno, in southern Peru. The first prospecting near Pusi was done by Brown Brothers, of Los Angeles, Calif., who obtained modest production about 1906; according to reports, activities ceased about 1914 or 1915. The maximum annual production was obtained about 1908, estimated at 76,000 barrels.

Peru possesses two distinct geographic provinces in which we may look for the development of a larger oil industry; namely, the coastal plain province, and the inter-Andean plains.

COASTAL PLAIN PROVINCE

The coastal province is most prominently developed in the Province of Tumbes and the Departments of Piura, and Lambayeque.[1] It

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[1] For details concerning the geology of this part of Peru, see Bol. No. 50, Cuerpo de Ingenieros de Minas "Informe preliminar sobre la Zona Petrolifera del Norte del Peru," por V. F. Marsters.

continues north into Ecuador, and southward nearly to the north boundary of the Department of Libertad. Its maximum width is attained between Cerro del Yllesca on the coast and Morropon on the interior edge of the plain. A short distance south of River Cana, the foot-hills of the Andes reach the coast. The south half of the north sector occupies a broad embayment with a few outliers of the Andes dotting the shore line, while the north half represents the inner rim of the coastal plain extending into Ecuador. Farther southeast the foot-hills of the Andes reach the coast. Wherever the main transverse valleys are well developed, their lower ends are V-shaped embayments partly filled by tertiary sediments. A partly marine-filled embayment of this type is found on the Jequetepeque River, which may be designated as the Pacasmayo embayment. Below this are the Chicama and Moche valleys, likewise partly filled by marine sediments, which may be designated as the Chicama and Trujillo embayments.

Similar conditions occur, but on a much smaller scale, at Chimbote, Casma, and Huarmey. From Haucho to the south side of the Rimac embayment is a group of little valleys connected by a narrow coastal rim. Lima stands near the inner apex of the Rimac delta. Marine sediments occur in the Lima embayment; similar, but smaller, embayments are also found between the Rimac and the Chincha Rivers.

From Chincha to Atiquipa is the central sector of Peruvian coastal plain.[2] Along the shoreward edge, as well as in sub-central locations, there are pre-Tertiary formations projecting through Tertiary sands and clays. At Paracas there are well-defined Carboniferous strata, but the interior hills are made up of Cretaceous and older rocks. The part of this sector of special interest is that between the lower part of the Ica River on the northwest and Nasca and Porthachuela on the southeast.

The south sector of the Peruvian coastal plain extends from Chala southeast into Chile, beyond the Sama River. The outstanding geographic feature of the south sector is that the present coast is occupied mostly by a small coastal pre-Tertiary ridge. Between it and the foot-hills of the Andes is a plain varying in width from 20 to 60 kilometers.

Stratigraphy of Coastal Plain

The south sector shows considerable variation in stratigraphy. The south part, along the border of Chile, is made up largely of Tertiary

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[2] For details on central and south sector of the coastal plain see Bol. No. 59, "Dos Informes sobre Ho, Moguegrea ó Ica," and No. 70, "Informe sobre la Costa Sur del Peru," por. V. F. Marsters.

sands and clays. The central and north part, however, shows a mass of lava flows interbedded with sands and clays; this feature is particularly well shown in sections from Vitor to Mollendo. They also continue to the north.
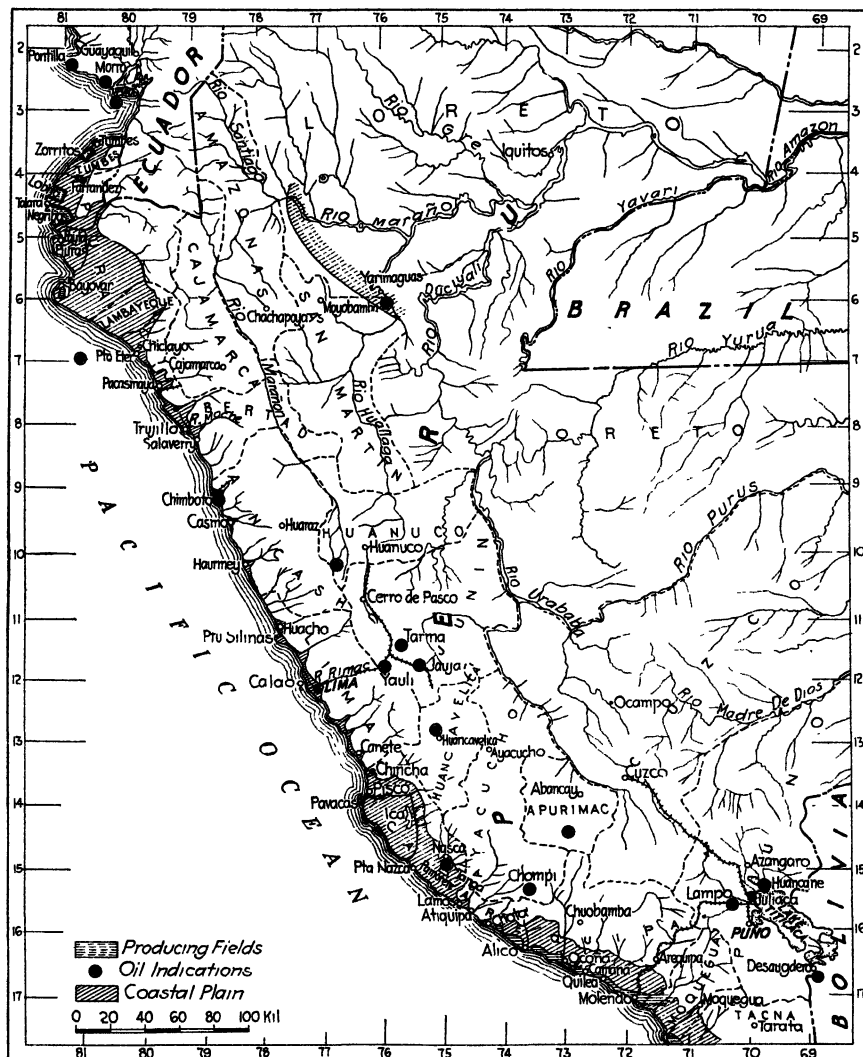
FIG. 1.—PRODUCING OIL FIELDS AND OTHER KNOWN PROSPECTS OF PERU.

The plains on the south end of this sector lie from 100 to 500 ft. (30–152 m.) above sea level, while the north end of the south sector, on its nner edge, rises more than 1000 ft. (305 m.) above sea level.

In the central sector, extending from Chincha on the north to Ati-

quipa on the south, the formations entering into the body of the coastal plain are entirely sedimentary and composed of sands, clays, and local conglomerates, which attain their greatest thickness between the lower part of the Ica River and the Grande River, and Pampa de Tunga on the south. Asphaltic sands are known to exist on the lower part of the Ica River, and likewise limestones in the Nasca-Portachuela were found to possess a strong petroleum odor upon fresh fracture. The limestones are doubtfully Cretaceous.
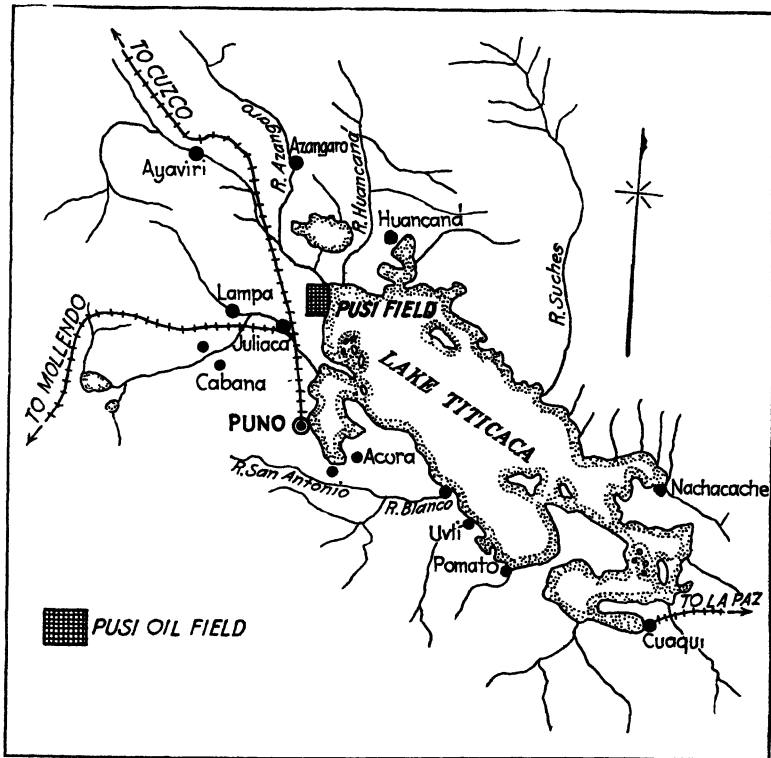
FIG. 2.—LAKE TITICACA REGION SHOWING LOCATION OF PUSI OIL FIELD.

Although the north half of the central sector is thought generally to possess a comparatively thin series of Tertiary strata, it is not improbable that a large body of sediments may be found in the region of Pampa de Chunchanga. The Pampa de Pisco is covered with a thin coat of Tertiary sediments. Two of the many embayments existing between Chincha and the south end of the north sector of the coastal plain that deserve reference are the Trujuillo-Chicama and the Pacasmayo embayments.

The greater part of the Moche River area (Trujillo) is covered by

fluviatile, post-Tertiary deposits; only on the outer edge may Tertiary clays be seen. The Trujillo area is connected by a narrow strip of coastal plain with the much larger Chicama embayment. This area also is almost completely covered by fluviatile post-Tertiary deposits, but Tertiary clays and sands are exposed at a few places on the shore line. Doubtless a considerable body of Tertiary sediments exists on the shore line of this embayment. One deep well has been drilled on the area for water, in which the occurrence of asphaltic sands was reported.

The Pacasmayo embayment is covered mainly by recent alluvium; in only a few places along the shore may Tertiary sediments be seen. The large number of pre-Tertiary outliers suggest that no great thickness of Tertiary sediments may be expected, except near the shore of the Pampa de Charcape.

North Coastal Sector[3]

The Pacasmayo embayment connects northward, by a very narrow strip of coastal plain, with the north sector, which is the only important oil-producing part of the coastal plain. This area begins to widen in the region of the Saña River and reaches its greatest width between Cerro del Yllesca and Morropon or Chulacanas. The Chira River lies on the north side of the main embayment. Facing the Cerros de Amotape the inner rim of a half-drowned coastal plain passes into Ecuador.

The north sector has the greatest development of Tertiary strata known in Peru. The upper member of the Tertiary is recognized on the plains behind the town of Pyta (or Paita) and consists of clays, sands, and lenses of conglomerate. These reappear in the table lands northeast of Lobitos, and likewise are recognized in the plains southwest of the valley of the Tumbes. The middle member of the Tertiary, composed largely of clays and sands, is exposed on all the sea cliffs from Paita to Zorritos. From an investigation of well logs, it is apparent that also a lower member of the Tertiary exists. The deepest well drilled thus far has reached a depth of 3900 ft. (1189 m.) and is still in formations regarded as Tertiary.

Oil Fields of North Coastal Plain

Zorritos Field

This is the most northern producing area. Drilling has been carried on along the coast for a distance of 10 km. and 3 km. inland; about 90 per cent. of the drilling is confined to the seacoast. In the northeast, exploration for oil was conducted by a French company in Quebrada Heath, but no commercial results were obtained, although oil sands were reported to have been encountered.

Favorable structural conditions may be easily detected in parts of the

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[3] See Bol. No. 50, *ibid.*, p. 2.

Zorritos field; but some of the prospecting done in this region seems not to have been guided by geological and structural conditions. The most recent developments are not fully known to the writer. At the end of 1915, 313 wells had been drilled in this field and 116 had been productive. There are three main producing sands, varying in depth from 194 to 1640 ft. (59–500 m.). The deepest well, drilled in 1915, reac ed 3020 ft. (920 m.).

During 1918, the Zorritos field is reported to have produced 76,190 bbl.; its maximum output, 107,000 bbl., was reached in 1910.

Lobitos Oil Field

The greater part of the present productive area is located near the coast, but production is obtained also farther inland. Favorable structural conditions can be detected easily in the developed area and in many other parts of the concession, as ascertained by the writer when making investigations of this region for the Peruvian Government. The early prospecting was done evidently without reference to geological conditions, and hence failed.

In 1917 the Lobitos field had about 143 producing wells. Four producing sands are known, at depths between 400 and 2500 ft. (122–762 m.). The deepest well reached a depth of 3435 ft. (1047 m.). The total production of 1918 was estimated at 639,098 barrels.

Negritos Oil Field

The Negritos field is located southwest of Lobitos and 10 to 12 km. south of Talara, the nearest shipping port. It occupies the largest single concession granted by the Peruvian Government, and also is the largest producing field in the Republic. The first oil wells were located close to the shore line, as at Zorritos. Exploration soon extended northeast toward Talara and likewise into the interior. This general region has several favorable structures.

At the beginning of 1918, the Negritos area had brought in a total of 695 producing wells, including those of the Lagunitas area, a block of ground developed by a subsidiary company. Not less than seven oil sands are known, ranging in depth from 300 to 3000 ft. (91–914 m.). The deepest well was 3900 ft. (1189 m.), and was probably still in Tertiary sediments.

Analysis of Oils from Fields Previously Mentioned

	PER CENT.
Gasoline	15 to 25
Kerosene	20 to 24
Lubricating oils	20 to 30
Asphaltic base.	
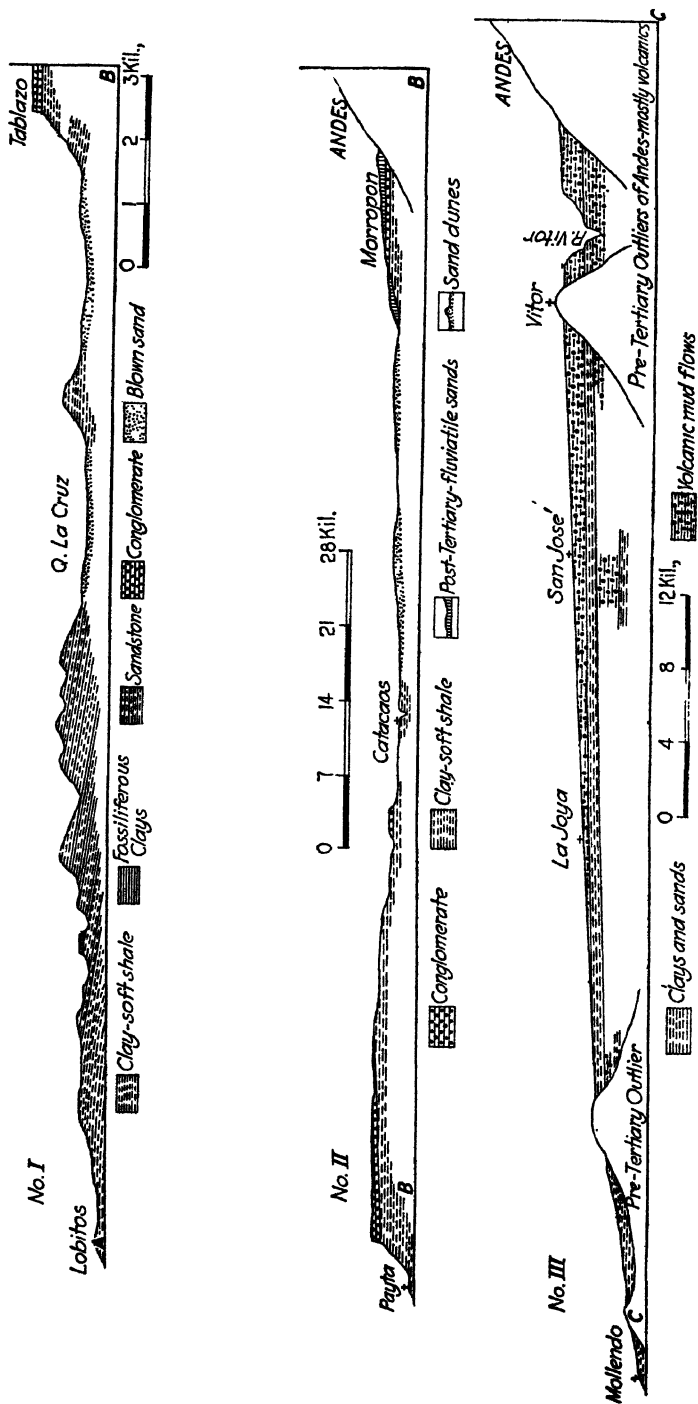
FIG. 3.—SECTIONS OF PERUVIAN OIL FIELDS. NO. I, LOBITOS TO TABLAZO; NO. II, PAYTA TO MORROPON; NO. III, MOLLENDO TO VITOR.

Pirin or Pusi Field

This field, formerly productive on a small scale, is located in the Department of Puno, southern Peru. It may be reached by railway from Mollendo to Juliaca, the junction of the railway from the coast with the Puno-Cuzco line. The oil field is about 25 or 30 km. northeast of Juliaca. According to published statistics, the field became productive about 1906, but activities ceased about 1915.

Following the northwest extension of the broken plains at the north end of Lake Titicaca, I found a series of Tertiary sediments resting upon formations probably of Cretaceous age. On the southwest side of the plain, in the Lake Titicaca lowland, both Tertiary and Cretaceous formations are folded and in some places faulted. The production of the Pusi field was obtained apparently in Tertiary sediments and down the longer slope of asymmetrical anticlines. Faulting is not prominent where production has been obtained. I have not investigated the entire area covered by Tertiary sediments, but enough is known about this general region to lead me to believe that Perin is not the only locality where we may expect to find production of at least modest proportions.

About 10 wells had been drilled when I visited this locality; the producing sands were found between 242 and 750 ft. (74–229 m.).

After the Lake Titicaca Oil Co. ceased operations, their holdings were acquired by the "Sociedad de Petroleos España," which drilled about 2 wells and cleaned out the old ones. Of the 10 wells drilled by the Lake Titicaca Oil Co., five were productive. The first 10 wells were drilled from 193 to 815 ft (59–248 m.); the production was obtained between 242 and 750 ft. (74–229 m.). The 12 wells drilled by the Sociedad de Petroleos España ranged in depth from 170 to 240 ft. (52–128 m.) and obtained a little production from 193 to 249 ft. (59–76 m.). The sand ranged from 4 to 13 ft. (1.2–3.9 m.) in thickness.[4]

The petroleum of the Pusi field is reported to have the following composition:

	PER CENT.
Gasoline	0 to 7
Kerosene	7 to 20
Lubricating oils, main part of the product.	
Paraffine base.	

GEOGRAPHICAL DISTRIBUTION OF OIL SEEPAGES

In the south part of Peru, seepages are known to occur in many directions from the Pirin or Pusi field. In the Department of Puno, they have been recognized near Huancané, to the east of Pirin; to the northwest in the region of Yaraviri; and to the west near Lampa and Maravillas. Much of the area between these points, as well as far to the northwest,

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[4] See tabulation of wells in "El Petroleo en El Peru," por R. A. Duestua (1921).

is occupied by Tertiary and Cretaceous formations. No detailed geological information has been published concerning this section of the inter-Andean plain, but personal investigations indicate that these formations extend across a part of the Department of Cuzco into the Department of Apurimac. This area seems worthy of investigation.

Seepages are known also to the southwest, near Desaguadero, and on the border between Peru and Bolivia. Similar occurrences are reported to exist to the southwest and within the limits of the high plains of Bolivia. Seepages occur in the Department of Apurimac, as well as in the Department of Cuzco, near Pillpata in the Province of Canas.[5]

In the high plains near the eastern edge of the western range, seepages have been recognized in the southwest corner of the Department of Ayacucho. In the Department of Junin, there are a number of localities where oil-saturated formations and seepages have been found. These center around Jauja, Yauli, and Tarma. Apparently, they are associated with Cretaceous formations. The structure is so complicated, that commercial production is not to be expected.

There are rather large seepages farther northwest, on the west edge of the high plains, in the southwest corner of the Department of Huanuco. Details of the geology are not available.

On the coastal plains of Peru there is some evidence of seepages at points remote from production. In the central sector, or the part extending from Chincha on the northwest to Atiquipa on the southeast, in the lower Ica valley, there is considerable evidence of asphaltic sands, exposed not far from the coast.

In the region of Nasca and Portachuelo there are oil-bearing limestones, probably Cretaceous. The structure is complicated and the region is not regarded as promising for oil. There is, however, a section of the coastal plain between Cerro Yungi and the ocean, occupied by Tertiary formations, where it is believed a large body of these sediments may be found. The same is true for the lower part of the Ica River region. No seepages are known on the south sector of the coastal plain. The north half of the south sector is made up of mud flows interbedded with sands and clay, and is believed to offer favorable suggestions for a test. The south half, adjacent to Chile, is composed, so far as known, entirely of sedimentary formations, largely sand and clay, and increases in width in passing into Chile, and very probably increases in thickness. The south half of the sector merits serious consideration.

Seepages of fair size are known, also, near Chimbote, but it is believed that the area covered by the coastal plain formations is too small to warrant serious consideration.

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[5] For additional information, see "Industria del Petroleo en El Peru durante 1915," por R. A. Duestua (Secunda congreso cientifico Pan Americano de Washington).

Seepages on the East Slope of the Andes[6]

There have long been reports of oil seepages on the east slope of the Andes, somewhere in the vicinity of Yarimaguas, or according to the official map of Peru, near the border of the Department of San Martin with the Department of Loreto, or on the Haullaga River. From the available information the belt of lowland adjacent to the foothills seems to be composed of some Tertiary and probably of a much larger group of Cretaceous sediments, which should be well worth investigating.

In the Yurimaguas region Duestua reports also the following oil seepages: (1) Quebrada de Chipeza and Chipauta, District of Chasuta, Province of San Martin, Department of San Martin; (2) Quebrada de Potoyne, probably a tributary of Rio Santiago, Province of Bongará, Department of Amazones; (3) Quebrada de Camaya, tributary to the Rio Maranon, District of Pecca, Province of Bongará, Department of Amazones; (4) Numerous places in the district of Barranca, Department of Loreta; (5) Quebrada de Ahuaya, tributary to Quebrada de Cashiboya running to the River Ucayali, District of Contamana, Province of Ucayali.

POSSIBLE, PROBABLE, AND FAVORABLE OIL-BEARING TERRITORY OF PERU

	ACRES
1. Area of the coastal plain of Peru considered as developed and in part proved, approximately.	30,875
2. Total promising area outside of the proved areas of the coastal plain, all of this being in the "north sector".	500,125
3. Total possible producing area of the coastal plain.	1,620,610
4. Total probable producing area of the coastal plain.	940,305
5. Total possible producing area of the Pirin region, Lake Titicaca, Department of Puno.	98,800
(The above figures do not take into consideration the territory outside of the general lowland at the northwest end of the Titicaca Lake. It should be stated that Tertiary and Cretaceous formations occupy an enormous area to the northwest and west of the Juliaca-Yaravari-Hauncane district.)	
6. Total coastal plain area, approximately.	3,091,915
Total interior area, Pirin region approximately.	98,800
Total area.	3,190,715

OIL SHALES

Apparently little attention has been paid to the possibilities of shale formations of Peru as a source of petroleum.

[6] Information on this subject may be found in a publication by R. A. Duestua, "Estado actual y Porvenir de la Industria Petrolifera en El Peru," Lima (1912). See also "El Petroleo en Peru," same author; Ministerio de Relaciones Exteriores (1921).

While investigating coal deposits in Peru, I learned something of the distribution of shales associated with them. It is my opinion that the shale deposits known to exist on the west slope of the Andes in the Department of Libertad and Cajamarca may contain oil-bearing members. I am informed that one section of the west range in the Department of Ancash likewise displays a very considerable thickness of carbonaceous shale.
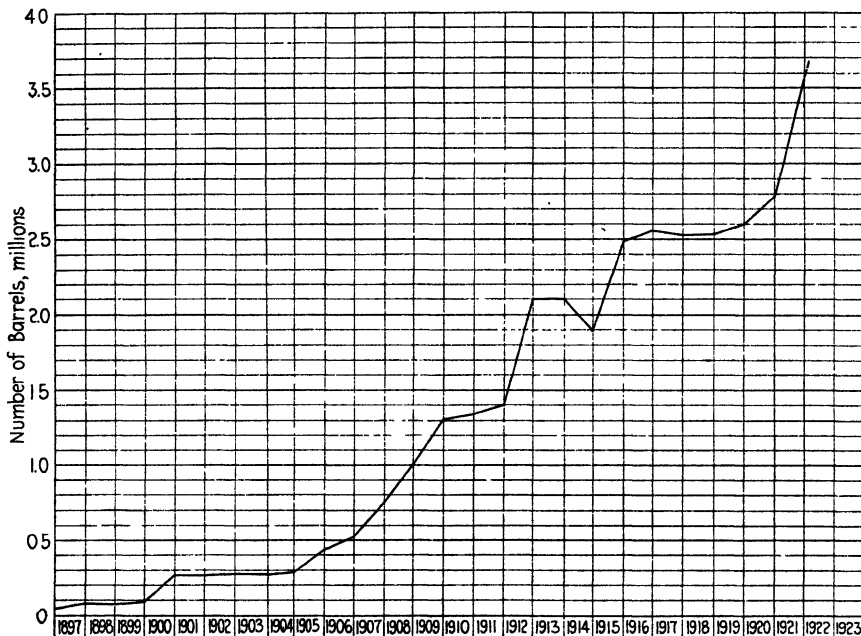
FIG. 4.—PETROLEUM PRODUCTION OF PERU FROM 1897 TO 1922.

In the Department of Junin, there are large bodies of shale in the vicinity of the coal mines operated by the Cerro de Pasco Mining Co.

In the Cuzco region there are very large bodies of shale, but in neither of the last two regions did I find any locality where it would seem that they might possibly be oil bearing. It is believed that the belt in the Department of Libertad may contain oil shale. The large shale deposits of Cerro de Amotape, Department of Piura, are too thoroughly metamorphosed to be of any importance.

FUTURE PRODUCTION OF PERU

An analysis of the rate of increase in the annual production of the Republic, considered in terms of new territory recognized as either probable or favorable ground for future development, should render some

assistance in formulating a fairly dependable opinion on possible future production.

Production of Petroleum in Peru, 1896–1922, in Barrels

YEAR	LOBITOS	NEGRITOS	ZORRITOS	PIRIN[1]	LAGUNITOS	TOTAL
1896						47,536
1897						70,831
1898						70,905
1899						89,165
1900						274,800
1901						274,800
1902						286,725
1903						278,092
1904						290,123
1905	75,000	335,160	37,720			447,880
1906	162,000	330,510	42,419	1,365		536,249
1907	279,000	396,750	65,476	15,000		756,226
1908	319,898	543,750	71,429	76,103		1,011,180
1909	421,195	740,070	70,750	76,103		1,316,118
1910	400,083	773,025	107,000	50,000		1,330,105
1911	391,290	882,698	94,048	30,000		1,398,036
1912	587,048	1,071,000	87,095	15,000		1,760,143
1913	557,355	1,136,490	83,343	10,000	346,073	2,133,261
1914	504,743	1,032,210	88,136	10,000	282,713	1,917,802
1915	664,972	1,355,925	72,736	1,000	392,618	2,487,251
1916	654,060	1,822,733	73,852		[2]	2,550,645
1917	686,595	1,771,560	75,262		[3]	2,533,417
1918	639,098	1,820,814	76,190		[2]	2,536,102
1919					[2]	2,616,000[3]
1920		· 1,993,331[5]				2,790,000[4]
1921						3,568,000[5]

[1] Estimated.
[2] Included in Negritos.
[3] Quoted from Richardson, U. S. G. S.
[4] Am. Petr. Inst.,
[5] *Petroleum Digest*, Jan. 25, 1922; *Tulsa World*, Jan. 21, 1921; *Wall St. Journal*, Jan., 1922.

Inspection of the production table brings out the following conclusions.

1. The Zorritos field showed its maximum production in 1910 with 107,000 bbl., falling by 1915 to 72,736 bbl. By 1918 production had increased to 76,000 bbl., which so far as I know has not been exceeded during the last three years. The production of the field seems to be declining, or at least to have passed its maximum.

2. The figures on the Pirin field would lead the casual observer to suspect that the area is practically condemned. On the other hand, detailed investigation will show that it can be made to produce, but geographical conditions render operation difficult.

. 3. As a rule, the annual production of the other fields has shown a marked increase. The greatest total annual increase. of production

appears in 1920 and 1921.   The peak of annual production of these fields has not yet been reached.

Calculations indicate on the north sector of the coastal plain about 500,125 acres (202,550 ha.) regarded as *promising ground*.   The area of about 30,875 acres (12,484 ha.) developed and proved is approximately but one-sixteenth of the total acreage which unmistakeably is worth testing, and some of which the author believes will prove to be commercially productive.   This statement does not include the probable producing acreage, which the author estimates may be 940,305· acres (380,823 ha.).

OIL PRODUCTION OF PERU WITH ESTIMATES ON POSSIBLE RESERVES

	BARRELS
Total production from 1896 to 1921...............................	30,176,355
Total production for 1920..	2,790,000
Total production for 1921..	3,568,000
Increase of total production for 1921..............................	778,000

Assuming an increasingly active drilling campaign, not only on the known extensions of producing structures, but also on other coastal areas where probably favorable conditions are believed to exist, it is estimated that not less than six years will be required to prove the areal extent of productive territory throughout the entire coast.

Adopting the above increment of production, placed at 770,000 bbl. per annum, Peru should be producing approximately 9,180,000 bbl. annually by 1928.   This is regarded as the peak of production for the Peruvian coast.   The total production of the Peruvian coast from 1921 to 1928 will approximate 33,818,415 bbl.; hence the total production of the coast from 1896 to 1928 would be............................... 63,994,770

After proving up the areal extent of producing territory of the coast it is estimated that the peak of total annual production can be held near the maximum for a period of from five to seven years.   Hence from 1928 to 1934 we would expect to obtain approximately...................... 54,000,000
Or a total production from the coast alone by 1934 of................ 117,994,770

As to the rate of decrease of production, after the peak has been passed, very little published data seems to be available, but such as we have on the question of permanent settled production points to the conclusion that we may expect the passage from the peak to settled production to occupy about two years.   On this basis the annual production of the coast by 1936 would be not less than 4,000,000 bbl.   Hence, at the close of 1916, the total production of the Peruvian coastal district will be approximately.. 128,769,777

The period of settled production is estimated to cover not less than ten years.   This period is estimated to make a total showing of 30,000,000 bbl.   The total production of the Peruvian coast is therefore estimated at.. 158,769,770

For the Pirin area (Lake Titicaca region) the small amount of exploration carried on has resulted in the total production of 284,571 bbl. between 1906 and 1915.   It is believed that this region possesses favorable structures and if developed with caution and economy will add not less than 30,000,000 bbl. to the total production of the Republic.   This makes a grand total of.. 188,769,770

The author believes that a vigorous drilling campaign on the coastal plain alone could bring the total annual production of Peru up to about 9,000,000 bbl. The production of Peru seems to be far below its possible maximum. In the last five years drilling has been confined to practically proven ground. The fact that the Peruvian Government has removed a portion of the coastal plain from denouncement and has placed it on the reserved list is, in part, responsible for the lack of increased activity in new localities, the development of which would be advantageous to Peru.

Using the available published statistics on production, the preceding summary seems reasonably fair. After making some calculations on the acreage basis, and knowing that from four to seven sands exist in certain fields, the author feels forced to admit that the above figures must be too low.

Applying the few facts at hand to the acreage regarded as probable to favorable territory, the author is confronted with the unsuspected and surprising conclusion that he must increase the final production from 3.5 to 5 times the total given above, or from 660,694,195 to 943,-848,850 barrels.

Attention is also called to the fact that the above estimates do not include the eastern interior belt, or the east slope of the Andes proper in the north part of Peru. Although no detailed work has been published on this great interior belt, significant observations are known that force us to foresee great possibilities for this region, which may be designated as the southwest corner and border of the Department of Loreto and portions of the Departments of San Martin and Amazones.

Petroleum Resources of Venezuela

By Ralph Arnold, Barnabas Bryan, New York, N. Y., and Geo. A. Macready, Los Angeles, Calif.

While much geological work and drilling have been done in Venezuela, the incompleteness of geological evidence obtainable and the restricted areas in which drilling has been done make any èstimates of oil resources extremely speculative. Nevertheless, it is felt that the data given are justified, in the belief that the ideas of a geologist familiar with the region are better than those of a promoter.

The oldest beds of interest to oil men are those of the Cretaceous, which are well developed in eastern Venezuela and about parts of Lake Maracaibo. In both cases, a hard massive limestone is overlain by a great thickness, most of which is shale. In the west, in places this shale is more than 2000 ft. (610 m.) thick, while in the east it may attain 3500 ft. (1067 m.). The limestone and parts of the shale are oil producing. In the east, the Tertiary where present is almost entirely covered by the Quarternary, but in the west it is well developed.

In probable unconformity over the Cretaceous shales is found a series of sandstones and shales, which may be called the lower coal series. It is well developed southwest of the lake, where it is the oil container, and is probably the lowest series exposed east of the lake.

Above this is the Pauji shale, which is probably the point of origin of all the oil east of Lake Maracaibo and in the State of Falcon. It has a thickness of about 3000 ft. (914 m.) in the Mene Grande region.

The upper coal series is developed as a great thickness of sands, shales, and clays, with some partly consolidated limestone. Its thickness may be as great as 10,000 ft. It is the oil container of the Falcon-El Mene region and a probable location for oil in some of the area about the lake. It may or may not include the so-called Maracaibo series in which the oil is found at Mene Grande.

Over much of Venezuela, the Quarternary is present in unknown thickness, which probably is not great. It obscures oil structure in all cases and presents the greatest hazard of oil exploration for the fields which probably exist below.

RELATIVE IMPORTANCE OF REGIONS

From the standpoint of oil production at a profit, the Lake Maracaibo region contains over 95 per cent. of the probabilities of Venezuela. Here the combined action of the major mountain ranges pinched the intervening area to produce the geosyncline of the lake and the surrounding wrinkles that contain the oil possibilities.

In the accompanying map, those areas shown in black have, through drilling, shown some production in what could be commercial quantities if located in the United States. The double-hatched area includes that part of the region which has the greatest possibilities of new discoveries and wherein some new fields will be found. The part hatched down to the east includes the area where further exploration and carefully considered drilling may be justified, but regarding which too optimistic an opinion should not be entertained. The part hatched down to the west is the area where there is some slight possibility of oil in non-commercial quantities, but where the present expenditure of development or exploration money is not justified. In the areas left blank, there are no oil possibilities.

In eastern Venezuela, the possibilities are entirely based on geology with only negative evidence from the drill. While work in the region has been done with great care and forethought, the most encouraging results have been the discovery of semi-liquid asphalt of no present commercial value. Nevertheless, the evidence is so great that the authors have no hesitancy in saying that oil in commercial quantities eventually will be discovered by the drill.

In the great region south of the mountains, there are evidences of oil and the probability of structure, but the difficulties of transportation and the numerous troubles and costs necessary to the development of oil will prevent these fields from producing at a profit for many years to come.

COSTS

An estimate of possible oil to be produced from Venezuela would be grossly misleading without some estimate of what that oil will cost per barrel. A number of people are today playing with fire by attempting to make money in Venezuela without sufficient resources in land or money. Costs of development are enormous, while to achieve success the investment must be carried for many years before the profits will appear. The best-managed companies have not recovered a gross return equal to their investment, after 10 years of development. Any work in Venezuela based on a return of less than $2.50 per barrel for the oil f.o.b. tanker is dangerous and probably unjustified.
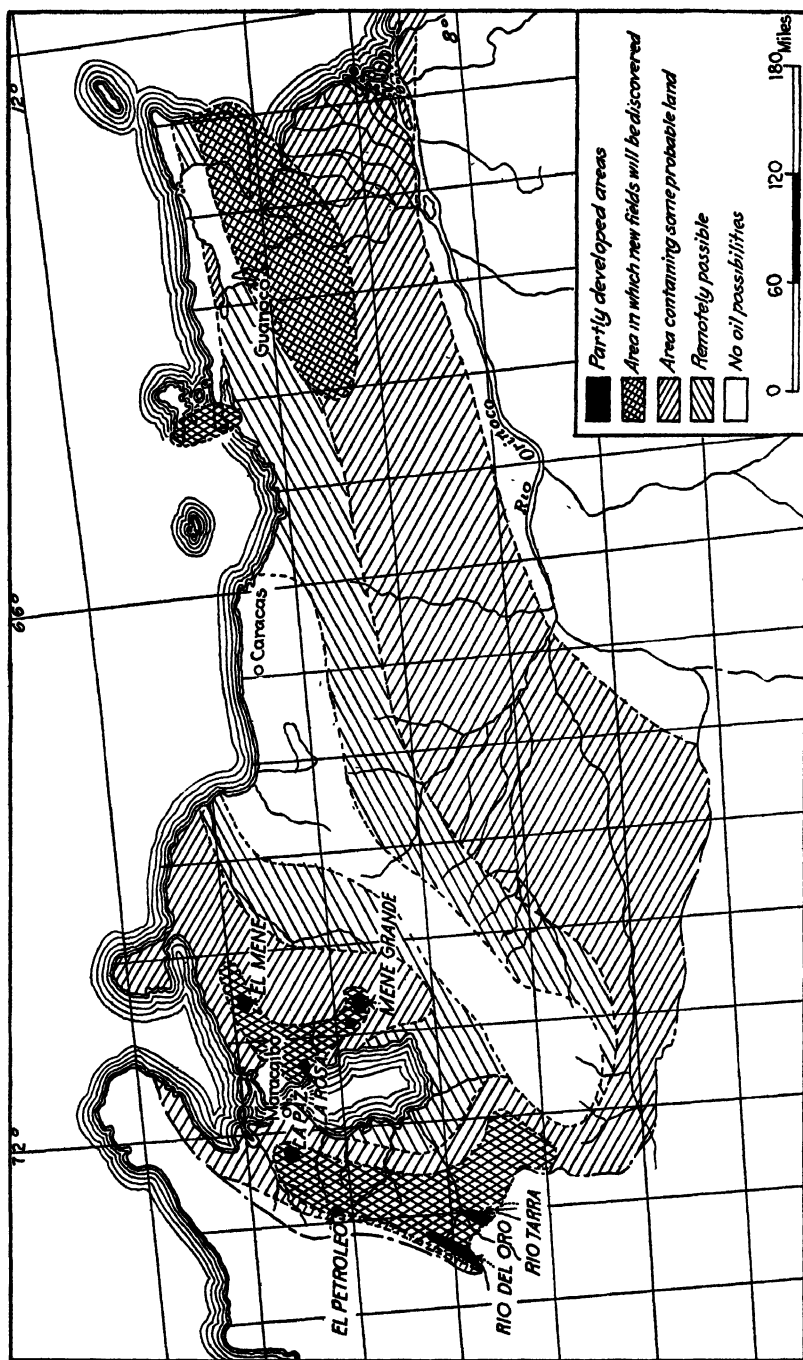
1.—Petroleum deposits of Venezuela.

TRINIDAD

The oil of Trinidad is found in true sands in the lower part of the Tertiary series, which is well developed in the southern half of the island. The structure is mostly anticlinal, although a few domes have developed oil. Nine fields have been developed by about five hundred wells, two hundred of which are approximately 3000 ft. deep. One of these fields produces oil of about 40° Baumé while the remainder give a grade running from 14° to 22°. The development area is about 5 sq. mi., or 3200 acres, which will produce about 6250 bbl. per acre. Thus the total production of the known fields will be about 20,000,000 barrels.

The total production from Trinidad through the year 1920 was 12,-000,000 bbl., leaving the probable production from the developed fields about 8,000,000 bbl. Possible increases in the probable production of known fields may somewhat increase these totals, while other developments in the southern half of the island are of course possible, but such possibilities are indefinite at present.

Referring to Trinidad, Hubert May, associated with A. Beeby Thompson of London, states:

Approximately, I should estimate the producing area at about 10 sq. mi. The possible area may be considered to be the whole of the southern part of the island, or say about 1000 sq. mi., but the probable area will be somewhere in the neighborhood of 200 sq. mi., extending northeast and southwest of Tabaquite: a region bounded on the north by a line connecting Brighton and Princess Town, the eastern and western boundaries being lines drawn from these to a point on the south coast, including the Point Fortin area to the west. A producing area will probably be developed within a region bounded on the east by the coast line, on the south by the coast line, on the north by the Ortoire area, and on the west by a north-south line through Rio Claro. This must naturally be considered approximate, but there is little doubt that production will be obtained from those areas extending from fields already producing.

THE GUIANAS

Due to the presence of igneous and metamorphic rocks, and the absence of the oil-forming series of Venezuela and that of the southern half of the continent, the Guianas may be eliminated from serious considerations for oil. The only possibilities are in the swamp area along the coast, where geological evidence is lacking.

PERSONAL OBSERVATIONS

With full knowledge that many men will take exception to these remarks, it may be stated that in the future less and less oil will be imported into the United States. The writers agree with the estimates of the "probable" oil reserves of the United States, but it is known that

these estimates do not include "possible" oil reserves as distinguished from probable oil reserves.   Certainly there will be discoveries that will add to reserves as there have been in the past.

As regards South America, it is the writers' opinion that many of us are gazing into a mirage.   After visiting a large part of the prospective territory of that continent and having some experience with the costs and difficulties of development, it is the belief of the writers that ten cents spent in increasing the percentage recovery of oil from known American fields will be more productive of profits than a dollar spent in the tropics and that new methods of recovery may double our probable reserves.   It is likewise their belief that shale oils produced in the United States can be laid down in the great markets from Chicago and St. Louis westward at less cost per barrel than oil produced from sand-storage fields in the tropics.

Oil Possibilities in Brazil

BY JOHN C. BRANNER,* STANFORD UNIVERSITY, CALIF.

FIVE of the geologic horizons that yield oil in other parts of the world are represented in Brazil; namely, the Devonian, Carboniferous, Permian, Cretaceous, and Tertiary. Thus far, the first two have shown no evidences of being oil bearing within Brazilian territory. Not enough exploring has been done to permit trustworthy comparisons of the relative importance of the Permian, Cretaceous, and Tertiary as oil-bearing horizons in this country. The most important, indeed almost the only, information available relates to the locations of the areas and to the structural features of the several horizons. The theories regarding the physical conditions under which these rocks were laid down, however, I regard of the utmost importance.

PERMIAN

Rocks of Permian age cover an enormous area in Brazil, extending from Rio Grande do Sul to Maranhão on the north, and to Matto Grosso on the west. These Permian rocks in many places are known to include oil-bearing shales and in some locations contain small veins of gilsonite. The geologic map of Brazil shows that the Permian area is widely distributed, but there is considerable doubt about the origin of some of the rocks. Some of the lower Permian beds in southern Brazil contain marine fossils, but in Minas no fossils have as yet been found in them, and in Bahia, at only one locality have a few Permian plants been found.

Indications of oil have been found in the states of São Paulo, Paraná, and elsewhere farther south, but wells drilled in the Permian of São Paulo near the Morro do Bofete did not find oil. One well has been started, by the Brazilian Government, in the Permian of Paraná near Marechal Mallet, close to latitude 26° south, but thus far oil has not been found.

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* President Emeritus, Stanford University. Doctor Branner died in Palo Alto, Calif., on March 1, 1922, shortly after writing this paper.

CRETACEOUS

Rocks of Cretaceous age are found in many Brazilian states, but while some of them are certainly of marine origin, others seem to be land and fresh-water deposits. Those best known are the marine beds in the states of Sergipe and Bahia, where they are more or less folded and exhibit what petroleum geologists consider favorable structure. In some places, especially at Marahú in Bahia, marine Cretaceous beds are accompanied and overlain by oil-bearing shales.

It is quite probable that the Cretaceous rocks are more widespread than the geologic map suggests, for the Tertiary beds, where they form rather wide zones along the coast, may have Cretaceous rocks beneath them, because at a few places where the Tertiary is well exposed on the seashore, to the landward the Cretaceous is the surface rock. This is the case at Parahiba and the island of Itamaracá, just north of Pernambuco. At Bahia, the map shows a considerable area of Cretaceous rocks extending northward to the Rio São Francisco. The age of these beds has been determined beyond question for the region about the bay of Bahia, but there is doubt as to whether the horizon continues to the Rio São Francisco. The rocks in this zone are folded, some of them standing at an angle of 45° or even more, and there are many evidences of faulting. The rocks of this particular belt, so far as they are exposed, are shales and sandstones. In the state of Sergipe, some of the Cretaceous beds are highly fossiliferous limestones of marine origin; it is quite probable that these Cretaceous rocks extend into the southwestern corner of Alagoas, where they are overlain by the Tertiary sediments. In Maranhão, also, the Cretaceous seems to underlie a large area in the eastern part of the coastal belt.

I know of no test wells having been drilled in the Cretaceous rocks in any part of Brazil. There has been talk of drilling such wells, but if any have been sunk, it is to be assumed that they were unsuccessful, as the results have not been reported.

TERTIARY

The chief Tertiary area of Brazil lies along the eastern base of the Andes, covering almost all of the Acre territory, the greater part of Amazonas, and forming broad belts across Pará on both sides of the Amazon river. This area is narrow and patchy along the rest of the east coast of Brazil from Maranhão, at the north, to Rio de Janeiro, on the south. In some places the coastal Tertiary beds are known to have Cretaceous rocks beneath them, but for the most part they rest upon Archean granites and gneisses. They are folded in some places but, for the most part, are nearly horizontal. The rocks are sandstones, shales, and conglomerates, and are greatly weathered and highly colored where

they are exposed along streams and on seashores. They locally contain marine or brackish-water fossils, and in places the shales are petroliferous. Here and there, through the interior, are fresh-water deposits of Tertiary age, evidently laid down in lakes. The rocks of these fresh-water deposits are mostly shales containing the remains of fresh-water fishes and plant impressions, and yielding considerable oil upon distillation.

Beginning with the less important of these Tertiary areas, the fresh-water deposits contain bituminous shales that have been used for the manufacture of gas at Taubaté, in São Paulo, though I understand that this use has been stopped. The areas covered by these fresh-water deposits are probably too small to offer reasonable promise of yielding petroleum.

For many years, attention has been called to the broken and irregular Tertiary coastal belt, which extends with interruptions from Cape Frio, near Rio de Janeiro, to Maranhão and beyond as a possible source of oil. It seems quite possible that this zone may contain petroleum where it widens out and extends well into the interior, as at Bahia for 300 miles, at Mossoró in the state of Rio Grande do Norte, and at Maranhão; but elsewhere, I doubt its existence because this horizon is too narrow, too fragmentary and too thin to furnish collecting grounds for much petroleum.

In the great Tertiary area of the Upper Amazon region, we are certainly not limited for room, for within Brazilian territory alone the Tertiary area, in Acre and Amazonas, covers more than 1,000,000 sq. km. The thickness of these deposits, except along the margins, is unknown. At many places they contain beds of lignite. Regarding this great area as an oil field we have but little more than opinions and theories. In 1908, I. C. White surmised that if large deposits of petroleum were found in Brazil they would "be located in the territory drained by the great Amazon;"[1] but he does not say why he reached this conclusion. In 1920, Horace E. Williams, of the Geological Survey of Brazil, said that this same region "and the Peruvian frontier . . . seem to be the most promising field for explorations of any in the country."[2] What these opinions are based upon I do not know, but presume that it is inferred that the oil regions known to exist in Argentina, Bolivia, Venezuela, and Colombia are probably more or less continuous along the eastern base of the Andes and across western Brazil. I am disposed to attach considerable importance to the fact that fossils found along the western frontier of that Tertiary region are brackish-water forms that mark probably the period following the elevation of the Andes. It seems probable also

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[1] "Coal Measures of South Brazil," 247. Report of Chief of Coal Commission of Brazil. Published by Brazilian Government in 1908.

[2] *Trans.* (1921) 65, 69.

that the drainage of the upper Amazon region formerly flowed westward into the Pacific, and consequently that the sediments would accumulate toward the western end of the area drained. When the mountains rose only the brackish-water forms of life survived for a while.

I know of only one attempt to drill an oil well in the Tertiary area of Brazil and this was made in the state of Rio de Janeiro near Campos; it is said to have reached a depth of 30 m. without finding oil.

Attention should be called to the great areas, amounting to more than 1,000,000 sq. km., of which the geology is entirely unknown. What promise these unexplored areas may hold out to the petroleum geologist it is impossible to predict.

Oil Laws of Latin America

By Edward Schuster and Frank Feuille, Jr., New York, N. Y.

(New York Meeting, February, 1922)

As the time allotted is short, we can present only a general idea of the oil laws in the Latin-American republics, as a supplement to Bulletin 206 of the Department of the Interior compiled under the direction of J. W. Thompson of the Bureau of Mines and entitled "Petroleum Laws of All America." We shall not discuss the European colonies in Latin-America nor the island republics. As to the countries discussed, we shall attempt to give the main features of such special petroleum legislation as has been adopted prior to the early part of this year; where no special petroleum laws exist, we must look to the mining codes for light on the petroleum policy.

As the laws and regulations for the extraction of the metals are often difficult of application to, and are lacking in many essentials for, the petroleum industry, it is best for persons or companies desiring to enter, on a large scale, into the petroleum exploration and development of countries not having special petroleum legislation, to obtain a concession contract from the executive of the country under consideration, and to have the concession approved by the congress. If this concession, which would then be a special law, contains nothing contrary to the constitution of that country and does not infringe on vested rights of third parties, it should be reasonably safe. If in such country there is some doubt as to whether the petroleum subsoil is owned by the nation or the private surface owner, only public lands should be covered with the concession.

Guatemala is the only republic in the group that discriminates directly against foreigners. Bolivia, Ecuador, Mexico and Venezuela require waivers of diplomatic protection.[1] Colombia requires foreigners to agree to be bound by its alien law. In addition, Mexico makes it unlawful for foreigners to acquire direct ownership of real estate, including mines, within a zone of 50 km. along the coasts and one of 100 km. along its land

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[1] The U. S. State Department has consistently held that such waivers do not fore-close the sovereign right of the American Government to protect its nationals, in a case of confiscation or denial of justice; hence these waivers have little practical value.

frontiers; and Peru prescribes a like prohibition as to properties within
50 km. of its boundaries. Most of the Latin-American republics have
provisions in their laws that would tend to prevent foreign governments
from acquiring mining and petroleum rights, either directly or indirectly.
In some of them, local companies must be organized; but the expense
thus entailed is not great and the affairs of a subsidiary are easily con-
trolled by the parent company. Under the international precedents,
the stockholders of such local companies are entitled to diplomatic
interposition, in proper case.

All of the Latin-American countries have written constitutions,
largely modeled on our federal constitution. Their laws are contained
mainly in a series of codes, the basic code for property rights in general
being the civil code; for commercial matters, the mercantile code; and for
mining matters, the mining code.

The question of federal versus state rights enters very little into the
problems of foreign-oil companies operating in Latin-America, for all of
the republics enjoy the unitary form of government, except Argentine,
Brazil, Mexico, and Venezuela, which are federations. In these four
republics, mining legislation is reserved to the federal government, so that
questions arising with local authorities are mainly those incidental to
business enterprise in general, rather than to the specific business of
petroleum mining. In Argentine and Brazil, the several states own the
mines that lie under the surface of their public lands and thus control
the disposition of the petroleum subsoil as to such lands.

As all petroleum legislation and policy take, as the point of departure,
the mining laws and policy in general and as, even in countries having a
special petroleum legislation, questions not solvable by that special
legislation are resolved by reference to the provisions of the respective
mining codes, it is advisable to outline the sources of Latin-American
mining legislation.

While the ideas of Mirabeau, as given effect in the French mining
code of 1810, have had some influence on the mining legislation of the
Latin-American countries, the basis of Latin-American mining law has
continued to be the legislation decreed by the Spanish crown for the Ameri-
can colonies, from the time of their discovery until their emancipation
early in the nineteenth century.[2] The early Spanish legislation consisted
of Royal decrees or ordinances; the earliest issued after the discovery of
the Americas being Philip II's Ordinance of 1559.[3] This degree incor-
porated in the Crown, mines of gold, silver and quicksilver. Then
followed the Ordinances of 1563;[4] the "Ordenanzas del Nuevo Cuaderno"

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[2] We except the Portuguese crown colony which eventually became the Republic
of Brazil, as to whose legislation we shall refer later.

[3] Novisima Recopilación (1803), Book ix, title xviii, law iii.

[4] Book vi, Title xiii, Law v—Nueva Recopilación.

which date from 1584; the "Ordenanzas del Perú" of the year 1654, which were mining laws and regulations framed especially for Peru; the "Recopilación de Indias," which was promulgated in 1680 for all the colonies and constituted a general code of laws therefor; the "Ordenanzas Mineras de Nueva España," prepared for Mexico in 1783 but extended in 1785 to Chile and Peru, and eventually to practically all of Central and South America. In 1789, Charles III issued a law[5] around which the arguments as to the governmental, as against the private ownership of the petroleum subsoil, have surged in Mexico, Colombia, Panama, and Costa Rica; this law declared that coal was not a metal or a semi-metal and belonged to the owner of the surface. There were, of course, other special laws and decrees issued during the colonial period; but at the time that the Spanish colonies revolted, it may be broadly stated that the Mining Ordinances of 1783 provided the mining law for all Spanish-America.

THEORIES AS TO MINING PROPERTY; AND THOSE PREVAILING IN LATIN-AMERICA

There are four theories and practices in the world as regards the subsoil:

1. The theory of occupation or appropriation, which considers mines to be *res nullius*, until reduced to ownership by discovery and appropriation; this was the policy premised in the mining laws of the state of California beginning in 1848. It does not prove successful in practice.

2. The doctrine of accession, whereby the surface owner's title includes the subsoil. This was the Roman doctrine and is the English and American doctrine. Under it, the mining industry is controlled by individual enterprise; and it is this liberty of individual initiative that has made the United States the leading mining nation of the world.

3. The pure regalian or royalty theory, under which the government owns the subsoil and exploits it directly or under contracts made with individuals, who cannot acquire anything but working rights, private property in mines not being recognized. This is the Turkish doctrine.

4. A mixed theory of royalty and appropriation, under which denouncements of the subsoil and issuance of patents carrying a property title are allowed, by permission of the government, and the payment of annual rentals or percentages in the product to the government. The mining ordinances of the Spanish crown issued for the Americas, and also the French law of 1810, applied this mixed theory.

After they achieved independence from Spain, the Latin-American republics nevertheless continued to apply the Spanish mining legislation, either in practice or by express decree. An example of express re-enactment was Bolivar's decree of 1829, which declared the Spanish laws to be in effect

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[5] Nov. Recop., Book ix, title xx, Law ii.

in the Federation of New Granada, which then included Colombia, Panama, Venezuela and Ecuador.

Whether the mineral subsoil, other than that of metals and semi-metals, belonged to the Spanish crown and, therefore, was inherited by the several governments of the Latin-American republics, is a much disputed question. The Federal Supreme Court of Mexico, in 1882, rendered two opinions by a divided court to the effect that such subsoil did belong to the crown. It reached this conclusion as an interpretation of the effect of the Spanish mining legislation in force in Mexico and went on to admit that if the congress should expressly legislate on the subject, the court would respect a grant of the mineralized subsoil to the surface owners. Such an express grant of the petroleum and coal in the subsoil was made by the Mexican mining code of 1884 (article 2).

As regards the ownership of the petroleum subsoil, the present tendency of the Latin-American countries is such that the courts would sustain the government's claim thereto, unless the legislatures have expressly granted the subsoil to the surface owners, as has occurred in Brazil, Colombia, Mexico and Panama. It is to the credit of Colombia and Panama that their supreme courts have had the courage to halt the later attempts of their legislative or executive departments to confiscate the surface owners' rights, after the commercial value of the petroleum deposits came to be realized. But in Mexico, the present administration has not abandoned the program of confiscation that First Chief Carranza initiated in article 27 of the present Mexican constitution.

In Chile, Costa Rica, El Salvador, Honduras, and Guatemala, the ownership of the petroleum subsoil is open to question, but we are inclined to the conclusion that the petroleum subsoil should be construed as belonging to the owner of the surface. In Argentine, Bolivia, Ecuador, Nicaragua, Paraguay, Peru, Uruguay, and Venezuela, the question would undoubtedly be decided in favor of the nation. In Mexico, Brazil, Colombia, and Panama, it unquestionably belongs to the owners of the surface who acquired prior to certain dates.

In this connection, we should note that there exists a general tendency in the Latin-American countries, as regards their petroleum legislation, to substitute for the mixed doctrine of royalty and appropriation, inherited from the Spaniards, the Turkish or regalian theory; this tendency is a retrogression. The Turk has never been noted for contributions to the advancement of civilization.

COUNTRIES THAT HAVE NO SPECIAL PETROLEUM LEGISLATION

Brazil

Brazil was a Portuguese colony and under the laws of Portugal the mineral subsoil was reserved to the Crown. During the period that Brazil was an independent empire, a like principle prevailed; a law of

1871 declared the petroleum subsoil to belong to the state. But under the constitution of the Republic, promulgated in 1891, mines belong to the owners of the surface, subject to regulation by the Federal Government; this constitution revoked the law of 1871. As yet, Brazil has no special petroleum legislation, and such exploitation and development as are carried out must be effected under the mining code. Under the provisions of the mining decree of January, 1921, alienations of public lands will not include subsoil rights unless expressly so stated.

Chile

Chile is not vitally interested in petroleum. From its declaration of independence in 1811 until 1874, Chile operated under the Spanish mining legislation; in the latter year it issued its first mining code. Paragraph 2 of article 2 of this code states that the exploitation of coal and other fossils not included in the first paragraph passes to the owner of the surface who can exploit them by fulfilling the requirements of the mining law. In 1888, this code was replaced by the code now in force. Should petroleum in commercial quantities be discovered in Chile, the past policy of that republic would indicate that its laws to cover the industry would be reasonable.

El Salvador

A law of April 18, 1918, repealed the mining decree of 1899 and declared that the Government owns all the mineral subsoil, including petroleum. At present petroleum exploitation contracts can be obtained only through the Compañia Salvadoreña de Perforaciones, a company with Salvadoran, Italian, French, and Austrian stockholders, which enjoys a concession covering the entire republic. This company has found no petroleum.

Honduras

The mining code of 1895 reserves mines to the state but does not mention petroleum. The earlier code of 1880 reserved coal to the owner of the surface, but the code of 1895 re-nationalized it. As there is no special petroleum law, the Honduran executive and legislative departments are executing and approving concessions for petroleum exploitation, based on the code of 1895. "The Petroleum Laws of All America," contains a translation of one of these concessions.

The most important concession was granted to English interests in April, 1920. It is for exploration, with preferential rights to 5000 ha. for each well brought in; the tracts to be in any shape and in any direction, provided that the well be included. Authority is granted to drill and to build pipe lines, docks, tanks, etc. Machinery and goods destined for use of persons employed by concessionaries are admitted free of duty. Concessionaries have the right to the gratuitous use of the public lands,

timber, and materials thereon for the erection of buildings and the installation of machinery, and also to expropriate private lands where necessary. Hondurans are to be employed in preference to any other nationality except English. Drilling must begin within 1 year after Congress approves the contract and 10,000 meters must be drilled in 5 years, completing at least one well. The concession is for 50 years, and the consideration stipulated is $50,000 per year for the first 5 years, together with a 10 per cent. royalty in cash or petroleum at the option of the Government. The concession is freely transferable to other ndividuals or commercial companies, but not to foreign governments.

Paraguay

Paraguay operates under the mining laws of 1878 and 1914; the state claims the minerals. Concessions are for not to exceed ten claims per person. There is no petroleum legislation and, so far as we have learned, no important concessions have been let. As the present mining legislation requires work to begin in 5 months, no great development of the petroleum industry can be expected in Paraguay, except under special concessions entered into with the executive and approved by the congress.

Uruguay

The Uruguayan constitution of 1829 declared, in article 148, all previous laws not in conflict with the constitution and laws to be passed by the congress, to be in force, thus continuing the Spanish legislation. The mining code of 1884 reserved mines to the state. There is no petroleum legislation. Concessions under the mining law cannot exceed 180 ha. per individual, nor 324 ha. per company of three or more persons. As in the case of Paraguay, the petroleum resources could not be developed in the present state of its legislation, without special concessions from the executive approved by the congress.

COUNTRIES IN WHICH THERE HAS BEEN PETROLEUM LEGISLATION BUT IN WHICH A SPECIAL PETROLEUM LAW HAS NOT BEEN ADOPTED

Argentine

By article 108 of the constitution of 1860, only the Federal Government is competent to enact mining legislation and the mining code of 1887 reserves petroleum to the nation. Private property in mines is established by legal concession. The mining industry is declared a public utility. Only three pertenencias, of 81 ha. each, constitute a working petroleum claim under the mining law but liberal interpretation of the law by the mining authorities has increased this number to seven in the case of companies. When, in 1907, petroleum was discovered in commercial quantities, the Argentine had no petroleum legislation but allowed applications for petroleum claims under the mining laws. However,

in 1910, the Government took advantage of the provisions of its general land laws, to prohibit denouncements or concessions covering a radius of five kilometric leagues from the center of Comodoro Rivadavia, only authorizing denouncements under the mining code, outside of that area. Later it reserved 5000 ha. in this zone for 5 years, which reservation was renewed in 1915. The rest of the Comodoro Rivadavia zone was divided into parcels of 625 ha., for adjudication to private parties. As these resulted in speculation and no development, in 1913 the Government forbade denouncements in the region of Comodoro Rivadavia, comprised between the parallel of Pico Salamanca on the north and parallel 46° on the south, and a north-and-south line running 30 km. to the west of Comodoro Rivadavia. All previous concessions on which work had not begun were revoked. This decree was stated, in 1914, not to include private lands lying within the prohibited zone. A petroleum law is now pending in the Argentine Congress.

Cost Rica

The petroleum subsoil question in Costa Rica is of uncertain solution. The civil code of 1841 contains a pronounced doctrine of accession; and under the provisions of the mining code of 1868 it would seem that the nation had parted with its petroleum subsoil in favor of the owner of the surface. But a law of 1913 attempts to put petroleum subsoil under the dominion of the nation. Concessions under this law have been granted, the most notable being issued during the Tinoco regime from 1917 to 1919 and being owned now by American and English interests. The Tinoco government was not recognized by the United States Government and the Costa Rican Congress recently declared illegal all acts of Senor Tinoco between Jan. 27, 1917, and Sept. 2, 1919, including the constitution of June, 1917, and the concessions let by him during that time. A petroleum bill was introduced at the last session of the Costa Rican Congress, but it is probable that only court decisions can straighten out the situation.

Guatemala

The mining code of June 30, 1908, reserved certain minerals to the nation but did not mention petroleum. The decree of Dec. 10, 1915, known as Decree No. 722, attempted to reserve the petroleum subsoil and stated that it did so, as petroleum was not reserved under the mining code of 1908. Under this statement, it would seem that the petroleum subsoil should be construed as belonging to the owner of the surface except as to public land alienated by the nation after Dec. 10, 1915. Decree No. 722 also provided that the Secretary of Public Works and the

President might execute petroleum leases to citizens of Guatemala for a period not to exceed 10 years; these lease contracts could be transferred only to Guatemalans. (We understand that decree No. 722 has been repealed but have not a copy of the repealing decree.)

Mexico

The original basis of Mexican mining legislation was the "Ordenanzas de Nueva España" of 1783, which were promulgated especially for Mexico. After Mexico gained its independence, the mining situation became more and more uncertain and chaotic until, in 1883, an amendment to the federal constitution of 1857 gave the Federal Government the right to enact mining legislation. Under this authority, the mining code of 1884 was adopted; it expressly granted the petroleum subsoil to the owner of the surface and this grant was confirmed by the codes of 1892 and 1909. In 1901, a law was enacted authorizing the executive to grant concessions for the petroleum exploitation of lands and waters under federal jurisdiction. In late years, the attempt has been made, by article 27 of the constitution of 1917, to wipe out the rights of the surface owner to the subsoil granted him by the mining codes of 1884, 1892, and 1909.

As the Mexican Congress passed no petroleum legislation based on the new constitutional article 27, President Carranza, in the illegal exercise of purely fiscal powers conferred on him by the congress, issued a series of decrees in 1918 purporting to put into effect the petroleum paragraph of this constitutional article. These decrees are considered illegal by members of the Mexican bar in general, except the official element. Nevertheless, the Federal Supreme Court of Mexico decided recently in the Texas Company "amparo" case that they are legal.[6] The present government holds to this confiscation of the petroleum subsoil of the surface owners and attempts to justify its course on fallacious theories of rights in expectancy and retroactivity. To the owner of the subsoil it is immaterial whether the Mexican Government declares this article to be retroactive or not, so long as in practice and based on a sophistical theory of vested rights it applies the article retroactively.

A petroleum bill is now pending before the Mexican Congress but it also proceeds on a palpably false theory of vested rights and offers little promise as a solution of the unsatisfactory situation that now exists. The acquisition of petroleum subsoil rights is at a standstill in Mexico as notaries are forbidden to execute contracts covering petroleum subsoil and the registrars of deeds are forbidden to record them should they be authorized.

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[6] An analysis of this decision and a discussion of its bearing on the Mexican petroleum controversy will be found in a paper entitled "The Texas Company's Amparo Case," published in the *American Bar Association Journal*, November, 1921.

COUNTRIES THAT HAVE ADOPTED SPECIAL LEGISLATION

Bolivia

Bolivia, by a decree of 1872, expressly reserved the petroleum subsoil to the nation and its petroleum laws of 1916 and 1921 reiterated this principle.   The law of June 20, 1921, which is at present in force, provides that the petroleum subsoil can be explored and exploited only by the executive power, either directly or in association with individuals or companies, under concessions granted for not to exceed 55 years, over an area not to exceed 100,000 ha., and for a minimum royalty of 11 per cent. of the gross product, payable in petroleum or cash at the option of the Government.   Payments in cash are to be determined on the average price for the preceding quarter.   There is a surface tax of from 10 cents Bolivian money per hectare for the first year, up to 50 cents in the seventh year, on partnership contracts with the Government.   On other petroleum concessions, the tax is slightly less and over a longer period of time, but with a 30 per cent. surtax, which goes into effect in 1927, on holdings over 100,000 hectares.

Exploration permits are granted for not to exceed 4 years and cover 300,000 ha., on the payment of 2½ cents Bolivian money per hectare per year, and a deposit of 10 cents for each "pertenencia."   The unit of measure is the hectare, concessions being granted in the form of a rectangle, the sides to be in a proportion not to exceed 5 to 1; and the shorter side must follow rivers and lakes when the concession is bounded by such water.   The claims are to be continuous and without breaks.   Explorers are to have the preferential right, during the 6 months following the completion of the explorations, for a concession covering the property explored.

Concessionaries are required to put up a bond of 250 bolivianos for each 1000 ha. at the time the contract is signed.   They are required to drill one well for each 50,000 ha., to a depth of 500 meters, during the first 5 years, and one well for each 10,000 ha. during the next 8 years. They must survey the area contracted for and submit plans for locations of wells, buildings, installations of machinery and equipment, and every 6 months submit reports showing the status of development of the enterprise.   The concession carries with it the right to engage in the petroleum industry in all its phases; to occupy national lands free, and to expropriate private lands on payment of damages; to install telegraph and wireless systems with the permission of the Government, the latter to use them free of charge; to construct and acquire railroads, canals, docks; to import necessary tools and machinery free of duty; to use the pipe lines of others if their capacity will admit, under a tariff approved by the Government, pipe lines being declared public utilities.   The Government may require pipe lines and refineries to handle its petroleum in proportion to

the capacity of the pipe line or refinery, the Government paying the cost thereof.

The Supreme Court has original jurisdiction over all differences arising between the concessionaries and the Government and the former must keep a representative at the Bolivian capital fully authorized to act for them in all matters arising between them and the Bolivian Government. In addition to the petroleum taxes mentioned, there is a profits tax to which oil companies and individuals are subject.

Colombia

Under the constitution of 1886, Colombia took over all lands and mines belonging to the several states that had formed its union prior thereto, without prejudice to rights previously acquired by individuals. Under the decree of June 20, 1919, the petroleum in privately owned lands was subject to denouncement; but in the same year the Supreme Court held that the Government, under the constitution of 1886, had acquired only the mining rights held by the states; that these had reserved only mines of gold, silver, platinum, and precious stones and these reservations had been ratified by the mining law of 1887; that petroleum had not been included in these reservations; and hence, that the petroleum subsoil of privately owned lands acquired prior to the law of Oct. 28, 1873, was not subject to denouncement.

In view of this decision, the Colombian Congress passed Law 120 of Dec. 30, 1919, which div des the country into three zones: the first being 200 km. or less from the sea; the second, 200 to 400 km. from the sea; the third, more than 400 km. from the sea. Contracts covering national lands in these zones must pay, respectively, minimum royalties of 10, 8, or 6 per cent. This law further provides that owners of private lands containing known petroleum deposits must exploit them in 20 years, or pay rental to the Government; and that holders of lands ceded or adjudged unclaimed later than Oct. 28, 1873, and before the taking effect of fiscal Law 30, 1903, shall receive preference in the granting of petroleum contracts, all conditions being equal, for 2 years from the date of the law (note that this preference has expired). Holders of lands prior to Oct. 28, 1873, may lease them, but these lease contracts are subject to operation taxes of 8, 6, or 4 per cent. according to the zone. Permits for exploitation of public lands are granted on parcels of not less than 1000 nor more than 5000 ha. Applications for the same must state the location of the deposits; define the zone of exploitation and attach topographical and geological maps; state the names, nationality, and interest of each would-be lessee; in case of companies, give the name and address of their legal representative; if foreigners, agree to abide by the provisions of the law covering aliens; submit to all the provisions of the law, especially as regards supervision, taxes, right of lease, grounds of

forfeiture; and also express the amount of rent applicant is willing to pay. Such applications must be published in the *Official Daily* and are not acted upon for 90 days, it being necessary for them to be passed upon by the Ministry of Public Works, the Board of Finance, and the Council of Ministers. The successful applicant has 3 months, after his application is favorably acted upon, within which to sign the lease contract. The lease is for 20 years, with the privilege of a 10-year extension, and must be finally approved by the President, unless it covers territorial waters, lakes and navigable rivers, in which case it must be approved by Congress also. No company or person can secure a lease by direct contract, concession or transfer, of more than three zones of 5000 ha. each in the same department, intendencia, or comisaria. The successful applicant must also file a bond for not less than 20,000 Colombian pesos and must agree that the Government may fix the minimum production and supervise the exploitation of his wells; that he will employ not less than 50 per cent. of Colombian laborers; that he will not transfer the contract without the Government's approval; and that at the expiration of the contract and in any one of the cases of forfeiture specified in the law, the nation will acquire, free of charge, the ownership of the machinery and all other improvements and structures that may be found within the concession. If the deposits are not continuously exploited within the terms of the law, after 5 years from the date of the concession or when the working of the deposit shall have been suspended for a whole year, the lease shall be subject to forfeiture, notice being given in advance of the declaration of forfeiture. Should the Government prefer its royalty in cash, payments will be based on New York prices for the preceding quarter.

In the zone defined as "from a point 18 km. east of Punto Arboletes a straight line ending at Cape Tiburón; to the east and west of the given points, two parallel lines running south until they reach a distance of 60 km. north of the extremity of the Gulf of Uraba; on the south join these two parallels above described by a line drawn from the east to west," there shall be no preferential right by reason of discoveries; the minimum exploitation tax shall be 20 per cent. of the gross product; and ordinarily no lease shall be given for a tract of land larger than 5000 ha., except for special fiscal operations.

Ecuador

Bolivar's decree of 1829, which was issued at Quito, applied to Ecuador and reserved the mining subsoil to the Government. Article 508 of the Ecuadorian civil code of 1887 declared mines to be the property of the Government; and the mining law of 1892 expressly reserved petroleum to the nation.

The present petroleum law was passed in September, 1921, and authorizes the executive to grant leases not to exceed 20 years from the

date on which exploitation begins, with an extension of 10 years, the latter being subject to the laws in force at that time; the royalty to be not less than 5 per cent. nor more than 12 per cent., depending on the zone, and to be paid in kind or money at the option of the executive. In the latter case, the price is to be based on three factors: the average New York prices during the preceding quarter, less transportation charges; the effective cost of production plus general overhead expenses, plus a reasonable profit; and the prices current in Ecuador during the preceding quarter. The rentals on national lands are 20 cents Ecuadorian money, during the first year; 40 cents the second; 80 cents the third; and 1 sucre per hectare from the fourth year onward. The lease includes the right to use land, water, timber, and construction materials, in the area contracted on national lands, and to expropriate those found on privately owned lands in accordance with the mining code.

The application should state the name and nationality of the petitioner; declarations as to the possible existence of petroleum in the area described; undertaking to put up the bond required by law, and give proof of the applicant's financial and technical ability. A map of the property must be attached. Not less than 500 nor more than 5000 ha., can be leased to one individual or company in one canton, nor more than 15,000 ha. in one province; surveys to be made by the Government and applicant jointly, at latter's cost. The tracts do not have to be continuous but may be split into several parcels of not less than 500 ha. each. The lessee must agree to keep a representative in Ecuador; must file a bond of from 1000 to 5000 sucres, and 100 sucres additional for each 100 ha., which bond shall be forfeited to the Government if exploitation is not begun within 4 years; must obtain permission of the Government to transfer lease contracts; must agree to file reports the first of January of each year, or oftener if called upon by the Government; must employ 50 per cent. Ecuadorians as laborers and experts, unless excused by the executive; and the books must be kept in Spanish.

Among the grounds for revocation is the failure to exploit within 4 years or a suspension of exploitation for three consecutive months, except for *vis major* and fortuitous accident. The lease contract carries with it, for 20 years, an exemption from fiscal, municipal, and other taxes on the capital employed in the exploration, exploitation, refining and transporting of petroleum, machinery and other things necessary for the construction and conservation of pipe lines, tanks, and pumping stations; the right to use national lands, on the payment of 1 sucre per annum for each 10 ha. outside the lands covered by the concession; the right to use national lands along the sea and banks of rivers and to establish new ports, if indispensable. The law further provides that if a concession is abandoned or cancelled for cause, all machinery, buildings, and equipment pass to the state without remuneration. If otherwise abandoned or

cancelled, the lessee has 6 months in which to remove the same, although the Government shall have the preferential right of purchase during that period. It further provides that roads built by the lessee are to be public, and that all such roads, wharves, and docks are to become the property of the state. The Government is to be supplied with petroleum at the cost of production and transportation plus a small profit. Prices fixed for petroleum sold in the Republic are to be approved by the Government, but the lessee must be left a reasonable profit. Controversies in which the Government is a party are to be decided by the Supreme Court. Disputes arising between individuals shall be decided by the Ministry of Public works subject to review by the Council of State.

Holders of petroleum lands under previous laws shall enjoy them for 50 years from the date of the new law, by paying 15 sucres rental per pertenencia per annum and a royalty of from 5 to 12 per cent., depending on the zone. They are also required to exploit within 4 years from the date of this law.

Panama

The legislation in regard to petroleum in Panama is in a very unsatisfactory state owing to the existence of three or four laws having conflicting provisions. However, the basis of Panamanian mining legislation is found in that of Colombia, as Panama prior to 1904 formed a part of that Republic. The constitution of that year reserved mines to the state, without prejudice to acquired rights. Panama, by laws of 1909 and 1913, attempted to vest the petroleum subsoil in the nation. On Jan. 31, 1914, the Supreme Court decided that the petroleum subsoil belonged to the nation, even in privately owned lands; but it reversed itself on Jan. 15, 1915, deciding that the petroleum subsoil in lands acquired prior to February, 1904, when the Panama constitution was adopted, belonged to the owner of the surface and must be respected as acquired rights. As a result of this decision, the legislature passed Laws 6 and 38 of 1915, Law 57 of 1917, and a fiscal law effective July 1, 1917, under which the executive may grant exploration permits on national lands, the explorers to have a preferred concession right to any lands explored. Contracts of exploitation are for 20 years, and call for the payment of a royalty of 5 per cent. of the gross product or 10 per cent. of the net profits to the Government at its option. The concessionary must show that he is financially able to handle the enterprise. He shall have a preferred purchase right for national lands necessary in exploiting the petroleum concession and for the expropriation of private lands when required. The contract also carries exemption from export duties on the petroleum obtained and from import duties on the machinery necessary for refining purposes.

· As a further result of the second decision of the Supreme Court, under Law 38 of 1915, owners of private lands can contract the exploitation of

the same, but when contracted through the Government, they must exact a 20 per cent. royalty for the owner and a 10 per cent. royalty for the discoverer, should there be any.   The owner has a 6-months' preference after notice that a third party desires to exploit his property, and if the owner signifies his desire to exploit and fails to do so in 1 year, his rights pass to the discoverer.

Peru

The mining code of 1900 reserved all mines to the Government except certain non-metallic substances; petroleum was not included in the exceptions.

The petroleum law of Jan. 2, 1922, declares deposits of petroleum to belong to the nation. It provides for a petroleum advisory board. Petroleum concessions shall be solicited through the Ministry of Fomento. The unit of measure for concessions is the pertenencia, which comprises 4 ha. Applications have precedence in order of date, all things being equal. Concessionary companies must be constituted in accordance with the code of commerce, and must maintain a domicile in the Republic and a representative at Lima. Foreigners cannot acquire petroleum claims by any title within 50 km. of the frontiers. Concessions are transferable only with the consent of Government, under penalty of cancellation. The executive can reserve two or more lots in each pool, whose exploration and exploitation shall be granted only to national companies organized with local capital. He can also reserve for the state such zone or zones as he may consider advisable. Special permits to exploit must be obtained even as to lands covered by the concession. The petroleum industry is declared a public utility and the state may expropriate concessions when its safety so requires. Applications for concessions for exploration must be prefaced by a deposit of £200, Peruvian money, for each 1,000 pertenencias.

Exploration concessions are subject to: Limit of 15,000 pertenencias on the coast, 20,000 on the "Sierra," and 30,000 in mountains; annual exploration tax per pertenencia of 1 sol on the coast, 40 centavos on the "Sierra," and 20 centavos in the mountains payable every semester; term of from 2 to 4 years, with possible extension of 2 years.

Petroleum claims can be grouped without reference to continuity, forming rectangles whose sides shall not exceed the proportion of 10:1. In concessions bordering on the coast and rivers, the short side of the rectangle must rest on the river or coast as the case may be. Exploration operations must be effected in accordance with the plan presented by the applicant, within 6 months after the granting of the concession. The concessionary must advise the Ministry of Fomento as to the progress and and results of the exploration, and deliver annually copies of the topographical and geological maps, filing complete map at end of period of exploration. Products found during the exploration belong to the con-

cessionary, except 10 per cent. to the Government within 150 km. of the coast, and 6 per cent. elsewhere. Explorers have the exclusive right to obtain concessions to exploit all or part of the lands explored by them within the term designated for the exploration. Concessions are to be cancelled if transferred without consent of the Government; for failure to pay the tax; expiration of the term; failure to file maps; pay royalties; complete suspension of the work of exploration for 1 year; and at the request of the concessionary.

Exploitation concessions are granted for an indefinite period, in one or more lots of 1000 pertenencias each, grouped as provided for exploration grants. The annual tax is 1 Peruvian pound per pertenencia, until production is obtained; and thereafter the tax is payable under a scale running from 0.9 pound per pertenencia, when an average of one ton is produced, down to 0.05 for an average of ten or more. This scale is reduced by 50 per cent. on concessions covering the "Sierra" and mountains. In addition, there is a 10 per cent. royalty within 150 km. from the coast, and 6 per cent. elsewhere; the payment to be in kind or cash at the average prices current in Lima in the preceding semester. Deliveries in kind are to be at port or railroad station used by concessionary. The concessionary must supply, pro rata according to production, the petroleum necessary for the country, and may export the excess. He must offer the Government or Peruvian capitalists 25 per cent. of company's shares and must employ natives in a proportion to be established in each case. The minimum of production fixed must be obtained from the third year onward. There is a 5 per cent. tax on the consideration expressed in transfers, when made. An export tax must also be paid, but the scale existing at the time the concession is granted shall not be increased for 20 years from the date of this law. The right to erect docks, pipe lines, etc. and to expropriate private property, when necessary, is granted.

Concessions for exploitation may be cancelled: For failure to obtain minimum production required for 5 consecutive years; for failure to pay tax or deliver production; for failure to deliver petroleum for local needs; for transfer without permission of the Government. Pipe-line owners may be compelled to carry oil for the nation and for neighboring concessionaries not having pipe lines. The executive shall fix the tariffs for such service. Holders of old concessions shall pay the new rentals and taxes.

Venezuela

Under the constitution of 1904, the Federal Government has all the revenues yielded by mines and public lands. The present petroleum law became effective July 30, 1921; under it, as under previous legislation, the Government claims the petroleum subsoil, private ownership in

the well not being obtainable.   This law makes a distinction between exploration and reserved zones.   In the exploration zones, exploitation contracts may be closed only with a legitimate explorer or his assignee.   In the reserved zones, it is not necessary that the party who closes the exploitation contract be the explorer.   Exploration permits cannot cover more than 240,000 ha. in parcels of 10,000 ha. each, at a rental of ½ bolivar per hectare.   They are irrevocable for two years, although they may be declared null for certain causes, and may be transferred by giving notice to the Minister of Fomento.   Applications for exploration permits must remain on file 8 months.

The exploitation contracts are granted for 30 years and, when based on exploration permits, cannot be denied unless the exploration permit was invalid.   They may cover up to 120,000 ha., but cannot cover more than one-half of the property explored, the plats being so selected that the result will give a sort of checker-board arrangement, the concessionary having the right to select tracts of five plats or less, covering 200 ha. each, arranged in a square or rectangle, the Government reserving every other tract.   The concessionary must pay 5 bolivares per hectare covered by the exploitation contract within 1 month after the publication of his contract in the *Official Gazette* and the same amount per annum thereafter. He must also agree to pay 15 per cent. royalty (this was 50 per cent. under the law of 1920) to the Government in kind or 15 per cent. of the commercial value of the oil extracted, although the executive is authorized to reduce this percentage if there are special difficulties to overcome.   However, the royalty shall never be less than 10 per cent. in kind or 10 per cent. of the commercial value, and in no event less than 2 bolivares per ton of the product exploited.   Refined products consumed in the Republic must pay 50 per cent. of the import duty on petroleum products, although the President may waive this clause.   Machinery and installations, except refinery equipment, which have become fixtures, as well as pipe lines, belong to the Government at the termination of the contract. Exploitation contracts must be published in the *Official Gazette* and must be approved by Congress.   The concessionary is furnished with a certified copy of the contract and the law approving it and can have the file recorded in the local registry office of the district in which the land is located.

Reserved zones comprise lands under direct exploitation by the Government, those contracted for prior to the present law, those declared forfeited under this law and the tracts consisting of five plats or less, of 200 ha. each, cut out from the tracts explored by concessionaries, and such others as the Government may reserve.   They may be contracted by bids, which are never to be less than the conditions imposed on the concessionary of the exploration zones.   With respect to the exploitation tax

of these zones, the basis of the bidding may be set at 25 per cent. of the commercial value of the product or 25 per cent. in kind. Reserved zones may also be contracted for without bids by the executive and on the same basis as exploration zones, and for a period not to exceed 40 years.

Concessionaries are not permitted to suspend exploitation without good cause. The contracts are transferable with the consent of the Government, but only to legal residents of Venezuela and no person or company can acquire more than 120,000 ha. Petroleum required for public services shall be furnished by the concessionary at a discount of 20 per cent. from the price on the day of sale. The contracting parties, although subject to the taxes of stamped paper, stamps and others of a general character established by law, are exempt from any others that refer especially to the exploitation of mines and make the obligations assumed by them more burdensome. The concessionaries enjoy rights of expropriation of necessary lands; the usual mining easements; the importation, free of duty, of the machinery, buildings, implements, instruments and other things used for exploration and exploitation of petroleum deposits, and the refining and extraction of their products.

No contracts can be effected with persons or companies not legally residing in Venezuela. Rights acquired under previous laws will be respected.

CONCLUSIONS

It may fairly be deduced from the foregoing summary that the tendency in Latin America is to encroach upon the rights of the individual and to create a government monopoly in the rights to the petroleum subsoil, either by express legislation or by excessive taxation, and by administrative policy.

Panama and Colombia endeavored to give effect to this policy by express legislation; and when their supreme courts declared the laws to be illegal in respect to the owners of lands acquired prior to certain dates the congress of each country passed new legislation, purporting outwardly to respect vested rights as defined by the respective supreme courts and yet by taxation and regulation to attain the object pursued theretofore.

Guatemala candidly admitted, in Decree No. 722, Dec. 15, 1915, that it had not reserved the petroleum subsoil, but nevertheless proceeded to take it regardless of possible vested rights of private property.

El Salvador, by a law of April 18, 1918, repealed its mining decree of 1899 and declared that consequently the state owns all the minerals or subsoil substances including petroleum. El Salvador salved its public conscience for thus despoiling the rights of surface owners, by stating that the effects of this law shall not prejudice the rights of surface owners

who may have previously discovered and reduced to possession minerals within their respective properties, or the legitimately acquired rights of third parties. Like Mexico, it erects a mentally dishonest theory of "rights in expectancy," in bland disregard of the legally vested rights of its surface owners.

Honduras, after granting coal rights to the surface owner by its mining code of 1880, took them back by its code of 1895 and, without having specifically mentioned petroleum, has granted petroleum subsoil rights, by means of concessions, without regard to whether the lands thus granted lay in the public domain or were privately owned.

Brazil's mining law of January, 1921, provides that in the future transfers of public lands shall not carry the subsoil rights unless there is a clause to that effect. The provisions of this law also infringe on the rights of the surface proprietor, as his right to prospect for petroleum on his own land is now limited to 1 year after notice, and should he fail to prospect, a third party may do so by paying him for such damages as may be incurred in the use of his property. If the owner does not begin exploiting the mines discovered on his property within 1 year, the Government or third parties may do so, paying to such owner either the damages caused his property, or a portion of the net profits, not to exceed 3 per cent. These provisions would seem to be in violation of article 72 of the federal constitution, granting the mine to the surface owner, and of the prohibitions levelled against the taking of private property without due process of law and against retroactive laws, as they permit the confiscation of the mining property given to the owner of the surface by the constitution, for the sole reason that the owner of the surface is financially unable to exploit his property or cannot, in the space of a year, interest other capital in developing it for him.

Mexico, in 1917, under the provisions of article 27 of the constitution of that year, attempted to nationalize the petroleum subsoil; there is now pending before the Mexican Congress a petroleum law to give effect to this article. The result of the law, if enacted and enforced, will be to confiscate to the Mexican Government the subsoil rights of 95 per cent. of the surface owners of the Republic. Notwithstanding this effort, the Minister of Industry, Commerce, and Labor recently stated before the Mexican Congress that the executive department was opposed to the retroactive application of article 27. The word "nationalization," means that the nation is expropriating to itself something that has not belonged to it. Like El Salvador, Mexico endeavors to assuage its international conscience by stating that surface owners who have taken steps to make effective their "rights in expectancy" to the petroleum subsoil granted them by the mining codes of 1884, 1892, and 1909, will be protected. Yet the Mexican Government, in all of its branches, would contend that this is not retroactivity and that this is not confiscation.

. However much we may wish to decry the immorality of depriving the surface owner of his subsoil rights, from a purely commercial point of view, outside of Mexico, it is of secondary import to foreign oil companies, whether the petroleum in the subsoil is owned by the private surface owners or held by the nation. In the first case, the foreign companies would set about entering into contracts with the surface owner for the exploitation of his subsoil; in the latter, they would make contracts with the nation. From a practical standpoint, title questions are much more simple in those countries in which the petroleum subsoil clearly belongs to the nation, as titles of many of the surface owners have become clouded and involved through a failure to observe the formalities of the laws as to wills, probate, and the like.

But the oil companies are vitally interested in the extent of governmental restriction and interference with individual enterprise in the development of the petroleum wealth of any of the Latin-American republics. State monopolies are bad for the industry, for individual interests, and for the state itself. States are not constituted to handle any industry economically, efficiently, or intelligently. Capital, in the large amounts required for the petroleum development of any of the Latin-American countries, must hesitate to enter any in which the petroleum industry is a government monopoly and wherein the petroleum laws tend to throttle private initiative and enterprise and to disregard world conditions. The exploitation of the petroleum should be left to the free play of economic laws rather than to be hampered by the short-sighted policy of chauvinistic legislation or the arbitrary discretion of government officials. As one South-American writer has indicated, the old Spanish mining legislation was excellent from the standpoint of mining regulations in themselves, but from the economic standpoint it was a complete failure.[7]

When Guatemala attempted to limit its mining concessions to its own citizens and for a term not to exceed 10 years, it sounded its own death knell as to petroleum development. Where in all Guatemala could be obtained the millions necessary for the petroleum development of that country? Would Guatemalans have found capitalists willing to risk large sums on the development of properties that could be held only for 10 years? When the Argentine fails to enact a petroleum law permitting the acquisition of petroleum lands in tracts large enough to warrant a heavy investment, can it complain that its petroleum industy is developing too slowly? Can Venezuela expect foreign capital to hasten the development of its petroleum industry, when the Government desires a 15 per cent. royalty for itself and assumes none of the losses? Can Panama expect a rational and sound development of its petroleum industry, when its surface owners are coerced into exploiting their property

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[7] Gonzalez, *La Propiedad de las Minas*, Buenos Aires, 1917.

within a certain time upon penalty of losing it, and when the surface owner's right of contract is so interfered with as to require him to exact 20 per cent. royalty for himself irrespective of the conditions under which exploitation must take place, and irrespective of the workings of economic laws in general?

If the United States is, today, the wealthiest nation in the world, it is because individual initiative and enterprise flourish therein. If some of the countries in which the doctrine of state control is in vogue have progressed, it is in spite of it, not because of it. Chile, Brazil, and Argentine are the leading Latin-American countries; it is no mere coincidence that they are also the republics that have most encouraged foreign capital and individual enterprise. During the early years of the present century Mexico enjoyed great economic prosperity, its people were not heavily taxed, its treasury had a comfortable cash balance, its credit was good; but today the credit of Mexico is bad, taxes are heavy, its treasury is empty, and it is importing corn to feed its people—all because property rights have come to be regarded as a social function, and leaders ignorant of all economic law have assumed to determine the restrictions on private property that such philosophy requires.

If experience has proved in the United States, with its stability of property rights, its abundance of transportation facilities, its near markets, its comparatively low interest rates, the accessibility of first-class machinery and the abundance of skilled labor, that 12½ per cent. is all the royalty that the petroleum industry will bear, how can Panama, Colombia Venezuela, and Bolivia, with their lack of all of these things, expect a rapid development of their petroleum resources on the royalty bases they exact? They remind us of the millions of dollars that the Standard Oil and the Dutch Shell companies have made, but they ignore the millions that have been lost by these and all the other companies in the search for new fields. In no industry is the gamble so great, and the returns, when success comes, must be high to compensate for the failures.

If the Latin-American countries desire the development of their petroleum deposits by British and American interests—at present they alone have the capital available and, incidentally, they are the foremost exponents of the doctrine of accession, inviolability of the rights of private property, and individual enterprise—these countries must be more liberal in matters of taxation, royalty, reservation, compulsory development, size of petroleum properties, interference with personnel and management, and the like. Too many of these republics tend to plan their budgets around what the petroleum industry will produce in the way of taxes for current expenses, rather than to take into consideration that the greatest benefits that they can derive from the petroleum industry do not arise from the temporary taxes that the industry may pay, but from the economic advancement and industrial employment of their people

and from the increase in the wealth of the nation as a whole.   Royalties paid to the owners of the surface add to the wealth of the individual citizens of the country, make available more capital for the development of other industries, and thus add to the permanent wealth of the nation with greater and more lasting benefits to the national treasury.   Royalties paid to the government are too often spent for non-wealth-producing activities in the nature of current expenses, which add nothing to the welfare of the people or the permanent economic wealth of the country. Until these facts have been brought home to the Latin-American republics and more liberal legislation is enacted by them, the development of their petroleum resources must come slowly.

Petroleum Reserves of the West Indies

By Arthur H. Redfield,[*] Washington, D. C.

The West Indies are the summits of a submerged mountain chain, the continuation of which must be sought in the mountains of central Honduras. In ¦Haiti, the chain divides, one branch passing through Jamaica and the other through Cuba, the Cayman Islands, and the Misteriosa bank.

Suess divides the West Indies into three zones: The first zone is entirely of volcanic origin and includes many recent volcanic cones. It includes the arcùate string of islands, which extends from Saba and St. Kitts to the Grenadines and Grenada. The western half of Guadeloupe belongs to this group. The second zone consists of a geanticline, of which the sedimentary rocks range from the Jurassic to the Quaternary, but consist chiefly of Lower Tertiary sediments. In the west it is broad, including the whole of the Greater Antilles; but in the east it is restricted to a narrow belt, which includes the Virgin Islands (except Anegada), Anguilla, St. Bartholomew, Antigua, the eastern part of Guadeloupe and part of Barbados. The islands that compose the third zone are flat and low. Like the second, this zone is broad in the west and narrow in the east; it includes the Bahamas, Anegada, Sombrero, Barbuda, and part of Barbados. Geologically, Florida and the plain of Yucatan may be regarded as belonging to it.

Trinidad and Tobago, though geographically adjacent to the Lesser Antilles, are geologically a detached part of the South American continent. The crystalline complexes of northern Trinidad and of Barbados are a continuation, *en échelon*, of the crystalline axis of the promontory of Paria. The folded Cretaceous and Tertiary of the central and southern parts of Trinidad represent a continuation of the folded sediments of the Paria district of Venezuela.

Possible Oil-bearing Areas.

The volcanic islands of the inner arc of the Lesser Antilles may be excluded from consideration; these include Saba, St. Kitts, Nevis,

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[*] Published by permission of the Director, U. S. Geological Survey.

Redonda, Montserrat, the western half of Guadeloupe, Dominica, Martinique, St. Lucia, St. Vincent, the Grenadines, and Grenada. The low-lying islands of the outer arc may likewise be excluded because of their structure. They are composed essentially of flat-lying Upper Tertiary beds in which there are no indications of the presence of oil. To this group belong the Bahamas, Anegada, Sombrero, and Barbuda.

The prospective oil fields of the West Indies lie in the second or Great Antillean arc. The structure of these islands is generally anticlinal, though modified by faulting and igneous intrusion. Of the islands of this group Cuba, Haiti-Santo Domingo, and Barbados offer the greatest, promise of oil reserves.

The structure of Porto Rico is not favorable for oil. A broad zone of the crystalline complex occupies the center of the island; Tertiary and Quaternary sediments occupy coastal areas of relatively minor extent.. The structure of Jamaica is likewise unpromising. Folded Cretaceous sediments form the nucleus of the Blue Mountains, which extend in the eastern end of the island in a northwest-southeast direction parallel to the northeast coast. Similar folded Cretaceous rocks occur in the center of the island in the Clarendon and Jerusalem ranges. These mountain ranges are surrounded by a dissected plateau of more or less horizontal Oligocene limestones, which cover four-fifths of the island. Both the folded Cretaceous of the mountain ranges and the horizontal Oligocene limestones are intruded over a large part of the island by late igneous rocks. These have produced considerable alteration in the sediments.

CUBA

The oldest rocks known on the island of Cuba are a complex of serpentines, granites, slates, and schists, unless we except two exposures of limestones of doubtful Paleozoic age, north of Trinidad, one in the province of Santa Clara and the other near the north coast of the island,, crossing the border between Camaguey and Santiago Provinces. On this crystalline complex, which extends through the center of the island along its major axis, rest the later sediments. Jurassic limestones and altered shales form the crests of the Sierra de los Órganos and the Sierra. del Rosario in the province of Pinar del Rio; this is the only exposure of sediments of early Mesozoic age in the West Indies.

Cretaceous sediments overlying the Jurassic are exposed in the foot hills of the Sierra del Rosario, in Pinar del Rio. They occur in the province of Habana, and extensive areas are in the province of Matanzas. Cretaceous formations crop out about Cienfuegos and along the southwest coast of Oriente Province. The Tertiary sediments occur in longitudinal zones of varying width on both sides of the central mountain range, and

along the south coast of the province of Oriente.   Quaternary sediments form coastal plains at intervals around the island.

The general structure of Cuba is geanticlinal.   The older rocks, both sedimentary and igneous, crop out in a general way along the axis of the island; and the younger formations are exposed in irregular zones on both sides of this axis.   The geanticline is asymmetric; the slopes are broader and the dips gentler on the southern than on the northern side of the mountain axis.   The basement complex and the Jurassic rocks are, in general, intensely folded.   The Cretaceous and later sediments, which overlie the basement complex, have as a rule comparatively low dips. Both the Cretaceous and the older rocks are intruded by igneous rocks. Subordinate folds occur away from the central axis; these follow, in general, the trend of the island.

Asphalt seeps and veins or oil and gas seeps have been reported from every province in Cuba.   They are most common on the north coast, especially in a zone about 20 mi. (32 km.) wide, between Esperanza and the boundary of Santa Clara Province, and 475 mi. (764 km.) long. Most of the oil or asphalt seepages occur in fractured serpentine. DeGolyer believes the oil was derived from the Jurassic limestone or other sedimentary rocks; that the igneous rocks from which the serpentines were derived were intruded for the most part into Cretaceous rocks that overlie the Jurassic; and that the seepages found in the serpentine originate in the underlying sediments or in patches of sedimentary rocks caught up in the serpentine.   It is believed by others, however, that the Cretaceous beds are unconformable with some of the serpentine bodies and that the oil has accumulated near the unconformity.

Obviously, no serious attempt can be made to estimate the unmined petroleum reserves of Cuba until the geologic questions as to its origin and mode of occurrence are settled.   No well drilled to date has passed through the serpentine, in which the seepages occur, so that the underlying structure is a matter of conjecture.   Without more exact knowledge of the geologic occurrence of the oil, we cannot estimate even the extent of the probable oil-bearing formations.   DeGolyer believes that oil still remains in the island, but that the prospects of developing a great oil field are not good, and that the hazards of prospecting are more than the average.

HAITI-SANTO DOMINGO

The axis of the main range of the island, known as the Gran Cordillera or Cordillera Central in Santo Domingo and as the North Range in Haiti, is composed of a complex group of schists, serpentines, intrusive and extrusive igneous rocks, tuffs, and more or less altered sediments. This complex was mapped by Bailey Willis as Paleozoic, but a part of it is now considered to be no older than the Cretaceous; many of the intru-

sive rocks are of Tertiary age. The crystalline complex makes up a large part of the Samaná Peninsula; it crops out at Cabo Francés Viejo.

Undifferentiated Cretaceous and Eocene sediments, principally limestones, form the Sierra de Neiba of Santo Domingo and its westward continuation, the Central Range, or Chaine des Mateux of Haiti. The diagonal range of the Montagnes Noires and the south slopes of the North Range of Haiti consists of these limestones. The lower slopes of the Sierra de Bahoruco in the province of Barahona are composed of Cretaceous and Eocene sediments; the South Range of Haiti is a continuation of this sierra.

Oligocene limestones and conglomerates appear on the north slope of the Cordillera Central of Santo Domingo and in Pacificador Province on the south slope of the Cordillera Septentrional.

Miocene sediments fill the Yaqui Valley of northern Santo Domingo; they crop out on the north slope of the Cordillera Central and on the Samana Peninsula and are exposed on the north and west margins of the geosynclinal Plain of Azua and in the valley of Rio Yaqui del Sur. In Haiti, the Miocene series underlies the Central Plain and another area is south of Mirebalais; it probably underlies much of the lower valley of Rivière Artibonite.

The Pliocene is represented in Santo Domingo by the Las Matas formation, which is exposed at the village of that name in the San Juan valley, province of Azua. In Haiti, the Hinche formation of Pliocene age covers large areas in the northern part of the Central Plain.

Recent alluvium forms the surface deposits of the broad river valleys and the coastal plain of southeastern Santo Domingo, as well as numerous small coastal areas. In Haiti pre-Tertiary syenites and granites occupy large areas or belts more or less parallel with the axis of the North Range. Intrusions of quartz-diorite occur north and northwest of Gonaïves.

Late basaltic lavas form the crest of the Sierra de Bahoruco in Barahona Province; they crop out on the south slope of the Cordillera Central in Azua Province. In Haiti, andesite dikes cut the quartz-diorite intrusions in the region north and northwest of Gonaïves.

The general structure is geanticlinal. The oldest rocks are exposed in a broad strip through the center of the island, with the younger rocks flanking each side. The geanticline is modified by many flexures and faults. Except in the area of the basal complex, where folding has been intense, faulting has been the dominant element. The three longitudinal valleys that cross the island—Cibao Valley, Vallèe Centrale-San Juan-Azua depression, and Cul de Sac-Enriquillo basin—are of Miocene and younger sediments, partly at least outlined by faults. The mountain ridges that separate these grabens are hard, older formations.

· The diastrophic movements have occurred during several geologic periods. The intense folding and shearing of the basal complex took

place in pre-Eocene times; movements of less intensity occurred throughout the Oligocene. Considerable igneous activity took place before the middle of the epoch. The Oligocene earth movements were evidently complex, and at the close of the epoch further diastrophism occurred. No evidence of igneous activity in Miocene times is at hand. The latest Miocene deposits show faulting and vertical tilting.

The post-Miocene movements were accompanied by pronounced faulting and folding; the Cordillera Central was raised to a greater height than it now has and the Cordillera Septentrional, a block of Oligocene and older sediments, was lifted up. Structural deformation took place in the Miocene strata of Cibao Valley and still more intensely in the south; the great structural valleys of the island were probably produced during Pliocene time. The events of late Pliocene time include mild folding, repeated elevation and depression, and vulcanism.

The most promising area in Haiti and Santo Domingo, from the standpoint of petroleum production, is the central structural depression of the island. This includes the Central Plain of Haiti, and the Valley of San Juan and the Azua Plain of the Dominican Republic. This depression is a geosyncline, modified by secondary synclinal and anticlinal folds. The axis of the geosyncline and its secondary folds is northwest-southeast, parallel to the structural trend of the mountains that enclose it. In Santo Domingo, the synclinal basin is partly outlined on the north by a fault.

The oil-bearing horizon in the Azua Plain is a Miocene series of conglomerate, coarse sandstone and sandy shale. In Haiti, the most promising formation is considered to be the Thomonde formation (Lower Miocene) of shales, sandstones, conglomerates, and limestones. There are well-authenticated reports of seepages in the Miocene beds of the Azua Plain. Reports of seepages in the Thomonde formation of Haiti have not been confirmed. The area of the Central Plain of Haiti is about 850 sq. mi.; of the Valley of San Juan, 1050 sq. mi.; and of the Plain of Azua, 1350 sq. mi.

The minor folds with which the geosynclinal Central Plain of Haiti is modified include a marked anticlinal dome, a compound anticline upon which three minor asymmetric anticlines are superimposed, and a plunging asymmetric anticline. The San Juan Valley is so largely covered with Pliocene gravel that the folded beds beneath can be studied at only a few places. The section exposed along Rio Yaqui del Sur, near the east end of the valley, shows almost entirely coarse sandy materials that offer no promise of containing petroleum. In the Azua Plain, the presence of an elongated dome, which trends nearly north-south, is suggested near Higuerito. To the southeast the beds are greatly crumpled and faulted. West of Higuerito details of the structure are not satisfactorily known.

In general, too little is known of the detailed structure of the central depression of the island to hazard a quantitative estimate of the petroleum it contains. Apparently favorable structures occur in the Haitian end of the depression. In Santo Domingo, the petroliferous character of the formations is attested by seepages and by a short-lived production obtained by the drill. This area, of all the prospective West Indian oil fields, appears to be the most deserving of examination by the geologist.

BARBADOS

Six-sevenths of the island of Barbados is occupied by coral limestones, which apparently at one time capped the whole island. The denudation of these limestones in the east has exposed the Scotland series (Lower Tertiary?) of sandstones and shales. These are overlain unconformably by about 300 ft. (90 m.) of the Oceanic series (Miocene) earths and clay deposits. The Scotland series forms the core of the island. It is closely folded into a series of anticlines and synclines striking mostly east-northeast by west-southwest. The folds in some places are overturned for short distances and are broken by numerous faults, some of which are vertical while some dip 13° to 15°. The Oceanic series (Miocene) rests unconformably on the eroded and upturned edges of the Scotland and, in turn, is unconformably overlain by the Pliocene (?) or Pleistocene coral limestone which caps the island.

The structure of Barbados is complicated and the details are little known. The strata of the Scotland series are closely folded, considerably faulted, and much shattered. The area of Barbados is only 166 sq. mi., and the horizontal cover of limestones makes it impossible to ascertain how much of this limited area is underlain by the Scotland series. In view of all these circumstances, the petroleum reserves of Barbados may be considered to be insignificant in relation to the world situation.

SUMMARY

The West Indies (exclusive of Trinidad and Tobago) do not constitute a promising area of oil reserves. Most of the islands present unfavorable structures or composition. The smaller islands, with the exception of Barbados, are composed of late eruptive rocks or of flat-lying Upper Tertiary sediments. Of the Greater Antilles, only Cuba and Haiti-Santo Domingo appear to have the proper geologic structure for the accumulation of commercial pools of oil.

Any attempt at a quantitative estimate is futile. On the basis of the known data, however, the total unmined reserves of the West Indies are small in comparison with those of any one of the oil districts of the United States.

DISCUSSION

W. D. NOBLE, San Juan, Porto Rico (written discussion).—Contrary to the author, the writer believes that there are oil reserves in the northern and southeastern parts of Porto Rico.

Some geologists say that the West Indies belong to the northern Yucatan range, not to the Honduras. The writer thinks that Jamaica belongs to the Honduras chain but that Cuba, Santo Domingo, Porto Rico, etc., as far as Antigua, belong to the Mexican range. The same geologic formation is found in Cuba, Santo Domingo, and Porto Rico— the central axes of the three islands differ but slightly and in northern Cuba and Porto Rico the same formation (Jurassic limestone and altered shales) is found.

Although no serious attempt has been made by geologists to drill wells in Porto Rico, having studied the island for many years, the writer believes that the question of oil reserves in Porto Rico should be thoroughly investigated. The island lies directly north of the oil-bearing fields of Venezuela; while directly west are the Azua Plain of Santo Domingo and the Mexican oil fields. From the Bayamón district to 5 mi. southwest of Lares, from 6 to 10 mi. from the coast, there lies a belt of very probable oil-bearing formation 5 or 6 mi. wide and approximately 40 mi. long.

Porto Rico, during the Oligocene period, experienced a great deal of folding, intrusions, and faultings and afterwards a great deal of sedimentation occurred through erosion, forming the plains around the island today. The Cretaceous deposits on the northern coast rest on the older rock formation, and subsequently sank or were depressed, forming a deep ravine or gully where no doubt the probable oil-bearing belt of Porto Rico is located.

ARTHUR H. REDFIELD (author's reply to discussion.*)—The Antillean chain of submarine mountains is related, as Mr. Noble suggests, but not directly, to the structure of Yucatan as well as of Honduras. In the island of Haiti, the principal Great Antillean axis branches into two distinct arcs. One extends along the southern peninsula of Haiti through Jamaica and is connected through numerous banks and keys with the mountains of northern Honduras. The other axis extends through the northern peninsula of Haiti to southeastern Cuba where a second virgation occurs. One axis is represented by the Sierra Maestra and the Cayman Islands; the other extends longitudinally through Cuba, and is paralleled by the Yucatan Bank and by the slightly deformed Tertiary strata of northern Yucatan. The axial trends of Cuba and Yucatan,

* Published by permission of the Director, U. S. Geological Survey.

according to Vaughan,[1] are not coincident but are separated by a structural trough, the Yucatan Channel.

The structure of Porto Rico, like that of Cuba and Haiti, is geanticlinal. Its central range is formed by an older series, composed of pre-Tertiary igneous rocks, both extrusive and intrusive, both fragmental and massive; and of Cretaceous and perhaps Eocene (?) shales, conglomerates, and limestones, which are steeply tilted, closely folded, and considerably faulted and intruded. On the north side of the island, a younger series of Tertiary (chiefly Oligocene and Miocene) limestones and calcareous shales is exposed in a zone 6 to 15 mi. wide. These younger sedimentary rocks rest unconformably on the central complex. A Quaternary dune-sand formation fringes the coast. Along the south coast, a zone of similar Tertiary formations is faulted down in places against the central complex and elsewhere rests upon it in marked unconformity. The Tertiary zone extends, 3 to 9 mi. in width, from Puerto Salinas de Coamo to and including the Guanica Peninsula.

The younger series is represented on the north side of the island by the dark lignitic San Sebastián, Lares, or Collazo shales (middle Oligocene). The San Sebastián shales are succeeded by the Lares limestone (upper Oligocene), the Aguadilla limestone (upper Oligocene), and the Quebradillas limestone (Miocene). These strata are comparatively little disturbed; their greatest dip does not exceed 10°; and no definite indications of folding or warping within the formation have been discovered. In the southern part of the island, the younger series is represented by the sandy Juana Diaz shales and marls (middle Oligocene), the Guanica shaly limestone (middle Oligocene), and the Ponce beds (Miocene) of chalky limestone. In the interior, the beds of the younger series are highly tilted, locally showing dips as high as 30° to 36°. Toward the coast, the dips are successively less, approaching zero. Locally, gentle folds occur. In many places the strata have collapsed, through the development of sink-holes in the limestones.

The rocks of the older series may be eliminated from consideration, as they have been intensely folded, considerably intruded, and much eroded. The oil-bearing possibilities of Porto Rico may therefore be assumed to be restricted to the areas of the younger series.

Neither in structure nor in lithologic character do the younger series of the northern coast offer great promise of oil reserves. In the younger series of the southwestern coastal plain a few gentle folds occur. No evidence has been adduced, however, to show that these structures are petroliferous. Reports of seepages and other indications of petroleum have not been authenticated.

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[1] T. W. Vaughan: Biologic Character and Geologic Correlation of Sedimentary Formations of Panama in their Relation to Geologic History of Central America and the West Indies. U. S. National Museum, *Bull.* 103 (1919), 601.

The formation extending from the Bayamón district westward to the vicinity of Lares, that has been said to be probably oil-bearing, is probably the San Sebastián shales, which are middle Oligocene; these consist of thin-bedded reddish clays and sands containing lignitic beds. No evidence is at hand to show that they are petroliferous. The known Cretaceous sedimentary rocks are included in the area of the older series and are too disturbed and intruded to be a likely area for oil production.

Although Porto Rico lies more or less northeast of the Maracaibo oil fields of Venezuela and north-northwest of the Paria oil fields, the geographic relation alone has no especial significance. The prevailing strike of the minor folds in the Maracaibo structural basin, according to Sievers' map,[2] is about N 61° E; and the continuation of this structural basin must be sought, according to the same author,[3] beyond the Peninsula of Paraguaná in the gulf lying between the Leeward Islands and the Venezuelan mainland. In the Paria region, the anticlines strike about N 75° E, and are continued beyond the Gulf of Paria in the anticlines of Trinidad.

A closer analogy appears to exist between the younger series of southern Porto Rico and the Vallée Centrale-San Juan-Azua depression of the island of Haiti. In both places, Tertiary sediments have been apparently faulted down against the older central complex, and wrinkled by minor folds. In Porto Rico, however, the folding appears to have been slight. A similar analogy may probably be drawn between the northern zone of the younger series in Porto Rico and the Cibao Valley of the Dominican Republic. But the Cibao Valley is monoclinal in its western end and broadly synclinal in its eastern end, and broken by numerous faults; the younger series of the north coast of Porto Rico slopes gently toward the ocean. No structure favorable to the accumulation of commercial pools of petroleum has yet been recognized in either area.

There is no warrant, however, for inferring any structural continuity between the Mexican oil fields and the coastal sedimentary zones of Porto Rico. The tectonic lines of the northern Mexican oil fields trend generally north-southward from Ébano and Topila to Álamo, and have no connection with those of the Greater Antilles, which trend almost at right angles to them. The structural lines of the Tehuantepec-Tabasco fields, though parallel in trend to those of the Greater Antilles, belong to a distinct system.

[2] W. Sievers: Zur physikalischen Geographie von Venezuela: "Petermann's *Geog. Mitteilungen*," **42**, 125–129, 149–155, 197–201. Gotha, 1896.

[3] W. Sievers: Die Sierra Nevada de Santa Marta und die Sierra de Perija: *Zeits. Ges. Erdkunde* (1888) **23**, 65ff.

Possible Petroleum Reserves of Philippine Islands

By WALLACE E. PRATT, HOUSTON, TEXAS

THE Philippine Islands have produced no oil commercially; nevertheless, oil is known to be present at various places in the islands.[1] Although all attempts to produce oil commercially have failed, no intelligent, persistent exploration has been carried out. The Richmond Petroleum Corpn. (a subsidiary of the Standard Oil Co. of California) and a syndicate headed by Ralph Arnold are at present engaged in exploring Bondoc Peninsula in Tayabas Province, one of the promising Philippine areas, in a manner that promises to test the petroleum possibilities adequately and intelligently.

How can the petroleum resources of an entirely undeveloped area like the Philippines be estimated, where not even the areal extent of the various geologic formations is known? Obviously no calculated figure for assured future production can be derived. Yet a study of the evidence of the occurrence of petroleum, the character of the possible reservoir beds, the probable source rocks, the character and size of favorable structures, and the area over which petroleum-bearing rocks are known to occur, will throw some light on the question. The fact that rocks of the same geologic age and similar composition have been explored and are producing in Borneo, Sumatra, and Java to the south and in Formosa and Japan to the north also affect the problem, so that one can obtain some idea of the quantity of petroleum that may reasonably be expected from the Philippines. It is this measure of the possibilities rather than a definite calculation of petroleum reserves that is presented herewith.

This paper will not attempt to describe, nor even to enumerate in detail, the occurrences of petroleum in the Philippines, as such data have already been published. Fig. 1 indicates the localities where

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[1] For discussion of the petroleum geology and description of the natural occurrences of petroleum see: Wallace E. Pratt and Warren D. Smith: Geology and Petroleum Resources of Southern Part of Bondoc Peninsula, Tayabas Province, Philippines, *Phil. Jnl. Sci.*, Bur. Sci., Manila (1913), Sec. A., 5, 301-376; Wallace E. Pratt: Occurrence of Petroleum in Cebu, *Idem.* (1915) Sec. A, 4; Wallace E. Pratt: Petroleum and Residual Hydrocarbons in Leyte, *Idem.* (1915) Sec. A, 4; Wallace E. Pratt: Occurrence of Petroleum in Philippines, *Econ. Geol.* (1918) 11, 246; Warren A. Smith: Petroleum in the Philippines, *Trans.* (1921) 65, 47.
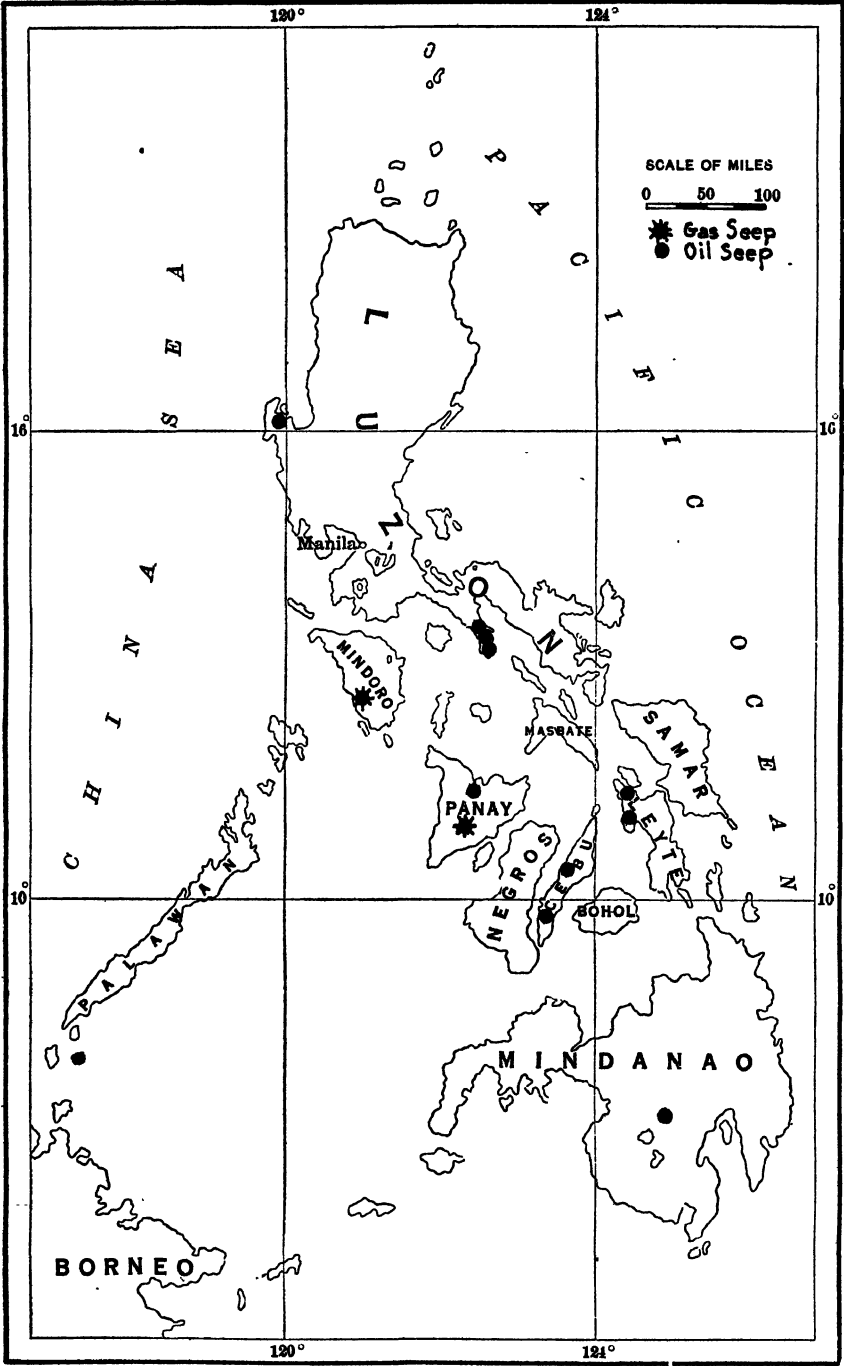
FIG. 1.—PETROLEUM-PRODUCING REGIONS OF PHILIPPINE ISLANDS.

there are known evidences of petroleum, and Fig. 2 shows the geographical relation of the Philippines to the nearest petroleum-producing areas.

The Philippine Archipelago comprises about 3000 islands with an aggregate area of 115,000 sq. mi., 90 per cent. of which is included in the
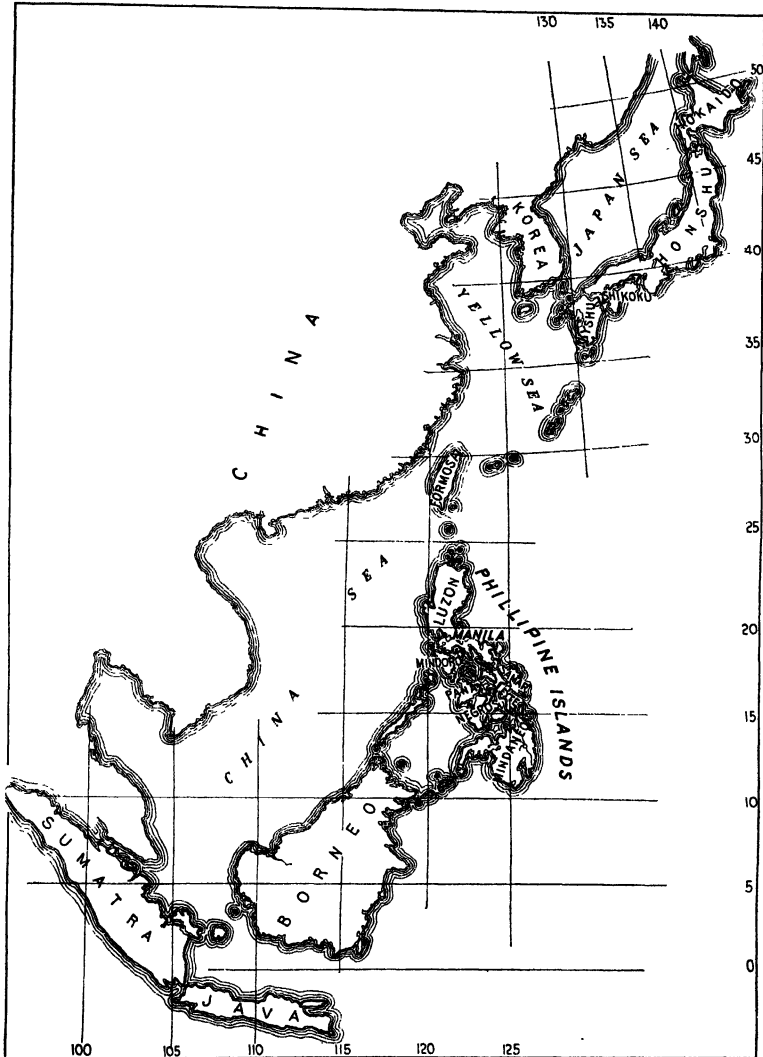
FIG. 2.—RELATIVE POSITION OF PHILIPPINE ISLANDS TO JAPAN, BORNEO, ETC.

ten larger islands: Luzon, Mindanao, Mindoro, Samar, Leyte, Masbate, Cebu, Negros, Panay, and Palawan. The Philippines lie about 800 mi. (1290 km.) off the eastern coast of China and, together with the geologically related Japanese islands to the north and the Dutch East Indies

to the south, mark the extreme eastern margin of the Asiatic continental area. Immediately eastward, and just off the continental shelf, are the greatest depths of the Pacific Ocean, but between the islands and the mainland the sea is shallow.

On all the larger Philippine islands, except Palawan, there are definite indications of the presence of oil. Luzon, Leyte, Cebu, Mindanao, and Panay have oil seepages. On Mindoro and Samar inflammable gas is encountered, and on Masbate and Negros there are the same Tertiary (Miocene?) shales that yield oil through seepages elsewhere, and which retain here their petroliferous character. Probably the same shales are present in Palawan, but little geologic exploration has been carried out on this island. The petroliferous shale series is of adequate thickness, 3000 ft. (915 m.) or more, and of suitable character to supply commercial quantities of petroleum. Intercalated with the shale beds are sandstones, which would serve as reservoirs. In places, the geologic structure is known to be favorable for the accumulation of oil in these sandstones. Seepages in regions of favorable structure indicate that accumulation has taken place. Other and more convincing evidence, perhaps, is found in the commercial exploitation of petroleum from rocks of the same geologic age and similar character in Formosa and Japan to the north, and in Borneo, Sumatra, and Java to the south.

The popular conception of the Philippines as a group of volcanic islands obscures the fact that a fair proportion of their area consists of sedimentary rocks, not severely broken nor folded nor otherwise impossible as retainers of petroleum. Luzon and Mindanao, the two conspicuously large islands lying at either end of the group, contain thousands of square miles of sedimentary rocks, while the Visayan Islands, occupying a central position in the archipelago, are even more largely of sedimentary origin. Samar with an area of from 5000 to 6000 sq. mi. consists almost exclusively of sedimentary rocks. Altogether in the Visayan Islands there are probably 12,000 sq. mi. of fairly flat-lying sedimentary beds. A large proportion, perhaps 50 per cent., of the sedimentary area includes, with other Tertiary formations, the petroliferous shale series in which oil seepages occur. Throughout the Philippines, these oil-bearing rocks may be estimated to be present over a probable total area of from 10,000 to 15,000 square miles.

The area over which actual seepages of oil are known is much less impressive. On Bondoc Peninsula, in Tayabas Province, Luzon, where there are a half-dozen good seepages of light oil, the showings are scattered over about 300 sq. mi. In the northeastern part of Leyte, in the vicinity of Villaba, there are oil seepages and asphalt (or paraffine) beds scattered over about 150 sq. mi. On the east coast of Cebu are two seepages, about 30 mi. (48 km.) apart, but it is doubtful if all the intervening area is similarly petroliferous. Other isolated seeps

occur in Mindanao and in Panay.   On the whole it seems fair to estimate the known area of petroleum-bearing rocks in the Philippines at 500 square miles.

The 300 sq. mi. of known petroleum-bearing rocks on Bondoc Peninsula have been surveyed in a reconnaissance way and several sharp anticlines are mapped, along which the largest seepages are found.   Thus, a half-dozen small sharp anticlines are known which might be expected to produce, if they produce at all, over an area ¼ mile wide and from 1 to 3 miles long.   If it is assumed that one such field develops from the six known anticlines, a productive area of 500 acres might fairly be expected.   As, probably, there will be several productive sands in the column, an ultimate yield of 10,000 bbl. per acre reasonably may be allotted to this area.   Thus a figure of 5,000,000 bbl. is obtained as the probable ultimate production of the 300 sq. mi. of oil-bearing rocks on Bondoc Peninsula.

The remaining 200 sq. mi. of known oil-bearing rocks are to be set down, from our present knowledge, as slightly less promising than an equal area on Bondoc Peninsula.   Therefore, a total ultimate probable production of 7,500,000 bbl. for the 500 sq. mi. of known oil-bearing rocks would appear a reasonable estimate.

The possibilities of the probable area of petroleum-bearing rocks in the Philippines must be estimated even less closely than the probabilities of the known area have been.   Presumably, it will be conceded that the probable area promises less per average square mile than does the known area, but it is difficult to make a logical comparison of chances in the two classes of territory.   For the purposes of this paper, the probable area of petroleum-bearing rocks is taken as 15,000 sq. mi. and to this area is allotted one-fifth the chance per average square mile of the known area of petroleum-bearing rocks.   This fixes the possibilities of the probable area at 45,000,000 bbl., and the total probable and possible total ultimate production for the Philippine Islands becomes 52,500,000 barrels.

Japan (including Formosa), to the end of 1920, had produced about 43,000,000 bbl. of oil and had a daily average production of about 5800 bbl. during that year.   Japan, accordingly, might reasonably be expected to yield 60,000,000 bbl. of oil ultimately, without important new discoveries.   The Dutch East Indies (Borneo, Sumatra and Java) had produced more than 221,000,000 bbl. of oil at the end of 1920; and in that year averaged 48,000 bbl. daily.   The ultimate production of these islands, without important new discoveries, ought to be 350,000,000 barrels.

These figures give Japan an ultimate recovery of 345 bbl. per sq. mi. of total area and the Dutch East Indies 670 bbl. per sq. mi. of total area.   If the Philippines, lying directly between these two producing areas and similar to both of them in geologic constitution, should produce

a volume of petroleum per square mile of total area equivalent to their average production per unit area (500 bbl. per sq. mi.), its total ultimate production would be 57,500,000 bbl., which figure checks closely with the results of the estimate just presented. In other words, if the total ultimate production of the Philippines is, as estimated, 52,500,000 bbl., the ultimate production per square mile of total area will be 457 bbl. as compared with 500 bbl. average for the probable total ultimate production per square mile for Japan on one side and for the Dutch East Indies on the other side of them.

The petroleum reserves of the Philippines, then, are estimated at from 50,000,000 to 55,000,000 bbl., but it should be remembered that these figures apply to a region that has never produced any oil commercially and about which we have very incomplete data and that they are intended to represent only the order of magnitude of the petroleum possibilities.

Petroleum Resources of Japan

By J. Morgan Clements, New York, N. Y.

Petroleum has been known in Japan since at least 668 A. D., for a picture shows the presentation, during that year, to the Emperor Tenchi (Tenji) of "burning water" and "burning earth" by his subjects from Echigo Province, which today is the largest producer of petroleum in Japan. The burning water was doubtless petroleum and the burning earth either bituminous shale or oil-soaked earth or rock from near the oil seepages.

The early methods of getting the oil were simple and similar to methods used today in Japan and in China. Where the seepages occurred, trenches or shallow wells were dug along the seepages and allowed to fill with oil, which was then bailed out; in some cases, they filled with oil and water and overflowed. During the 300 years following 1500 A. D., the use of petroleum in Echigo Province seems to have increased considerably, as compared with earlier centuries, and records show that in 1818 the depth of the wells had been greatly increased. Doubtless, they passed through the seepages and tapped the shallow oil-bearing strata; probably some of these deeper wells were dug before that time.

The real value of petroleum was not recognized until early in the Meiji era, which began in 1868, when the Japanese learned that the kerosene that was being imported was derived from petroleum. Guided solely by information obtained from books, they refined the oil and increased the depth of their wells in order to increase the production. The deepest well, dug by hand, about this time was 894 ft.[1] (273 m.) and by 1874 the recorded production reached 3499 barrels.

In 1876, Benjamin Smith Lyman prepared geological maps of the Echigo oil field, which were the first of such geological maps made in Japan, and based on these a more enlightened development of the oil fields began; the Japanese, however, still dug the wells by hand. Toward the end of the '80s, an American well-drilling outfit was secured, but the efforts to use it failed from lack of experience by the operators. In 1888, a complete well-drilling outfit was purchased and American oil-well

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[1] J. Morgan Clements: Petroleum in Japan. *Econ. Geol.* (1918) 11, 512–523.

drillers were engaged to operate it; in the fall of 1890, the drill rig was set up over a hand-dug well and in December drilling was begun. A well 1000 ft. (305 m.) deep was completed in 1891 and production began with 45 bbl. per day of 42° Baumé oil. Other wells were rapidly drilled, a depth of 1500 ft. (457 m.) being reached, and one small gusher was brought in. The petroleum industry of Japan may therefore be said to date from 1891, when it was placed on a modern basis.

In 1912, American rotary drills were introduced; the successful use of American drilling outfits and drillers greatly stimulated the industry and led to the employment of many more. Following their custom, Japanese were sent to the United States to study the petroleum industry, and returning to Japan with the information desired they began the manufacture of the necessary oil-drilling machinery and refining appliances, following closely the American prototypes, so that today everything is produced in Japan, except the casing, in sufficient quantity to supply the demand; also the Japanese have trained their own people to use the machinery and refining processes so that they no longer employ our men or purchase our supplies. But while American types of drilling machinery are extensively used, the primitive hand-dug well methods are employed today practically alongside rotary drills.

OIL DISTRICTS

There are five productive oil districts in Japan proper, as shown in Fig. 1. These are situated in the provinces of Echigo and Shinano, Akita, Totomi, and Hokkaido, and one in Formosa (Taiwan); in each district oil has been produced in one or more fields. Other fields have been tested and found to carry oil, but in such small quantities that the production is not profitable at this time. Some new fields may be discovered in these districts, but the relatively small area of unproved ground precludes the probability that any large fields will be discovered. Of a total daily production in Japan of about 6000 bbl., the oil districts of Totomi, Shinano, Hokkaido, and Formosa (Taiwan) produced less than 100 bbl. and are practically negligible from the point of view of present production. Echigo and Akita are the important fields, producing practically 99 per cent. of all the Japanese oil.

The oil fields of Formosa (Taiwan) are in process of development, a large part of the probable oil territory being reserved for the Imperial Navy, but the developments thus far have not been promising for a large production.

The west coast of the southern part of Sakhalin (Japanese Karafuto) has oil indications but no promising developments have been made thus far. Excellent oil indications have been reported from Tertiary deposits in the Russian part of Sakhalin and the Japanese Government has occu-

pied the territory and proceeded with the development of the oil fields; but from the meager reports filtering through it is now known that these developments so far have not been successful. Factors operating against their success have been the difficulty of access and the severity of the
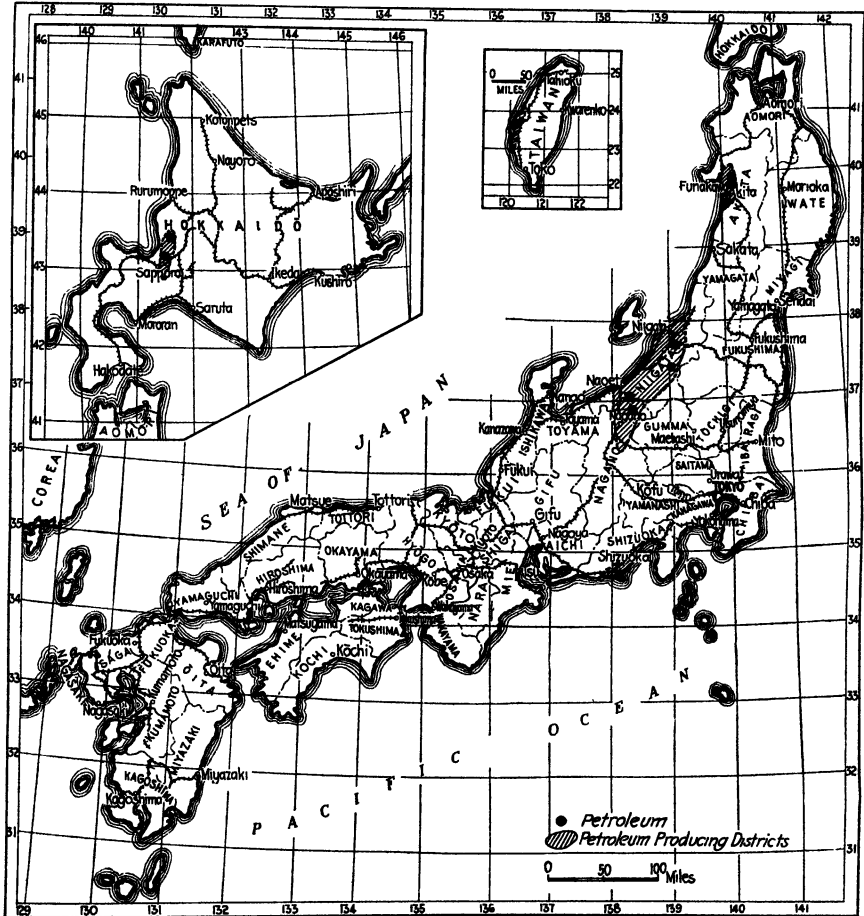
FIG. 1.—PETROLEUM-PRODUCING AREAS OF JAPAN.

climate, which permits the Japanese to carry on their active prospecting for only about 3 months in the year. However, the failure of the Japanese to have developed commercially productive oil fields in Russian Sakhalin, or even their failure to prove their existence, should not be accepted as proof that such oil fields do not exist.

GEOLOGY OF OIL FIELDS

Petroleum, in Japan, is associated in all cases with Tertiary deposits, though every area in Japan underlain by Tertiary deposits has not been

found to carry oil. The oil fields are distributed along the seacoast, except in Hokkaido where the Tertiary oil district is in a strip running through the center of the island.

The Echigo and Akita districts produce practically all the oil of Japan, and as they are the best known and will illustrate the occurrence in others, they are briefly described. These two districts occur in the extensive area of Tertiary sediments that extend almost continuously along the west coast of Honsu (the main island of Japan) bordering the Japan Sea, extending from the Province of Shinano on the south to the northern end of the island, a distance of about 300 mi. (480 km.) with a width of about 20 mi. (32 km.). The Tertiary deposits are classed as Miocene and Pliocene. These Tertiary deposits of the Echigo district have been closely folded, as the result of compression from the northwest and southeast, into rather long narrow folds striking northeast and southwest, the strata being not infrequently faulted. The oil fields occur along these anticlines. The Niitzu oil field on such an anticlinal fold is about 4 miles long by ⅓ mile wide. Most of the fields are much smaller than this however, the Ojya field being only ¼ mile square. The oil-bearing districts are, topographically, rather rough, on a small scale. They are broken up into a series of small mountain ridges, the highest rising to an altitude of 2300 ft. (760 m.), the general height however is about 400 ft. (120 m.). Between the main ridges in the valley of the Shinano River occur broad fertile plains covered with rice fields and dotted with villages, with much narrower valleys between the smaller ridges. These ridges normally correspond to the anticlines and have the same strike northeast and southwest, the valleys occupying the synclines generally speaking, though in future some subordinate productive anticlines may be developed in these rice plains by drilling.

There are three well-defined oil horizons. The top one consists of shale, sandstone, and conglomerate; the middle one, of shale with thin beds of sandstone; the lowest is subdivided into two subordinate horizons, the top one consisting of sandstone and shale and the lower one of sandstone and shale with tuff beds. The most productive horizons are the middle and lower horizons, in which the sandstone and tuff are the chief oil carriers, though some oil is obtained from the shale.

GRADE OF OIL

The heavy oils, as a rule, occur in the higher horizons. Where an oil horizon is penetrated by an intrusive rock, not an uncommon occurrence, the oils are of low specific gravity, even less than 10° Baumé having been found in small quantities. Generally speaking, the Japanese oils are heavy, a fair average for them being around 30° Baumé; the heaviest is the Akita oil of 20° Baumé and the lightest the Totomi oil of 42° Baumé.

Depth and Production of Wells

The deepest well so far sunk is 4613 ft. (1406 m.) deep in the Nishiyama field of Echigo district. The wells generally run from a few hundred feet to around 2500 ft. (760 m.). The shallow wells as a rule are very small producers.

The productive oil fields have, in many if not in most cases, been closely drilled. In the Niitzu field, oil wells may be found no more than 10 ft. (3 m.) apart, so that the upper oil carriers have probably been thoroughly drained and the use of air or water-pressure methods will in all probability not add much from the higher sands. But deeper wells nearly everywhere may be productive.

The well production throughout Japan is small. The largest gusher of which I have record flowed 11,350 bbl. per day at first but rapidly fell off and in 5 months it was necessary to pump the oil. At the end of 1912 there were 2740 producing wells of all types, machine drilled and hand dug, 2153 of which had been drilled with American-type drills. In March, 1917, there were 2840 producing wells in Echigo district alone, of which 2236 had been drilled with American-type drills. The total production, in March, for this district was 158,733 bbl. or at the rate of about 56 bbl. per month from each well, or, say, 1.8 bbl. per day. The average daily production of the Nishiyama field, the most productive in Japan, is 4½ bbl. per day and the average for the entire Echigo district, including the Nishiyama field, is only 2.25 bbl. per day. One company, in 1917, was producing 4,000,000 cu. ft. (113,270 cu. m.) of gas per day in the Nishiyama field, from which casing-head gas was being extracted, the gas then being piped to adjacent towns and used for fuel.

Japanese Petroleum Production

The published records of production date from 1874, when a production of 3499 bbl. is given. The production slowly increased until it reached 30,651 bbl. in 1880, when it began to decline and dropped to 7063 in 1884, when it again began to grow. By 1891, the annual production was 63,618 bbl. In the years immediately following, the successful use of American drill machinery and methods made itself felt in a steady and rapidly increasing growth in production until, by 1901, over 1,000,000 bbl. were produced, and by 1908 over 2,000,000 bbl. Production then fell off until 1912, when the introduction of rotary drills and the drilling of deeper wells rapidly increased the production, until in 1916 the maximum of 2,942,722 bbl., with a value of $7,468,354, was produced, when it again began to decline.

For the first 15 years, from 1874 to 1889, for which there are records Japan had a production of about 261,000 bbl.; for the next 10-year period, 1890–1899, 1,991,993 bbl. From 1900 to 1909, the production was 13,-

046,378 bbl., over seven times that of the preceding decade; from 1910 to 1919, it was 22,737,270 bbl.; and from 1920–1921, 4,413,000 barrels.

During the war, when high prices prevailed, strenuous efforts were made to increase production in Japan, with success for the early period, as is shown by the rapidly ascending curve in Fig. 2. In spite of these efforts, however, partly because of the inability to get casing from the United States, after the increase in 1914–1915 following the great increase of 1912–13 through the use of rotary drills, with deeper wells, the production began to decrease and has fallen rapidly. This is not because of a falling-off in demand, for Japan normally supplies only two-thirds of
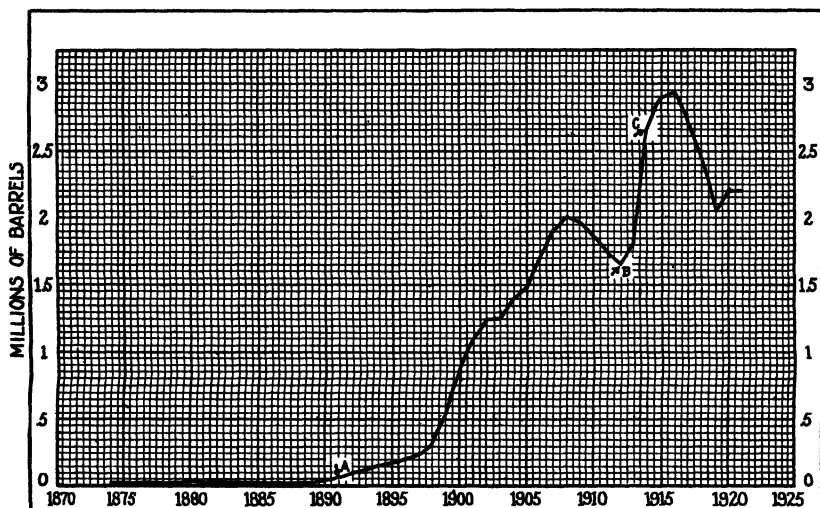
FIG. 2.—PETROLEUM PRODUCTION OF JAPAN. A. USE OF AMERICAN CHURN DRILLS BEGAN. B. INTRODUCTION OF ROTARY DRILLS. C. WORLD WAR.

the amount required for home consumption. It seems quite clear that, in spite of the new wells coming in, the production of Japan is declining and this is recognized by the official efforts to stimulate search for oil. But there is the possibility, which must not be overlooked, that the Government has issued orders regarding the conservation of the oil supply and limited the output, though it is not known definitely that such instructions have been given. The total recorded petroleum production of Japan from 1874 to the present time in round numbers has amounted to 43,000,000 barrels.

OIL RESERVES

I have been unable to learn of any estimates of the oil reserves of Japan that have been made by its oil geologists. Moreover, available data regarding the productive areas are so meager, and much more so regarding the non-productive areas, as to make rather presumptive any

attempt at a reasonably intelligent guess as to the reserves. Nevertheless such an estimate has been made and is here given under heads of oil from "proved," "probable," and "possible" ground.

	MILLIONS OF BARRELS
Oil from proved ground...........................	90
Oil from probable ground.........................	300
Oil from possible ground.........................	0–700

Under the head of "oil from proved ground" is included only the oil it is estimated will be recovered from the important producing districts, some of which fields have been well drilled, and regarding the occurrence of which certain data are available. It would seem that there is an excellent opportunity for the Japanese to continue production for some years from this proved territory. It is not to be expected, however, that any marked increase in production will come from it. However, production from this proved ground will come largely as the result of the drilling of areas in the productive fields that heretofore have been untested or very poorly tested, from the drilling of deeper wells, and by using new and improved methods of oil recovery, such as air pressure, water drive, etc.

Accepting the estimated reserve of 90,000,000 bbl. in proved ground as approximately correct, at the normal average production for the past 10 years of a little over 2,300,000 bbl. per year, this would be exhausted in about 40 years, on the assumption that production continues in the future at the same rate as at present. The production will certainly continue much longer than this, as under the normal conditions of drilling wells and developing oil fields it may be expected that the remaining oil could not be removed within that period of time. What will probably happen will be a slowly but steadily decreasing production from the oil wells, which will continue for many years with an occasional small spurt shown by the coming in of a few isolated new wells. In fact, I believe that from the present production of around 2,200,000 bbl. Japan's oil production will steadily decrease, barring developments of new oil districts in probable and possible ground.

Under the heading "oil from probable ground" is included that oil which will probably be derived from certain rather extended areas, totally untested, in the present productive districts and other much larger areas in which oil seeps are known and from which but small amounts of oil have so far been obtained, such areas, for example, as occur in Japanese Sakhalin (Karafuto), Hokkaido, and Formosa (Taiwan). Thorough testing of these areas is desirable and may add much to Japan's oil reserves.

Under the category of "oil from possible ground" has been included the estimates of oil in territory that is not definitely known to contain any oil, but in which such conditions of structure, etc. exist as to warrant the geologist in assuming that oil may occur there. It must be recognized

as a purely speculative estimate and one that is extremely doubtful, for it is both doubtful whether any oil will be found in the areas considered, and if oil is found, it is a question as to how much will be produced. Testing such areas, however, may add enormously to Japan's proved and probable oil-bearing ground and to the estimates of her reserves.

It may be well to add that the officials of the various government departments interested in the production and use of petroleum in Japan have been discussing an oil policy for Japan, and it is proposed to establish a Bureau to carry on investigations and outline and carry through an oil policy that will conserve the oil supply of Japan and, if possible, increase it. This increase, however, can only be brought about by active prospecting of the probable and possible productive areas of Cretaceous and Tertiary sediments, which occupy a much larger area than that which is occupied by the present important oil-producing districts of Japan.

Petroleum Resources of China and Siberia

By Eliot Blackwelder, Denver, Colo.

(New York Meeting, February, 1922)

For the purposes of this paper, the boundaries of China and Siberia will be taken as they stood about 1907.

Except in the Caspian region, it is doubtful if all the oil ever produced in these countries would equal one day's flow of a good Texas well. Considering the vastness of the regions and the variety of strata, it seems strange that these two countries should have made so small a showing in the petroleum industry. No doubt the condition is caused partly by lack of prospecting and of modern methods of development; in larger measure, however, it is evidently because of unfavorable geological conditions over nearly all of east-central and northern Asia.

China

The known oil occurrences in China are very few. For many years, perhaps a barrel a day has been obtained by primitive methods from wells near the city of Yenchang in the northwestern province of Shensi. A like amount was obtained from wells bored for salt water in the vicinity of Kiating in southwestern Szechwan. The little oil was separated from salt water by allowing it to stand in jars. The salt wells yield also considerable natural gas, which is used to evaporate the salt and also for domestic purposes. There is evidence that wells were bored in this part of China before the third century. It is said that more than 1200 wells have been sunk in that district to depths of 1600 to 2000 ft. (485 to 610 m.) in connection with the salt industry.

The scarcity of petroleum in China may be ascribed to three general geologic conditions: First, China contains practically no marine sediments of Mesozoic or Cenozoic age; second, the Paleozoic sediments are not for the most part of the types that generally contribute petroleum; third, the rocks of nearly all ages have been strongly folded, faulted, and more or less intruded by igneous rocks in all parts of China except certain areas in the west and northwest.

The mountainous eastern peninsula of Shantung and the similar district of Liaotung in Manchuria to the northeast have the general

characteristics of Nevada, the rocks being mostly Paleozoic or older and having complicated structure. That there is oil in either province is extremely doubtful. The great plain of eastern China is believed to be a recently sunken district thickly strewn with the delta deposits of the Hwang-ho and Yangtze-kiang. If oil ever is found in this alluvial region, it will be by accident. The geologic structure of the surrounding
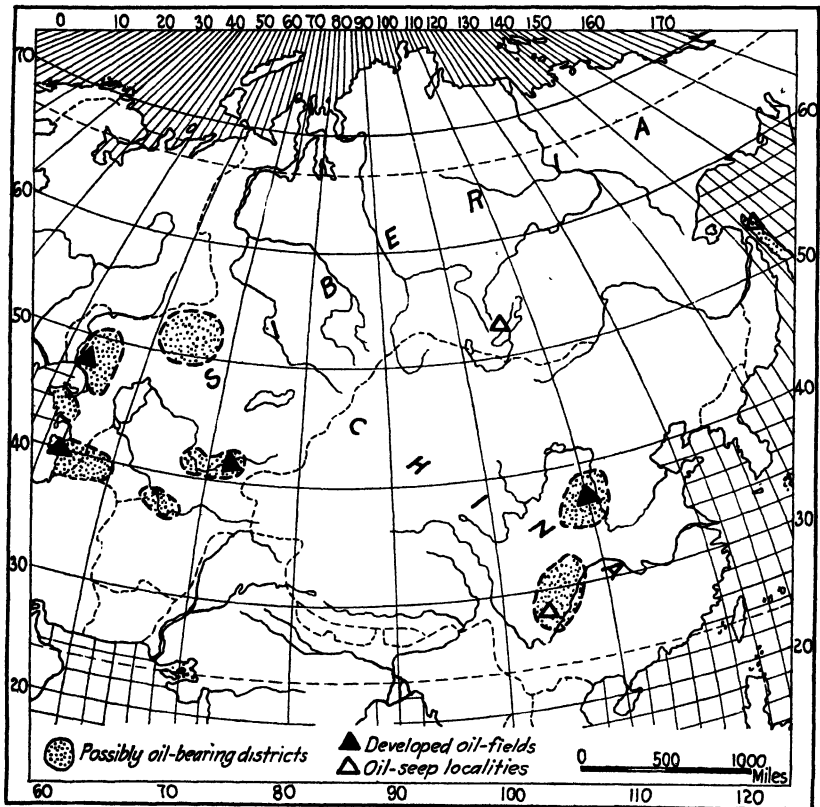
FIG. 1.—OIL-BEARING AREAS OF SIBERIA AND CHINA.

districts gives no suggestion of favorable conditions beneath the plain but indicates rather the continuation of the Shantung structures.

The southeast quarter of China consists entirely of pre-Cretaceous rocks, all strongly folded and faulted, and invaded by igneous intrusions. The probabilities of finding oil in this region would seem to be no better than in the barren Appalachian Mountains. In southwestern China, the likelihood of finding oil is still more remote, for although thick beds of Carboniferous, Permian, and Triassic rocks occur, they have been intensely folded and there are many granitoid and other intrusions.

The Red Basin of the western province (Szechwan) is one of the few places which seem to have even a little promise as a source of petroleum.

The Jurassic red beds are nearly horizontal in the central part of this great elliptic basin, but are more folded around the periphery. They are underlain by thin Lower Jurassic (Rhetic) coal measures and thick, more or less bituminous, Pennsylvanian limestone. It is probable that there are structures favorable to oil accumulation around the margins of the basin. Unfortunately, the Rhetic coal measures appear to be of fresh-water origin, and there are no known sandy formations in proximity to the Carboniferous limestone. Possibly there may be local porous zones in the limestone, which would afford oil fields like those of Kentucky. The small gas production and traces of oil in the southwest part of the province are at least suggestive.

In northwestern China, comprising the provinces of Shansi, Shensi, and Kansu, there is a large area within which the Paleozoic rocks are somewhat faulted but not much folded. Formations older than upper Carboniferous are chiefly limestones and dolomites without bituminous matter, and are not such as to afford hope of finding oil. In the coal measures, the hydrocarbon content is more favorable, but unfortunately these rocks generally lie at the surface in plateaus and have been deeply intrenched by streams. Perhaps in a few localities they may be under sufficient cover and in structural positions favorable for the accumulation of small bodies of oil. It is in this part of China that a trifling amount of oil is being produced at present; that it will ever be more than an unimportant field seems rather unlikely.

Of the outlying dependencies of China, Manchuria, like northern China itself, consists largely of rather old, strongly folded rocks, broken by faults and interrupted by igneous intrusions. It is not likely to contain petroleum in noteworthy quantities. Mongolia is largely unknown, as far as its geology is concerned. However, it has been crossed in several places by geologists and all their observations go to show that it is underlain largely by granite, gneiss, and metamorphosed sedimentary rocks, veneered with late Tertiary (Miocene-Pliocene) continental deposits like those of western Texas and Nebraska, and including some basaltic flows. Nothing in the published accounts available indicates in the slightest degree that there is any oil in Modgolia.

Thibet and Chinese Turkestan to the north are, likewise, very imperfectly known. The rocks are, however, intensely folded in most districts that have been studied. It is reported that Mesozoic and Eocene strata occur in southwestern Thibet; but their structure is like that of the Alps, and the opportunities for petroleum accumulation would seem to be about as poor.

SIBERIA

For the purposes of this paper, Siberia will be understood to include all of Russian Asia, as it was before the war. Of this territory, the most

promising part for future oil supply seems to be the extreme southwestern corner, extending from the south end of the Ural mountains southward along the east shore of the Caspian Sea and eastward to the high ranges of Turkestan. This region has been only partly explored, and only a few localities have been drilled, but much oil already has been produced: along the east side of the Caspian Sea, in the province of Uralsk north of that sea, and in the Khotan Basin at the eastern corner of the triangle. In all three localities development has been slow and is still in its infancy. In this general region, marine strata of Cretaceous and early Tertiary ages apparently are widely distributed and have been folded in varying degrees. Under the circumstances, it seems not unreasonable to expect that other valuable and probably large oil fields will be developed in the Aralo-Caspian part of Siberia.

That part of Siberia lying between the Ural Mountains and the Yenesei River is underlain largely by horizontal or slightly disturbed beds of Lower Cretaceous and early Tertiary age. In general, they appear to be thin and, in part, non-marine gypsiferous deposits. The marine strata resemble the American Chesapeake formation and the English Gault. There is not much in the descriptions of these beds to suggest that they will prove to be petroliferous. Yet it is possible that in the southwestern extremity of the district they may become productive, because it is not much farther southwest to the oil-bearing Tertiary strata of the Caspian region. Unconformably beneath the Cretaceous lie folded Paleozoic beds, which rise to the surface in the Ural, Altai, and Kirghiz mountains. This region, two-thirds as large as the United States, probably has not been closely investigated by geologists, and most of the Russian reports are inaccessible to Americans. Nevertheless, the general geologic conditions are known well enough to warrant a tentative opinion of the petroleum resources.

Between the Yenesei and the Lena Rivers, there are plateaus carved from flat-lying or gently inclined beds of early Paleozoic age. They consist largely of limestone with marl and some gypsum, and are overlain by Permo-Jurassic coal measures suggesting the Gondwana beds of India. Devonian shales and limestone intervene in the southwest part of the region. Toward the north the coal measures are covered by a thin sheet of upper Jurassic and Lower Cretaceous shales; locally there are extensive outpourings of Tertiary basalt. Around the southern border of this district the rocks have been considerably folded. In brief, the conditions are such that it would not be unreasonable to expect some deposits of petroleum in this great region. Traces of oil have been found in Paleozoic rocks near Lake Baikal.

East of the Lena River, the rocks generally are folded in Appalachian style and intruded by numerous igneous masses. The beds range from Cambrian to Devonian, with some Triassic sediments and Tertiary

volcanics. In the valley of the Amur and along the eastern coast, the Triassic contains fossiliferous limestone and thin coal measures. However, these have been folded so strongly as to suggest that no oil or gas will be found in them. At the extreme eastern projection of Siberia, the rocks are largely ancient metamorphic and igneous masses like those on the American side of Bering Strait.

The peninsula of Kamchatka consists partly of granite and porphyries, with a double line of volcanoes running longitudinally. There are, however, extensive outcrops of coal-bearing Tertiary deposits suggesting the Kenai formation of Alaska; this might conceivably yield oil. Oil seepages are reported in the northern part of the west shore of the peninsula.

Near the north end of Sakhalin Island, Tertiary deposits are reported to have petroleum seepages in at least two localities. The writer does not know whether oil has been produced in commercial amounts; but there seems to be reason for expecting that Sakhalin may eventually have some small oil fields like those of Japan.

SUMMARY

To give even a very rough quantitative estimate of the petroleum to be derived from China and Siberia, in the present state of our ignorance, would be unjustified.

China and Siberia have a combined area nearly three times that of the United States. In this vast region indications of the presence of petroleum are found at only a few widely scattered places. Only in the Aralo-Caspian deserts are there developed oil fields in the modern sense of the word. In addition, the island of Sakhalin offers promise of at least minor developments. We do not know even roughly the size of the petroliferous districts, nor the number of favorable structures in such districts, nor their areas. We know comparatively little of the average yield of the wells obtainable there.

The writer will venture only the opinion that China will never produce large quantities of oil, but that Siberia, with the Sakhalin and Caspian provinces, may eventually yield as much oil as Russia south of the Caucasus.

DISCUSSION

FREDERICK G. CLAPP, New York, N. Y. (written discussion).—The writer is much more optimistic than the author with regard to the oil possibilities of China. This may be due to the fact that different sections of the country were studied by each and the writer's studies were carried on much later than the author's.

Few persons have any conception of the lack of reliable information

as to the geology of petroleum and natural gas in China. Conditions in this respect may be compared fairly with those in our own country 100 years ago, when only a few geological studies had been made.

A further comparison is based on the fact that perhaps 50 small seepages have been found in China, all of them more comparable to the unimportant seepages of West Virginia and Wyoming than to any of the great South and Central American seepages. The best oil seepage seen, by the writer, in China was a few gallons of light green petroleum floating on the surface of a small stream. Most of the seepages consist of oozings of asphaltic material a foot or two in diameter, or spots where petroleum rises drop by drop in the water of a stream. No geologist would condemn any otherwise favorable part of the United States as regards oil possibilities simply because it does not have surface evidences; no more should this be done in the case of China.

Among the notes and maps made by the writer and his associates, the only ones that have been worked up in detail cover a portion of a single province, *viz.*, northern Shensi, in central northern China, where most of the work was done and consequently for which the most data are available. The provinces lying farther east are mainly of a highly metamorphic or igneous character, as explained by Mr. Blackwelder. The formations of northern Shensi are largely of Carboniferous age, overlain in the west by probable Permo-Mesozoic sediments. That region is a great sedimentary basin at least 200 miles in diameter, in which older rocks are only found near the edges. Monoclinal dips extend for scores of miles throughout the central portions of the basin with only small reversals, but near the edges some domes exist of the moderate Oklahoma type, and in addition a few prominent anticlines of the Eureka-Volcano-Burning Springs (W. Va.) type are found in some places near the edges. Igneous rocks are unknown in the basin, and in this respect the comparison is almost perfect between the North Shensi basin and some Carboniferous basins in the United States. A vast difference, however, lies in the fact that Shensi is covered by an immense thickness of loess, which is certainly over 1,000 and probably several thousand feet thick in places. It rises into a plateau over a mile above sea-level along the eastern boundary of Kansu. The underlying rocks are only locally exposed.

Another comparison between conditions in China and those in the United States may be made on the basis of the drilling done to date. Three wells, only a few hundred feet in depth, were drilled, by the Chinese, about 15 years ago at Yen-chang in eastern Shensi. Although the initial production of one of these wells is said to have been as high as 40 bbl. per day, the present aggregate production is believed not to be over one barrel per day for several wells. Geologically, the locality is merely a slight interruption in the fairly uniformly westward-dipping monocline,

but minute seepages appear at the surface. The analyses of this oil showed it to be above 40° Baumé gravity, if the writer's recollection is correct, and to contain about 15 per cent. gasoline and benzine, 15 per cent. of kerosene, and 40 per cent. or more of lubricating oil, with a high percentage of paraffin. Therefore the character of the oil is similar to that in rocks of similar age in the eastern United States, and the yield of the existing wells is what is found in many similar structures in eastern Ohio.

Besides the deep test well drilled as an offset to one of the Yen-chang wells, others were drilled elsewhere in Shensi; but one of them was situated on only a structural terrace, another on a moderate dome of the Oklahoma type, and a fourth on a great anticline in a region of probably too high metamorphism. All of these wells reported showings of oil, but the largest probably would not have made over 10 bbl. per day. There is no basis for the suspicion, which exists in China, that large amounts of oil were found in Shensi, and that good wells were capped in. Neither is there any basis for the reports frequently circulated that all conditions in China have been found unfavorable to oil occurrence. Other Carboniferous basins in China are similarly, or more, favorable.

Summarizing, it may be said that geological conditions probably are as auspicious for oil in certain places in China as they appeared anywhere in the United States previous to the actual discovery of oil by drilling; that a number of Carboniferous basins exist in which the superficial conditions may be considered almost ideal; that small seepages exist in considerable number; and that the salt and gas wells mentioned by the author in certain parts of China may be considered just as good an indication of oil possibilities as they would under similar circumstances in the United States. No person can yet say whether or not China will produce oil in commercial quantities, but conditions in certain provinces render them worth testing. The immense market for kerosene that exists in central China, in view of its teeming population, renders practically certain the success of a company that can enter the country and find and develop oil in commercial quantities.

A Possible Origin of Oil

By COLIN C. RAE, DENVER, COLO.

THE absence of paraffin and other oil hydrocarbons in the soil although they are concentrated in extensive deposits in some localities, the common distribution of plant remains through many formations that are not productive for oil, and many other factors, seem to indicate the existence of an intermediate organic compound between the complex plant compounds and the petroleum hydrocarbons; and much evidence points to the humus acids or associated organic compounds, which are probably collected by surface waters and concentrated in certain areas by means of precipitation by electrolytes, which would be contained in sea water or possibly in various inland seas or lakes.

PRESENT-DAY EVIDENCE

There are many present-day evidences of the association of petroleum with humus acids. Binney, in 1912, made observations without explanations on the peat mosses of Down Holland, England, where he found petroleum being formed by decomposition in the presence of an infiltration of sea water. The analysis of a typical peat will show its relationship to the humus acids under consideration.

	SAMPLE No. 1, PER CENT.	SAMPLE No. 2, PER CENT.
Water	20.0	29.5
Ash	3.0	3.05
Fiber	47.0	54.95
Humus acids	30.0	12.50

J. D. Haseman and others have noted the association of asphalt and hydrocarbons with the swampy, boggy areas, particularly adjacent to the sea.

Many of the rivers carry much humus acid in solution, and some tropical rivers carry in the aggregate millions of tons of humus each year. The amount of humus is a variable factor, depending on climatic conditions, and on the salts and alkalies present in the water, also on the density and temperature.

The analyses of various river waters show a close relationship between the amount of organic matter and the amount of free uncombined silica; the relative insolubility of silica in pure water is well known. By chemically equating the positive ions to the negative ions in the various analyses of river waters, there is apparently an excess of free silica, most probably in a colloidal state, varying from a fraction of 1 per cent., in rivers with little organic content, to 44 per cent. in tropical rivers, such as the Uruguay. The relation between the silica and humus is not clearly understood and prominent chemists have given conflicting views concerning the physical and chemical relationship.

CHEMICAL FACTORS

The combined silica is chiefly in the form of sodium metasilicate, which has been shown, by Kohlrausch and others, to be hydrolyzed in solution into colloidal silica and sodium hydroxide. Uncombined silica must be present as colloidal silica since silicic acid hydrolyzes forming the colloidal solution.

A colloidal suspension is finely divided solid matter in the presence of the solvent. It has not been definitely settled whether these colloidal suspensions manifest any of the properties of true solutions. There is satisfactory experimental evidence that the colloid particles are charged electrically, certain being positively charged, while others, such as colloid silica and sulfur, have negative charges.

It has been shown, by Mayer, Schaeffer, and Terroine, that the addition of traces of an alkali to an ultramicroscopic suspension increases the size of the colloid granules if the suspension is positive, and diminishes them if the suspension is negative, while the addition of acids produces the inverse effect. Thus, particles of the negatively charged colloidal silica and sulfur would be increased by the presence of humic acids; this is a probable explanation for the usual relationship of high organic to high silica content in river waters. A strict rule of relative solubility cannot be applied, for the action would vary with the solvent, depending on the salts and alkalies present in the water, also on the density and temperature.

The next factor to be considered is the possible precipitation of organic compounds by sea or inland lake waters. As mentioned, Binney, Haseman, and others have called attention to the association of formation of petroleum or asphalt with humus conditions in the presence of sea water. Analyses of river water at London, England, have proved that the sea water causes the precipitation of the humus or organic content of the water.

Chemically, the coagulation of colloids is effected by electrolytes such as $Al_2(SO_4)_3$, $Fe_2(SO_4)_3$, $FeCl_3$ and other salts, when present in sufficient quantity. Burton has shown that a marked difference in potential

exists between the colloidal particles and water. This diminishes the surface tension between the two, and there is nothing to draw the fine particles together to form larger particles and produce precipitation. When sufficient electrolyte is added to the colloidal suspension, the colloidal particles attract the oppositely charged ions, and the difference in potential between the colloid particles and water decreases, increasing the surface tension. As a result, the colloid particles are drawn nearer together and are precipitated. Hence, the coagulation of organic matter, colloidal silica, and sulfur can be effected by sea water, because of the presence of the needed electrolytes.

Experiments by prominent chemists have proved that the humus-acid groups, whose exact chemical composition is not well understood, can become colloids, and as colloids are easily precipitated by sodium chloride, calcium sulfate, and other salts. This fact probably explains the action of sea water in removing organic impurities from the river waters, and it may be shown in the future that the colloid humus has a definite bearing on the development of silica into a colloidal state through the electrical condition of the ions, provided the various factors heretofore mentioned as influencing the solution are considered.

Considering the probably colloidal relationship of high humic acid to high silica content of river waters, it seems logical to assume a simultaneous deposition of the colloidal silica and probably some colloidal sulfur at the time of the precipitation of humus by action of sea water. The resulting siliceous ooze might be incorporated into some form of marine deposition, probably forming a shale with the fine mud carried by the mechanical action of river waters.

TRANSFORMATIONS OF ORGANIC HYDROCARBONS

The gradual transformation of the organic hydrocarbons, precipitated by the electrolytes of the sea water, into the petroleum hydrocarbons could probably take place under several sets of conditions and would vary according to alkalinity, salts, and the presence of catalytic agents in associated formations, together with conditions of pressure and temperature. The evidence of action of sea water on waters of peat bogs seems to indicate that a slight amount of petroleum can be formed under proper conditions even where only normal conditions of pressure and temperature are present.

In the oil-shale beds of the west, intermediate organic compounds between the humus and paraffin hydrocarbons, which require heat to change into petroleum, are apparently present. Solution tests of the shales indicate that virtually no true hydrocarbons exist in the oil shales, although specks resembling gilsonite ranging from $\frac{1}{64}$ to $\frac{1}{32}$ in. in diameter are found. However, distillation tests show no apparent relationship between the oil content of the shale and the amount of the said hydrocarbons, for the richest shale is sometimes free from the specks.

The character of the intermediate compound has been the subject of much discussion and comment, the prevailing idea being that the organic matter was chiefly carbonaceous and plant remains. However, microscopic studies of the shales have shown that while pieces of plants, pollen grains, seeds and spores are numerous in the shale with low oil content, the rich shales have virtually no plant or organic remains but apparently have some intermediate organic compound between the organic material and oil, which compound may be changed to petroleum on application of heat and pressure.

It would be possible to have the organic material carried to the sea by rivers and streams precipitated over an area many miles in extent, as it is well known that river waters are not thoroughly mixed with the sea water for many miles from their mouths. The soundings of the Challenger expedition showed "siliceous ooze" existing along the "mud line" of the oceans many miles from the mouths of rivers; hence it would be possible to have diatoms or foraminifera in the formation formerly containing organic material.

HUMUS-ACID ORIGIN OF OIL

The occurrence of shales containing much finely divided silica associated with the petroleum deposits of California and elsewhere, together with the iron concentration and sulfur deposits, seem to indicate by chemical, geological, and historical evidence that it might have been possible to have had organic material in the form of the humus-acid group collected by the streams, precipitated by sea water, and later converted into hydrocarbons in the presence of catalytic agents or by means of heat and pressure. Various experiments along these lines may lead in the future to the discovery of a process by which oil may be produced from the organic acids under the proper conditions of heat, pressure, and catalytic agents.

It is not intended to ascribe all oil deposits to the humus-acid origin, but it is important to note that the humus-acid group may be formed from both animal and vegetable material. Local conditions may allow extensive deposits of marine life remains to accumulate, wherein it might be possible to have a similar leaching of humus by water, then a later precipitation by underground waters with subsequent heat and pressure. However, the humus-acid concentration by river waters and embodiment into marine formations seems to be the more plausible explanation of the origin and concentration of many oil deposits.

SUPPLEMENT

It is stated above that microscopic studies of the shales have shown that while pieces of plants, pollen grains, seeds, spores, and other organic material are numerous, much rich shale has no recognizable

organic material. As some scientists have endeavored to ascribe the origin of oil to the chief material appearing on the slide, it might be well to consider certain factors from present-day evidence.

The river and stream waters carrying much humus acids in solution carry plant remains in the form of spores, seeds, pieces of plants, and fibrous matter. The coagulation of the humus material by sea or inland-lake waters would collect much of this material. In the various geological ages, the organic material carried as sediment would vary. In the earlier ages, such as Silurian and Devonian, the plant life was in a low stage of development, probably mosses and ferns, so that the organic sediment might be chiefly spores or low forms of plant life; in later times, the organic sediment might be leaves, seeds and fibrous material.

As the water of rivers and streams does not completely mix with sea waters for some distance, part of the organic material may be precipitated over different zones of marine deposition such as shales, limestone, blue mud, or in diatomaceous areas. Consequently different forms of marine life would be found in the organic shales varying with the locality and the geologic age. Thiessen[1] has found spores (probably ferns) in the Devonian shales, but mentions a darker matrix between the spores which seemed to be indefinite except for a little cuticle. He calls it a probable form of plant degradation and assigns a probable origin of oil to the spores. Usually, in any certain organic shale, the darker portions are richer in organic and oil content. The spores are the lighter colored mass; the dark material contains few spores or remains, but seems to be a homogeneous mass adapting itself to the spaces between the plant material. It may be that this dark unknown plant degradation was formed from organic material in the form of humus acids precipitated in the organic shales and carried down much of the plant sediment in the waters.

Marcus Goldman,[2] in his preliminary report on the Bend Series of Texas, says that the dark rich organic matter of the dark shales and limes often has a flocculent, amorphous appearance with no definite plant forms, which is not explained; this material may be the coagulated precipitate of the organic humus acids. R. H. George[3] found that many rich black shales did not give any microscopic studies of plant life, although the brown shales usually had much organic evidence; this dark material without plant forms may be the precipitate of humus acids. C. A. Davis in his personal notes mentions the jellylike mass of organic material in which the plant remains were embedded. Thus, the various microscopic works on organic sedimentary deposits seem to show the existence

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[1] U. S. Bureau of Mines. Report on Devonian and Other Shales.
[2] U. S. Geol. Survey. Report on Bend Series of Texas.
[3] Oil Shales of Colorado. State Geol. Survey of Colorado.

of an amorphous, jellylike, rich organic material, which might originate from the humic-acid precipitate.

APPLICATION OF THEORY TO INLAND LAKES

As practically all commercially valuable deposits of petroleum have been found in marine formations or in close association therewith, only brief mention was made of precipitation in inland lakes and that alum, iron sulfate, gypsum, and other salts, particularly of the heavy metals, cause the precipitation of organic humus and material.

There is considerable interest in the oil-shale deposits of lakes in Colorado, Utah, and Wyoming. These large lakes received many thousand feet of sedimentary deposits during Eocene time and the soluble salts probably became more concentrated than in the stream waters. The saline condition of sea water probably did not develop but iron sulfate, gypsum, and other soluble salts were present to cause precipitation. The presence of gypsum seams, iron minerals and salines in the Eocene beds, together with oölitic lime deposits similar to those being formed today on the shores of Salt Lake, shows possible conditions for the concentration of lake waters.

A present-day example of the coagulation of organic material by alum and iron salts is the purification of drinking water. Iron sulfate, alum, or certain other salts placed in the sand of the filtering bank will cause a coagulation of the organic material on filter beds which will collect most of the sediment. This coagulation may include microscopic vegetable material and coagulated humus acids where there is much vegetation in the drainage area. The general chemical phases of deposition of the Eocene oil shales in Colorado, Utah, and Wyoming seem to indicate that the organic material may have been derived from precipitation of humus acids in the inland lake waters.

APPLICATION OF THEORY TO OTHER OIL OCCURRENCES

There has been considerable difference of opinion among petroleum geologists as to the amount of organic material in the shales or limes necessary to produce commercially valuable oil deposits, ranging from a small thickness in the Mid-Continent to many hundred feet in California. The type of organic material may have considerable effect, and the amount of coagulated humus in the brown or carbonaceous sedimentary beds, or between the various remains of marine life, may have some bearing on this subject.

There are many productive oil areas, similar in formation and structural conditions, adjacent to barren areas. Many times the large pro-

ductive areas seem to follow certain trends and .then stop, although structural and formational types are still similar. This factor is open to study and explanation. Perhaps the areas having the greatest organic deposits of precipitated material will have the greatest production.

Considering the occurrences of oil in connection with present-day humus conditions adjacent to the sea, the discovery of oil in the mud of the Gulf of Mexico is interesting. Dr. Harold Turner of Johns Hopkins University did some microscopic work on the muds. At first, the tendency was to ascribe the origin of oil to the diatoms. However, the mud with small oil globules showed chiefly "macerated organic material mixed with grains of silica," while other barren mud samples showed many diatoms and related life. This proved to Turner's satisfaction that the oil did not originate from the marine organisms. In view of the other associations of mud deposits with formation of oil, it would seem that the oil may have originated from the organic muds, probably containing the humus-group precipitate. Seepages in the Gulf are possible, but the method of oil occurrence is more favorable to formation of oil in the muds.

J. D. Haseman did considerable geological work at a point on the coast of Florida, where swampy conditions exist adjacent to the sea, and from the evidence of deposition concluded that asphalt has been formed in recent geological times directly from the geochemical transformation of humic acid in presence of sea water and various electrolytes. If this condition exists locally, the possibility of large concentration of the humic-acid group by river waters is an important factor.

In California, the Tertiary oil deposits occur at various horizons, and in different localities probably originate from organic shales ranging from middle Miocene to Eocene. The origin has been assigned to animal and vegetable material in the organic shales. While the latest report[4] puts special emphasis on the diatoms and foraminifera, still there are various factors to be considered. The pure deposits of diatomaceous earth in the upper part of the Kreyenhagen (Oligocene) are usually white, while the darker colored brown shales of Kreyenhagen and Tejon contain less diatoms. There are also brown organic shales in the Tejon (Eocene). Since nearly white, pure deposits of diatoms occur elsewhere similarly without much organic stain, the diatom alone does not seem the true source. Considering Mid-Continent conditions, there seems to be sufficient organic brown shales containing probably coagulated amorphous organic material to produce the oil, and the diatoms and foraminifera in the same and associated formations indicate rather a favorable condition for their growth than the source of the oil. The lack of commercially valuable oil deposits north of Coalinga, although similar diatomaceous

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[4] Anderson and Pack: U. S. Geol. Survey *Bull.* 603.

and foraminiferal deposition continues, may be due to a change in the brown organic shales rather than the lack of structural conditions, as oil is found under many different structural conditions in California.

CONCLUSION

The exact method by which various organic materials are converted into oil has been subject to radical disagreement, and microscopic work has not defined the exact agencies. Perhaps the coördination of chemical and microscopic work using various types of organic material to produce the humus-acid group, with subsequent precipitation and treatment, may prove matters of value to future oil exploitation. It may be that only certain types of organic life will produce the desired reactions. If the type of precipitate was definitely ascertained, it might be possible to predict whether an organic sedimentary formation was favorable to an accumulation of petroleum. The quality and character of the petroleum may vary according to the particular member of the humus-acid group precipitated; the rate of geochemical changes caused by pressure, heat, presence of catalytic agents; and filtration of the product.

DISCUSSION

S. LINKER, New York, N. Y. (written discussion).—In speaking of the oil-shale beds of the West, the author states that "microscopic studies of the shales have shown that while pieces of plants, pollen grains, seeds, and spores are numerous in the shale with low oil content, the rich shales have virtually no plant or organic remains but apparently have some intermediate organic compound." The writer has found that the amount of organic residue varies with the amount of mineral matter in the shale; also that oil shales that are light in weight have more organic residue and less mineral matter than the heavier varieties of oil shales. Distillation tests show that oil shales with a low specific gravity yield much more oil than those with a high specific gravity. One could, therefore, conclude that the greater the amount of organic residue, the higher is the yield of oil on distillation of the shale. This conclusion differs with the statements of the author. Dean E. Winchester found that the Green River oil shales showing the larger percentage of vegetable debris yielded the most oil, and vice versa. This will be found to be the case with oil shales of other localities.

COLIN C. RAE (author's reply to discussion).—In the statement quoted, the author intended to convey the idea that many rich shales in Colorado have no recognizable plant or organic remains, but have an unidentified organic compound called, by some authors, vegetable

debris, plant degradation, vegetal debris, etc., which resembles the precipitate of colloidal humic acids.

It is hoped that his paper has shown the importance of careful microchemical work on the various shales associated with oil deposits, in order that the source of part or all of the oil may be definitely assigned to the unidentified humic ground mass or to the various identified vegetal material.

Time to Pay Out as a Basis for Valuation of Oil Properties

BY W. IRWIN MOYER, M.S.,* PITTSBURGH, PA.

(New York Meeting, February, 1922)

Two methods for the rapid valuation of oil properties are in common use. The one best known and most widely used is the "per barrel" value, based on the present daily production of the well, without regard to its past performance or the history of the sand or pool from which it produces. The second method is the "time to pay out," or the lengt of time required for a well to place itself upon the credit side of the ledger, from which time the returns from its production (less the operating expenses) will be clear profit.

The time basis for valuation is also commonly used in the appraisal of an undeveloped lease. In most oil fields, it is not thought good business to invest in production that will not pay out in at least 4 to 5 years. Nearly all buyers (except in certain regions of very persistent production like the deep-sand territory of the Appalachian field) expect wells to pay out in considerably less time than this.

Like the barrel-per-day price, the time necessary for a well to pay out has been determined for each field, so that one frequently hears a field or district referred to as having a certain time to pay out. This is considered the norm for that field or district and the buyer and seller base their value, to greater or less extent, on their belief in the accuracy of this time.

The writer long ago became convinced of the inaccuracy of the barrel-per-day method, except when greatly modified for local conditions, such as the age of the wells, their economic limit, operating conditions in the field, the future price of oil, and the persistence of the sand as a producer. When so modified, this method becomes more of an analytic or predicted probable-yield valuation. While the appraiser may not go through each step in a valuation based on future returnable revenue, his knowledge of the field and the local conditions surrounding his leases make it possible for him to estimate a fair sales value that, in many cases, will be a close approximation to the truth. This is quite different from she commonly understood barrel-per-day value, in that the conditions turrounding the sale of one lease or property at a fair market value may

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* Geologist, Philadelphia Co.

be entirely dissimilar from those of a nearby property and the per-barrel price of the first property will form no criterion of value for the second.

The present paper is the result of an investigation made to determine the reliability of the time to pay out as a basis for valuation. The data used were decline curves of production from several widely scattered fields in the United States, the curves being compiled from the production records of a large number of wells in the same field. The method used was as follows: For any one field, an imaginary well was assumed to have produced a given amount of oil during the first year. Estimating that this well will produce, in the future, an amount of oil corresponding to the decline curve of the field, its value was ascertained, using a certain fixed price for oil. This valuation was based on future returnable revenue, the sum of the annual revenues to be derived from sales of oil from the well being the total expected revenue. From this was deducted the original drilling cost and the annual operating expense for as many years as were required for the well to pay out. As soon as the revenue received from the sales of oil from the well equalled the expense of drilling and operating, the well had paid out. Thus, in the Glenn Pool, if a well produced 10,000 bbl. during its first year, oil selling at $1, estimating the well's future capacity from the decline curve of the field, the well will not pay out until it has been operated for 2.6 yr. from the date of its completion, or 1.6 yr. after its first year. A similar evaluation was then made for a well of the same size but using a different price for oil. As soon as the well paid out, the evaluation was concluded as only the time required was desired. This process was repeated for a wide range in prices of oil. The production of the well was then varied and for each variation in production the corresponding time to pay out for each price of oil was ascertained. The result gave a considerable variation in time, covering a large range of annual production and a wide fluctuation in the price of oil. The points so found were plotted, using annual production as ordinates and years to pay out as abscissae; the price per barrel was plotted as the third variable.

In all cases, the original annual production, which is the basis for the future production of the well, and hence the index as to the well's time to pay out, is included in the valuation of the well. Thus, in the case given, the 10,000 bbl. produced in the first year is part of the revenue by which the well pays out. Obviously, it is not necessary to have the entire year's production in the case of a new well. From the records of previous wells and from field experience, the approximate production of an average well can be foretold by the production of its first few days or months. While, for the sake of clearness, it has been assumed that the original annual production is the production of the well used in estimating the time to pay out, it is not necessary that the well be a new one. A well may have produced many years but its future production can be reason-

ably ascertained from similar wells in the field, therefore its time to pay out can be estimated the same as a new well. In such case, however, the well's former production and operating expenses will be added to its future production and operating costs.

There was no discount made for the depreciation in the value of money returnable in future years, nor was there any account of the almost inevitable increase in the price of oil in future years; these two factors will tend to neutralize each other to some extent. To have discounted all the computations necessitated in making the curves would have been unjustifiably onerous; and to have attempted to predict, with any semblance of accuracy, the future trend in the price of oil would have been far from the purpose of this paper. However, both these factors should be taken into consideration when a curve is to be used as a basis of valuation in actual transactions.
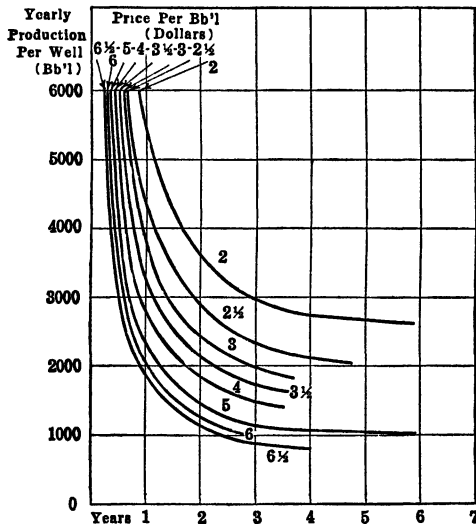
FIG. 1.—NORTH CENTRAL WEST VIRGINIA (BIG INJUN SAND).

Time to pay out for oil wells having a yearly production of from 800 to 6000 bbl., a drilling cost of $10,000, and an annual operating expense of $400; oil sells at from $2 to $6.50 a bbl.

Of the three variables—price per barrel, size of well, and drilling and operating costs—the first and second vary much more widely than the third. In fact, the influence of drilling and operating costs, as it increases or decreases in the same field, was so slight as to be considered negligible; so all valuations were made and the time to pay out was computed by using fixed drilling and operating costs. These costs will vary in time for the same field but the variation will in no case greatly influence the time to pay out. To demonstrate this, Fig. 5 has been drawn. Using the curve of the Big Injun sand production of north central West Vir-
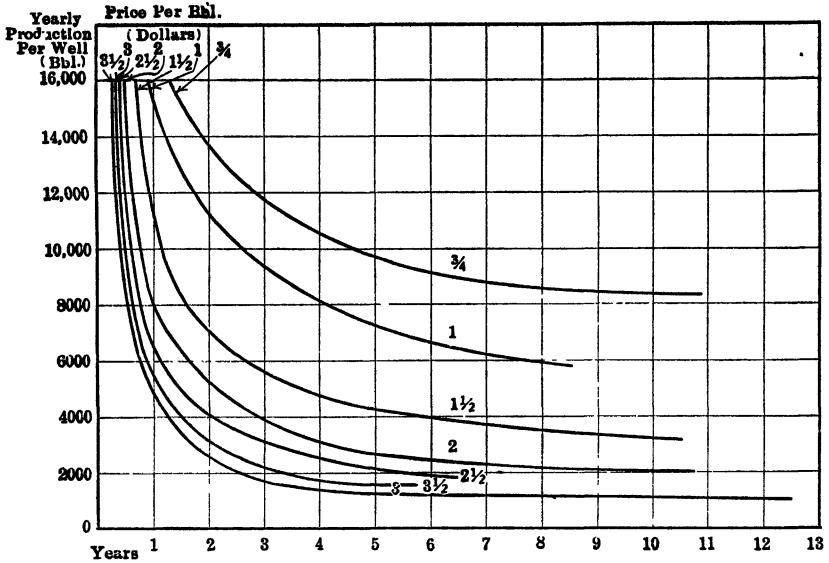
FIG. 2.—GLENN OIL POOL, OKLA.

Time to pay out for oil wells having a yearly production of 1500 to 16,000 bbl., a drilling cost of $15,000, and an annual operating expense of $150; oil sells at from $0.75 to $3.50 a bbl.
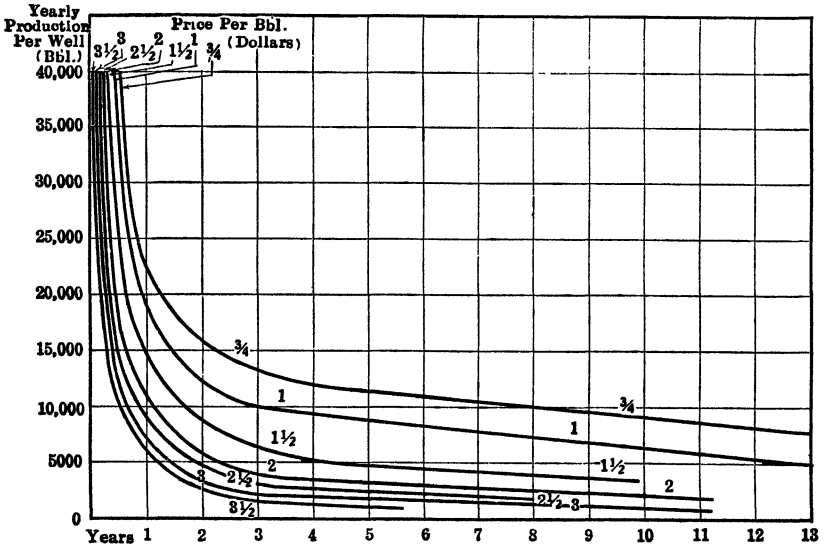
FIG. 3.—EASTERN OSAGE OIL POOL, OKLA.

Time to pay out for oil wells having a yearly production of 800 to 40,000 bbl., a drilling cost of $15,000, and an annual operating expense of $150; oil sells at from $0.75 to $3.75 a bbl.
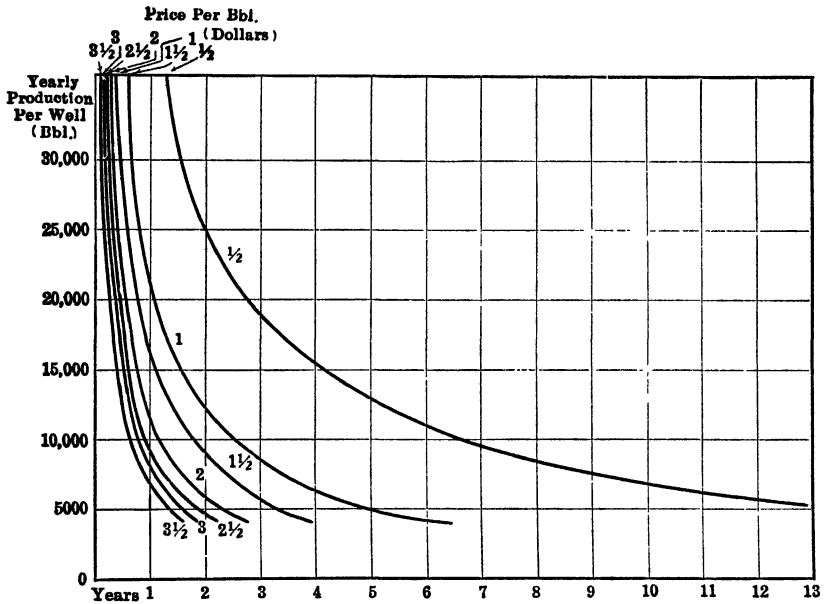
FIG. 4.—SALT CREEK OIL POOL, WYO.

Time to pay out for oil wells having a yearly production of 4000 to 36,000 bbl., a drilling cost of $20,000, and an annual operating expense of $700; oil sells at from $0.50 to $3.50 a bbl.
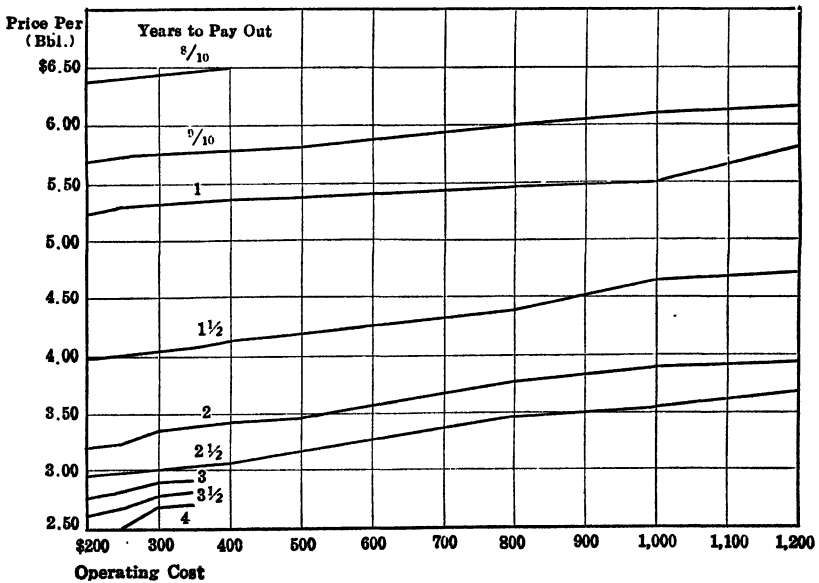
FIG. 5.—NORTH CENTRAL WEST VIRGINIA (BIG INJUN SAND).

· Time to pay out for oil wells having a yearly production of 2000 bbl., a drilling cost of $10,000, and an annual operating expense of from $200 to $1200 per year; oil sells at from $2.50 to $6.50 a bbl.

ginia, an average well was assumed to produce 2,000 bbl. annually, so the price per barrel was plotted against the annual operating cost and the time to pay out was shown as the third variable. The operating cost shows a negligible influence when compared to the much more powerful variable price per barrel. Thus, an increase in operating cost from $200 to $400 annually, or 100 per cent., in a well that requires 2½ yr. to pay out, corresponds to a reduction in price of oil from $3.07 to $2.96, or only 3.98 per cent. In a well that requires 2 yr. to pay out, an increase in annual operating cost from $200 to $1200, or 500 per cent. (something that could happen only under the most unusual conditions, and certainly for only a very brief period), corresponds to a decrease in the price of oil from $3.95 to $3.20, or 75 cents, a cut that the producer has undergone many times without great hardships.

The operator seldom realizes the inestimably great influence the oil market has on his business. A cut of 25 cents will wipe out all his economies in the field. A cut from $6.10 to $2.25 (as Pennsylvania crude experienced in the course of a few months) may reduce a property from a highly profitable to a heavily losing one. It is this influence, as much as the uncertain nature of the mineral deposits, that makes the production of oil such an aleatory occupation.

The curves given in Figs. 1 to 4 show a marked similarity to one another and the curves of all the fields studied are of about the same shape. However, the required time for the different fields varies exceedingly. Tables 1 to 3 show the variation in the time to pay out in the various fields and under market conditions as they existed in the high-price period of 1920, the spring of 1921, and the period of depression during the summer of 1921.

A few conclusions from the curves and tables are possible:

1. The time to pay out varies in all fields.

2. For different fields, the decline curve is an important source of variation.

TABLE 1.—*Time to Pay Out Under Prices for Oil Prevailing During Summer of* 1920

(*Pennsylvania crude*, $6.10; *Mid-Continent*, $3.50; *Salt Creek*, $2.75)

Field	Size of Well, in Barrels (Original Annual Production)								
	1000	1500	2000	3000	4000	6000	10,000	15,000	30,000
North central West Virginia.....	2.5	1.3	0.9	0.7	0.4	0.3			
Glenn pool. Okla	12.5	3.5	2.2	1.6	1.2	0.7	0.4	0.3	
East Osage, Okla................	5.1	3.2	2.2	1.7	1.4	0.8	0.5	0.3	0.1
Salt Creek, Wyo................	*	*	*	*	2.0	1.3	0.8	0.5	0.25

* Data not available.

TABLE 2.—*Time to Pay Out Under Prices for Oil Prevailing During Spring of* 1921

(*Pennsylvania crude,* $3.50; *Mid-Continent,* $1.50; *Salt Creek,* $1.15)

Field	Size of Well, in Barrels (Original Annual Production)								
	1000	1500	2000	3000	4000	6000	10,000	15,000	30,000
North central West Virginia.....	ᵇ	ᵇ	2.1	1.2	0.8	0.5			
Glenn pool, Okla................	ᵇ	ᵇ	ᵇ	10.5	6.2	2.7	1.2	0.7	
East Osage, Okla................	ᵇ	ᵇ	9.9	7.2	5.2	3.2	1.5	0.8	0.4
Salt Creek, Wyo................	ᵇ	ᵇ	ᵃ	ᵃ	5.8	3.6	2.2	1.4	0.6

TABLE 3.—*Time to Pay Out Under Prices for Oil Prevailing During Summer of* 1921

(*Pennsylvania crude,* $2.25; *Mid-Continent,* $1.00; *Salt Creek,* $0.50)

Field	Size of Well, in Barrels (Original Annual Production)								
	1000	1500	2000	3000	4000	6000	10,000	15,000	30,000
North central West Virginia.....	ᵇ	ᵇ	ᵇ	2.1	1.3	0.8			
Glenn pool, Okla................	ᵇ	ᵇ	ᵇ	ᵇ	ᵇ	7.5	2.6	1.1	
East Osage, Okla................	ᵇ	ᵇ	ᵇ	ᵇ	ᵇ	9.6	3.6	1.4	0.6
Salt Creek, Wyo................	ᵇ	ᵇ	ᵇ	ᵇ	ᵇ	11.4	5.8	3.8	1.6

ᵃ Data not available.
ᵇ Well will not pay out.

Blank spaces indicate the well will pay out at less time than nearest figure to left but accurate data are not available.

3. For the same pool or field, of the factors that vary, the time to pay out, sale price for oil and size of well are most important; of much less importance are drilling and operating costs.

4. When properly modified for these variations, by use of a series of time-to-pay-out curves, based on the decline curve of the field, the method is of value either as a supplement to a detailed appraisal or as a check on a rapid approximate valuation.

5. The method may be used in connection with a barrel-per-day value with much less likelihood of error than as though the barrel per day is based on the flush production of the well. After the well has paid out, its production will be settled to a degree permitting a more liberal use of the flat barrel price.

6. The curves show at a glance: The minimum size of paying wells; the minimum sales price for oil on which average wells can be drilled; the influence on profits of a fall or rise in price for oil.

DISCUSSION

H. J. SMITH, Winchester, Ky. (written discussion).—No rule can be depended on for placing valuations on oil properties, with the possible exception of the Pennsylvania, or as it is commonly known the Appalachian field. In most oil fields, value is generally figured on the barrel basis, which I believe, will continue to be the basis, but it is folly to think that all properties in any particular field should be priced on the same per-barrel basis, as there are so many items that enter into the problem.

The time to pay out, of course, really places the value of the property; all oil pools have a depletion curve and there is a time when the curve will be so low that it will be impossible to produce the oil at a profit.

There seems to be a difference of opinion as to when a well has reached what is called "settled production." In the writer's opinion a well is not settled until production stops its abrupt decline and becomes somewhat stationary. This generally takes place after the well is 12 months old, but no general rule can be followed as each well has its individuality. This is one of the most important points in the writer's opinion. Also the part of the pool in which the property is located, must be considered as well as the thickness of the pay sand, proximity of saltwater, and the possible depth of lower sands which would increase the value with a potential possibility.

To obtain a basis on which to value oil properties, many things must be considered, among which are the following:

1. In what sand is the oil found, its longevity, and thickness of pay sand?

2. What were the rock pressure and gas volume when the well was drilled in?

3. Did well flow when drilled in and, if so, how long did it produce without pumping?

4. What was the initial production, and what does the well now produce?

5. What is the age of the well?

6. What are the salt-water conditions?

7. How near to the center of the pool are the wells?

8. How far apart are the wells drilled?

9. Are the wells and leases properly equipped?

10. Has the property been operated by experienced oil operators?

So far as known, the sands that have produced for the longest period of time are the Appalachian. In Kentucky, most of the production is in the Corniferous. The oil sands of Illinois, Indiana, Kansas, and Oklahoma will produce a long time, while pools in Louisiana and Texas are of shorter life.

The next subject that must be considered is the equipment, so as to

determine the amount of money necessary to get the best results from the property.

By knowing the age of the wells and the initial production, a curve can be established to show the rate of depletion, from which an idea as to the life of the wells can be formed. In connection with this the location in the pool must be considered, as well as the thickness of the pay sand, whether the wells came in as gushers flowing by gas pressure, salt-water conditions, the proximity of the wells, and whether the property has been properly handled by experienced operators.

Another item that enters into the valuation is drillable acreage; it is generally assumed that only production is bought, with the drillable acres and potential value, and should be valued separately. Therefore it would be necessary to take into consideration the lifting expense to arrive at a value of the property and not the new production that would be brought in by drilling because in that case one could never arrive at anything but a potential value. In this case the cost of drilling the wells would not make any difference, only what it will cost, per barrel, to produce by careful experienced oil operators, and how long the wells will produce in paying quantities as the lease will some time be fully developed.

Other important factors are the quality or grade of oil and price. A well will produce only a given number of barrels per day, so in fixing a valuation the market-price fluctuations must be considered. The same value cannot be placed on one-dollar oil and five-dollar oil, so a medium mark must be used as a basis.

My experience has shown that in estimating values it should be expected that the properties will pay out in not over four years. Of course, if the potential values are to be considered, they must be arrived at by another method—the valuation of proved territory, which will take into consideration the drilling equipment; but as at some time the property will be fully developed, it is essential to separate the values; that is, the one on production and the other on proved territory.

Properties have been so grossly mismanaged in many instances that management must be considered in making a valuation.

Production Problems in the Grass Creek Oil Field

BY EDWARD L. ESTABROOK,* CASPER, WYO.

THIS paper gives a brief account of the geologic and production problems encountered in the Grass Creek oil field, the methods used in their solution, and the beneficial results obtained from the work. The author wishes to acknowledge the assistance received from J. H. Pearson and Miss Ninettta A. Davis in the collection of the data and the preparation of the cross-sections. Permission for its publication was given by the officials of the Midwest Refining Co. and the Ohio Oil Co.

The Grass Creek oil field is located in Ts. 98–99 W., R. 46 N., Hot Springs County, Wyo. The productive area includes about 2½ mi. (4 km.) on the apex of a great anticlinal fold occupying more than 150 sq. mi. Fig. 1 shows the structural contours of the field. Oil was dicovered in the field in June, 1914; development proceeded rapidly and production rose to a maximum in 1918. The marketed production has been approximately as follows:

	BARRELS		BARRELS
1915	94,073.5	1918	2,946,774.27
1916	1,404,183.97	1919	2,000,584.84
1917	2,780,697.51	1920	1,485,080.94

About 350 wells, 800 to 1500 ft. (243 to 457 m.) deep, have been drilled to reach the upper oil sands; two companies, the Ohio Oil Co. and the Midwest Refining Co., operate the entire field.

The principal producing sands in the field are members of the Frontier Formation, a series of sands and shales occuring in the upper portion of the Colorado Group of the Upper Cretaceous series. These Frontier, or "Wall Creek" sands are also productive in the Salt Creek, Big Muddy, and Elk Basin fields and, therefore, are much the most important reservoirs of light oil in Wyoming. At Grass Creek, the sands in the Formation are more numerous than in the other fields and much less uniform in thickness and character. Usually, eight distinct beds are encountered and in some wells twelve sands have been reported. The formation above

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* Petroleum Production Engineer, Midwest Refining Co.

the Frontier is a firm shale through which an "open hole" will stand for hundreds of feet, if the walls are kept dry. In a wet hole, troublesome caves start from streaks of bentonite, a mineral that readily absorbs water, expands, and breaks down to a soft mud.
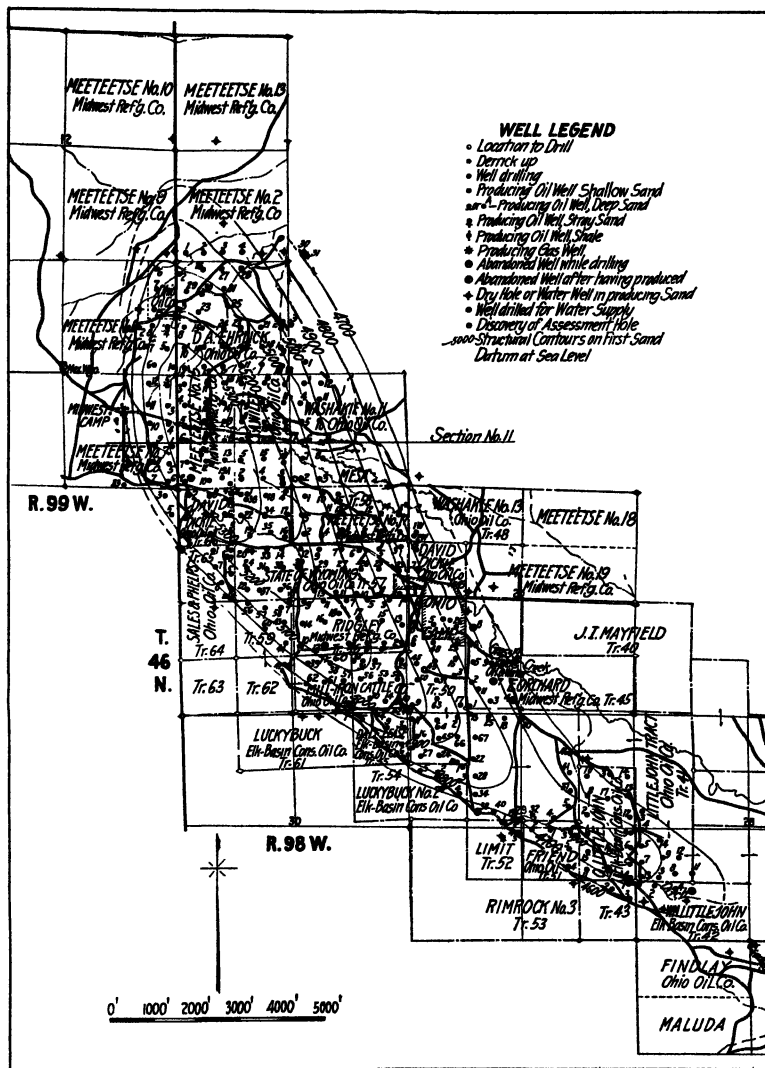
FIG. 1.—GRASS CREEK OIL FIELD.

The upper, or First, sand in the Frontier Formation is reached at depths varying from about 350 ft. (106 m.), in the center of the field, to 1000 ft. (304 m.) at the extreme edge. It ordinarily contains water in

moderate amounts, but sometimes is so shaly and close-grained that very little water entered the well and it is reported in the logs as "dry." In the southeastern part of the field, its development appears to be irregular for many well logs do not record its presence. The Second, Third, Fourth, and Fifth sands produce gas in a small area at the apex of the dome and oil over the rest of the field. The Sixth, Seventh, and Eighth sands are oil bearing over the entire field. All of these sands contain water on the flanks of the anticline outside of the oil-producing area. The relative positions of the water lines (the oil-water contact) in the different sands have not been accurately determined, but the information available indicates no wide variation between them except in the Fifth sand. At the north end of the field, water is found in that sand at least 200 ft. (61 m.) structurally above the water line in the other sands. Some of the first producing wells encountered three oil sands and then found water in the Fifth sand, so a rule was formulated that wells should be stopped after reaching the third oil sand.

The application of this "three sand" rule is responsible for the confusion in identifying the different sands, and for many of the water troubles in the field. The difficulty arose through the fact that a well drilled in an area of about 100 acres around the apex of the anticline encountered several small gas sands, after passing through the water sand, and then found three good oil sands; while a well ½ mi. out would, at about the same depth, have encountered a water sand and then three good oil sands. Drillers accustomed to the gentle dips of the Mid-Continent fields quite naturally considered that the three producing sands in each well were identical and they were accordingly named the First, Second, and Third sands. The water and gas sands were thought to be Strays. Actually, the water sand encountered in both wells was the First sand of the Frontier Formation, the gas sands of the first well were the oil sands of the second, and the oil sands of the first were not reached by the second well. The well on the crest of the anticline had penetrated the entire series of Frontier sands, while the well on the flanks was only half way through them. Intermediate wells encountered the first oil sands progressively higher in the Formation and were stopped after reaching the third oil sand.

The fact that many of the wells did not penetrate all of the oil sands in the Frontier Formation was noted in a report on the field by D. F. Hewett,[1] in 1917, but his findings did not receive the attention they deserved. In the course of some work on a revised structure-contour map of the field, during the winter of 1919, the writer arrived independently at the same conclusion. A comparison of graphic logs made it evident

[1] D. F. Hewett and C. F. Lupton: Anticlines of the Southern Part of the Bighorn Basin. U. S. Geol. Survey. _Bull. 656, (1917) 154.

that a large percentage of the wells would tap additional productive sands if deepened from 10 to 150 ft. Where the lowest sand in the Frontier Formation had been reached in wells on the crest of the anticline, it had been found to be the most productive member in the group; yet over the greater part of the field it had not been touched. The fact that the production of the field had fallen off rapidly during 1919 gave added importance to the possibility of obtaining oil in the lower sands.

The decline in the production of oil from the field had been accompanied by an increase in the amount of water being pumped from the wells. This was interpreted as the usual encroachment of edge water as the oil was removed from the sands, and to be indicative of the early exhaustion of the pool. A brief study of the situation showed that although edge water was appearing in certain wells near the outer limit of production, the greater part of the water being pumped had some other origin. This conclusion was established by the fact that wells at the top of the anticline were producing considerable quantities of water, while wells much lower structurally were yielding clean oil from the same sands. The explanation appeared to be that water from the First sand was entering the wells by leakage through defective shut-offs or by crisscrossing from well to well through some of the upper sands that had not been protected by casing.

In casing the wells, the practice of the field has varied. In the gas-bearing area, a short 10-in. conductor was used and an $8\frac{1}{4}$-in. string landed "in formation" below the water sand. The $6\frac{5}{8}$-in. casing was set with a rubber wall packer in the top of the First oil sand and the gas confined by a bradenhead between the $6\frac{5}{8}$-in. and the $8\frac{1}{4}$-in. Oil was produced through the $6\frac{5}{8}$-in. Outside of the gas area, some companies used the $8\frac{1}{4}$-in. casing to shut off the water in the First sand while drilling was in progress and then 'ulled it after the $6\frac{5}{8}$-in. had been landed on the First oil sand. Other operators used a short $8\frac{1}{4}$-in. conductor and ran only an "oil string" of $6\frac{5}{8}$-in. casing, which was set in formation or landed with a packer at the top of the First oil sand. As the so-called First oil sand was seldom the same sand in two adjacent wells, the casing point was equally varied. The situation is illustrated in Fig. 2. In other wells, the casing had been carried through several of the upper oil sands, where they happened to be only slightly productive, and was set on top of the first highly productive sand.

These discrepancies in the location of the shut-off in adjoining wells permitted the water from the top sand to pass down outside of the casing to the point at which the packer was set and to enter any of the intermediate sands exposed in the hole above that point. In Fig. 2, wells Nos. 7 and 14 on the Ehrlich lease, and No. 8 on Meeteetse 15, illustrate this condition. Water from the first sand in Well No. 7 could pass down outside of the $6\frac{5}{8}$-in. casing and entering the fifth sand cross over into

FIG. 2.—SECTION No. 11, GRASS CREEK OIL FIELD.

wells Nos. 8 and 14. In addition to water entering the wells by circulating through unprotected sands, it seemed probable that leakage of top water past the packers as they deteriorated with age was increasing. This small amount of water was not thought to do any great harm but it certainly did no good and threatened greater injury when the channels enlarged and permitted its entrance in larger amounts. Meanwhile, pumping water with the oil was increasing the lifting costs and making it necessary to steam the oil before delivery to the pipe-line.

In order to show, with the greatest possible clearness, the position of the casing in each well and the depth to the sands that had not been reached, a set of cross-sections similar to that shown in Fig. 2 were prepared, to include every well in the field. The sections were made on tracing cloth to a scale of 100 ft. to the inch. They show the sands encountered, their contents, data concerning shooting, the amount of casing used, and all other relevant information. The search through old drilling records, compiling the information, putting it on the cross-sections, and checking and rechecking to insure its correctness, was a work of months.

The correctness of the conclusion that there were pay sands that had not been reached by many of the wells was tested by drilling well No. 18 on the Ridgley lease in the southern half of the southeast quarter of section 19. This well was put down through all the sands of the Frontier Formation and each one tested for its content of oil and water. The First sand was found to be water-bearing as usual; the Second, which had been an oil sand when the surrounding wells were first drilled, was partly filled with water; the Third and Fourth sands, which had been yielding oil in offset wells for several years, were still light producers; the four lower sands, which had not been reached in the vicinity, gave an initial production of 90 bbl. in 24 hr. when pumped on the beam. Seven months later, nine wells out of the eighteen on this lease had been deepened and the production had increased 250 per cent. Similar results are being obtained on other leases in the field.

The problem of stopping the circulation of water from the First sand downward, and later through unprotected sands, appeared to be one of great difficulty if undue expense was to be avoided. It was impossible to replace the $8\frac{1}{4}$-in. water strings that had been pulled, and the $6\frac{5}{8}$-in. casing could not be reset without moving on a heavy drilling machine at considerable expense. Field superintendents R. B. Miller, of the Ohio Oil Co., and L. L. Bechtel, of the Midwest Refining Co., evolved a cheap simple remedy for the situation. A small galvanized iron tank was moved to the well, a thick fluid mud was mixed, and then poured into the well around the outside of the oil string. A well in which the $8\frac{1}{4}$-in. water string had been used and pulled and in which there had been no serious caving would hold around the $6\frac{5}{8}$-in. oil string about 1 bbl. of

mud to 20 ft. of hole.   If a well took the expected quantity of mud, it was assumed that there had been no great amount of caving and that the mud had filled all of the open space around the outside of the 6⅝-in. casing.   If the level of the mud fluid lowered slowly in the hole and yet no mud was pumped out with the oil, it was assumed that the mud was slowly entering and sealing the sands exposed in the hole and into which the water from the First sand might be entering.   More mud was added until the space around the casing remained full.   Occasionally the mud appeared at once in the oil, showing that the casing, the packer, or the formation shut-off were defective and that water from the First sand was free to pass directly down the hole into the oil-bearing sands.   To repair such wells, it will be necessary to pull the casing, set on a new seat, and remud.   In other wells, the hole outside of the 6⅝-in. casing was found to be clogged with cavings or other material and very little mud could be poured into it; nothing can be done to these until a machine can be moved on to handle the casing.   There are so many wells to be deepened and repaired that the order in which they are worked upon is determined by their condition.   When the casing situation and the presence of water with the oil in surrounding wells indicates that any particular well is feeding water into the sand, attention is given to it as early as possible.   The mudding is being carried on during odd times by the well-pulling crews and substantial decreases have been noted in the quantity of water being pumped with the oil on various leases.   No rubber packers are used in this repair work, for their value is only temporary and it is desired that the work being done shall endure as long as the field lasts.

The work outlined involves the deepening and repairing of about 280 wells, and its accomplishment will require several years, especially as its progress has been delayed by the depression in prices of oil.   Enough has already been done to demonstrate that there are substantial reserves of oil in the Grass Creek field that can quickly be made available when needed.   The entrance of top water into the oil sands is being cheaply and effectively stopped, and the only serious problem in the future will be the handling of the wells during the gradual encroachment of the edge water.   If the rising edge water in each sand can be confined to its own sand body and prevented from flooding the adjoining sands, it will be possible ultimately to recover a large percentage of the oil content of the field.   The water rising up the sides of the anticline under its natural hydrostatic pressure will gradually clear the sand pores of all of the recoverable oil and carry it toward the apex of the dome, where a few wells will continue to produce until the field is entirely exhausted.   Considerable quantities of oil might be lost if water from one sand floods another and bodies of oil sand between wells become entirely surrounded by water.   The greater fluidity of water is likely to cause the rising flood

of edge water to pass around such an isolated body of oil sand and leave it undisturbed.

The study of the problem of handling to the best advantage this advance of edge water must be carried on during the progress of the work of deepening and of resetting casing that has been described above. Whenever edge wells are drilled or worked on, the sands must be tested for their oil or water content and the productive sands protected from the water of the exhausted sands. Gradually, information will be obtained that will make it possible to determine in advance of work which sand in any edge well is yielding water and which should still produce clean oil. As they become exhausted, the sands should be cased or mudded off in succession until every oil sand in the field has been completely drained.

Subsurface Conditions on Portion of Arches Fork Anticline

BY KENNETH COTTINGHAM, COLUMBUS, OHIO

(New York Meeting, February, 1922)

THE area described herein is situated on what is known as the Arches Fork anticline in Roane and Calhoun Counties, W. Va. When the area was first mapped, it was felt that well records would give a more intimate idea of conditions, both structural and lithologic, than could be expected from surface horizons. The surface formations are chiefly members of the Monongahela series, though northward the hilltop outcrops are those of the Dunkard. Neither series offers a key horizon sufficiently definite to be satisfactory, although the structure as determined from well records was later checked against two of the more persistent surface sandstones.

In mapping the subsurface structure, the top of the Big Lime (Greenbrier) was employed as the key horizon. The average depth to the top of the Big Lime varies from 1600 to 2000 ft. (487 to 609 m.), while the average thickness as indicated by well records is 113 ft. (34 m.). The oil-producing formation is the Big Injun, lying just below the Big Lime. In two instances, well records show the Lime and Injun separated by a few feet of "break." The Big Injun is recorded as having a variable thickness. Seven logs report it as missing and one gives it the maximum thickness of 105 ft. (32 m.), while the entire average for thickness is 19 ft. (6 m.).

RELATION OF STRUCTURE TO THICKNESS OF BIG INJUN

The seven wells reporting the Big Injun as absent were located near the anticlinal axis. Moreover, the general tendency is a thickening of the Injun on the flanks of the anticline, increasing with distance from the axis. Manifestly, this apparent anomaly has no stratigraphic basis, as the actual thickness of the Big Injun and the overlying Big Lime was determined by original deposition which no amount of subsequent folding could appreciably alter. Fig. 1 is a skeleton map of structure based on the top of the Big Lime; Fig. 2 is a cross-section showing the thickness of the Big Injun.
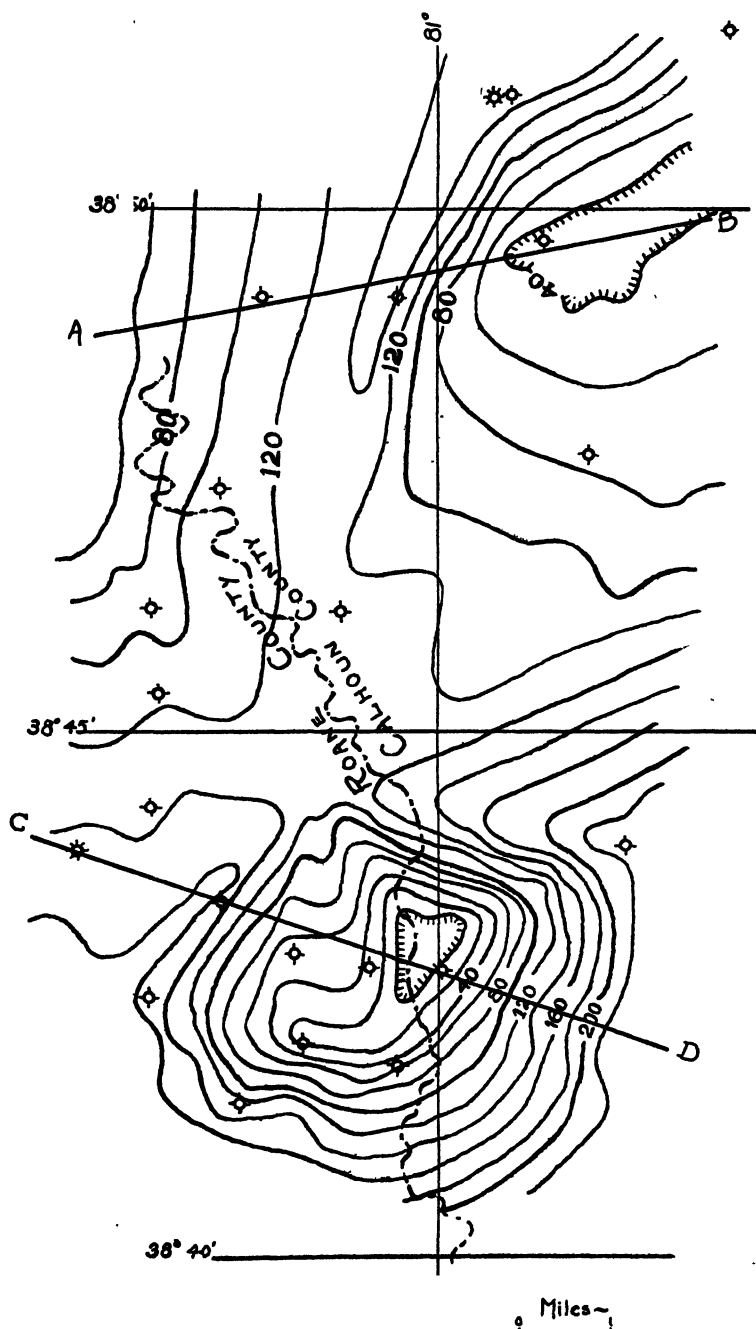
FIG. 1.—STRUCTURE TOP OF BIG LIME. CONTOUR INTERVAL, 20 FT.; DATUM, 1000 FT. BELOW SEA LEVEL.

The recorded thickness of the Big Injun seems to have been based on the degree of "show" as an oil sand rather than on the actual amount of sand present.   Extreme thicknesses of Injun, such as those in excess of 40 ft. (12 m.), are undoubtedly in part Big Lime.   On the other hand, when the Injun is reported absent, the inference is that it was included as part of the Big Lime.   It would seem that two formations as distinct as the Big Lime and Big Injun could hardly be confused; yet the tendency to so interpret conditions cannot be charged to inadvertence nor error where several different operators are concerned, and when the development was carried on over a period of 20 years.

It may be assumed that the Big Injun is more regular in thickness than the logs indicate, also that the actual thickness of the Big Injun bears no such direct relation to structure.   Fig. 2, rather than defining the true thickness of the sand, may indicate the attitude and capacity of a
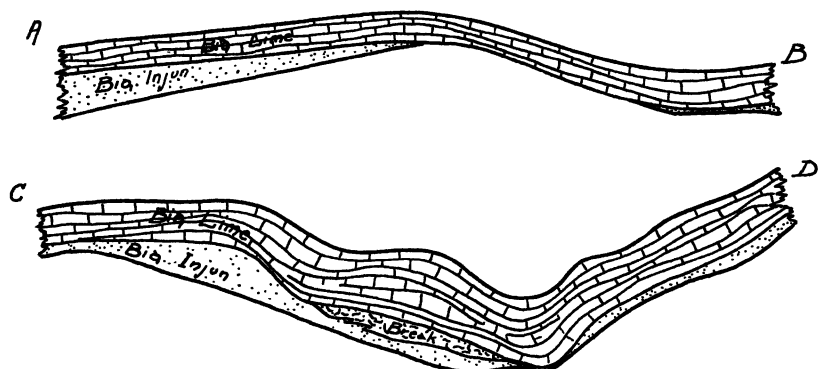
FIG. 2.—CROSS-SECTIONS A–B AND C–D.

former reservoir from which the oil has retreated with a subsidence of the water level.   The sand map, Fig. 3, shows that the apparent thickening of the sand west of the 'no-sand area' balances the amount of dip on that side of the structure.   Plotted in cross-section, it shows the base of the Injun dipping conformably with the top of the Big Lime, while the top of the Injun remains nearly level.   This upper level undoubtedly lies in the basal portion of the Big Lime.

The no-sand area is slightly east of the anticlinal axis.   Likewise, the oil sand is asymmetrical with respect to the structure, being thicker at a given distance on the western flank of the fold and lying structurally lower on the eastern flank.   This again suggests an advance of water from the northwest, away from the axis of the Appalachian geosyncline. The minor thickening of the sand southwest of the no-sand area bears out the suggestion.   The thickening of 20 ft. south of the same area is in part a break between the Big Lime and the Big Injun.
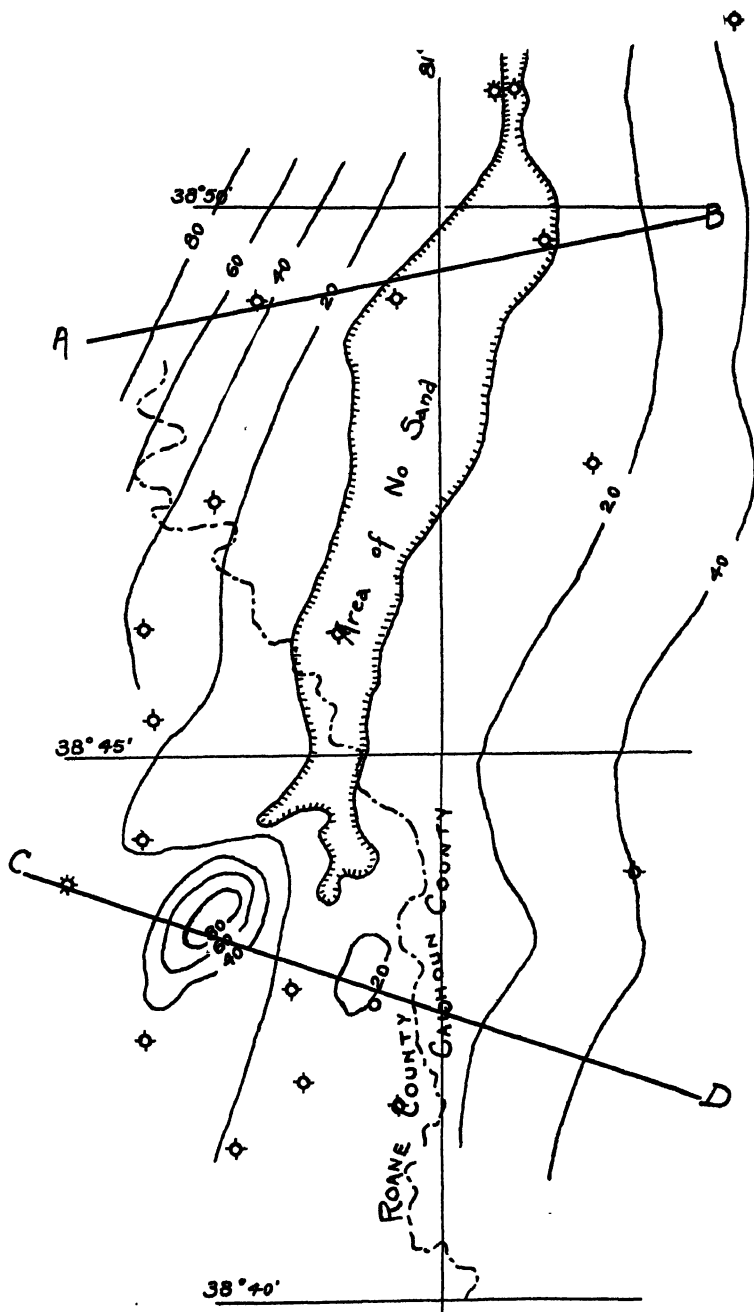
FIG. 3.—SAND MAP SHOWING THICKNESS OF BIG INJUN.

RELATION OF STRUCTURE TO PRODUCTION

All of the wells drilled along the axis of the anticline are reported to have yielded gas in some amount. Two are productive today, though the volume of gas is extremely low. There is also a small gas well on the western part of the nose, west of the structural basin. The sand in the structural basin yields a light production of oil different in quality from that produced a few miles to the west. None of the wells in the basin was commercially profitable and all were threatened by the water immediately after completion. The best production lies just west of the area outlined on the maps; it is known as the Clover field and apparently is situated on a narrow terrace.

CONCLUSION

The evidence seems to indicate an original advance of water from the northwest and a subsequent retreat in that direction. The leveling of the oil sand on the western flank of the anticline, the position of the no-sand area, and the asymmetry of the sand with respect to structure bear out this idea. The writer has found indications of similar conditions in adjoining portions of West Virginia, but the drilling was not extensive enough to give the data desired.

Whatever theoretical interpretation of the facts may be given, one practical demonstration is the frequent unreliability of an oil sand as a key horizon. In the present case, the sand indicates the extent and attitude of a former oil accumulation without giving an accurate conception of the structure.

DISCUSSION

DAVID B. REGER,* Morgantown, W. Va. (written discussion).—The region under discussion is one of the few localities of the West Virginia oil fields where the Big Injun sand is lenticular. Many thousand wells have been drilled through its horizon in the western part of the state, their records showing that it varies in thickness from 25 to 250 ft., its average being perhaps 100 ft.; the places where it is less than 50 ft. thick are comparatively few. In the pools where it is productive, the pay streak is seldom more than a small fraction of the total thickness, being usually 3 to 6 ft.; in some instances a second pay is found, and in rare cases a third. In well logs the custom of the driller is to give the total thickness of this sand from top to bottom, irrespective of pays, and to record the latter as a parenthetical item for the information and use of the operator. For this reason the view that only the pay streak has been recorded along the Arches Fork anticline is open to question, as nowhere in the state have soft layers of 20 to 40 ft. been found. If

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* Assistant Geologist, West Virginia Geological Survey.

such a condition had been found in the region under discussion, a vastly prolific oil and gas pool would have surely resulted.

Another point on which an adverse opinion may be reached is the interpretation of the structure as based on the top of the Big Lime, a horizon that has great constancy throughout the state. As the datum plane selected (1000 ft. below sea level) makes necessary the use of depression or minus contours, the writer has prepared a map, Fig. 4, of the same region from information secured by staff members of the West Virginia Geological Survey and published in the Wirt-Roane-Calhoun report of the Survey. The records of several wells outside the limits of the figure have been used to secure proper control.

The right-hand column in Table 1 shows the significant elevation above or below the Cottingham datum:

Well No.	Surface Elevation	Big Lime		Cottingham Datum (−1000 ft.)
		Top	El. of Top below Tide	
110	770B	1823	1053	−53
116	825B	1755	930	+70
119				
120	837B			
162	1045B	1960?	?	?
164	865B			
164A	(875)	1628	(753)	(+247)
164B	915B			
165	900L	1800	900	+100
165A	750B			
338	750B	1780	1030	−30
342	690L	1701	1031	−31
343	670B	1790	1128?	−128?
344	675B	1730	1055	−55
345	690B	(1718)	(1028)	(−28)
346	810B	1734	924	+76
347	815B	1744	929	+71
348	700B			
349	730L	(1675)	(945)	(+55)
351	725B	1575	850	+150
352	730B	(1500)	(770)	(+230)
360				
362	845B	1800	955	+45
372	745B	1700	955	+45
372A	750B	(1685)	935	+65
373	930B	1943	1013	−13
374	850B	1492	642	+358
375	775L	1460	685	+315

· NOTE.—Well numbers refer to Wirt-Roane-Calhoun report; *B* signifies an aneroid elevation, closely checked on spirit levels; *L* signifies a spirit level; parentheses indicate doubtful data.
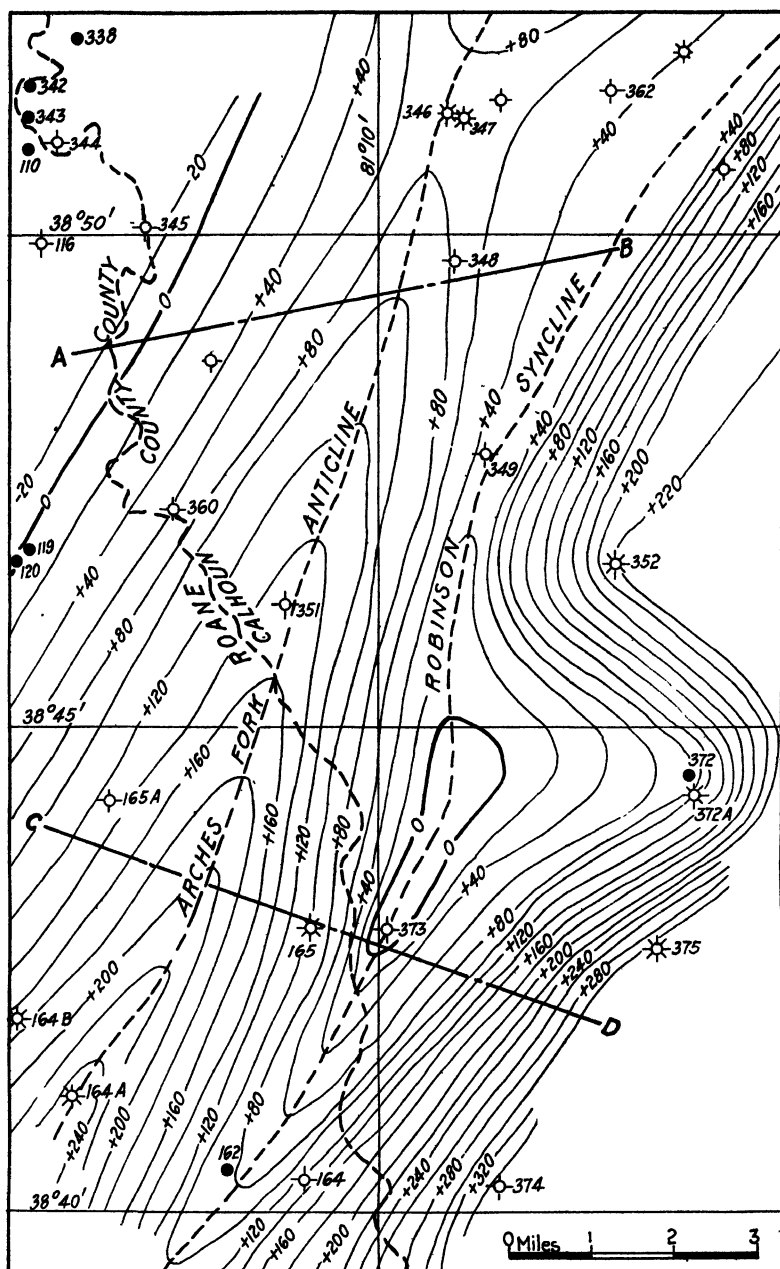
FIG. 4.—STRUCTURE TOP OF BIG LIME, CONTOUR INTERVAL, 20 FT.; DATUM, 1000 FT.
BELOW SEA LEVEL.

Fig. 4 shows that there is no saddle along the Arches Fork anticline in the vicinity of lat. 38° 45′ but that one occurs much farther north at well No. 346; also no apparent high saddle occurs along the Robinson syncline just north of lat. 38° 45′. That any pronounced depression could occur in the vicinity of lat. 38° 50′ does not seem possible in view of the Lime elevation (+45) at well No. 362. In certain cases there is a contour discrepancy of 50 to 80 ft. between the two figures, which is far in excess of any probable barometric error where many bench marks are available for checking the instrument. In general, Fig. 4 is much more nearly in harmony with the surface structure prepared on the Washington coal by Ray V. Hennen, and published in the report mentioned, and also more in harmony with the well-known type of structural wrinkles of the Appalachian region.

The conclusion that there was an original advance of water from the northwest and a consequent retreat to its source is also open to question, as water can neither advance nor retreat through a region where there is a non-porous barrier, such as is shown along the anticline in Fig. 3. It is rather to be supposed that the water in the Robinson syncline is entirely connate, or possibly in part derived from a more northern region along the axis of the basin.

By far the most interesting conclusion that may be developed from a study of the well logs of this region is that the absence of sand over this rather extensive territory affords an explanation for the anomalous production record of the Clover Pool, lying just west of long. 81° 15′ and just south of lat. 38° 45′, where many wells kept up their original production for several years and in some instances actually increased production for a year or more after completion. The apparent reason for this unusual phenomenon is that the southeast migration of oil from the Appalachian geosyncline was effectually halted by the absence of sand along the Arches Fork anticline, thereby enriching to an abnormal degree the sand content in the Clover Pool.

Federal Taxation of Mines

REPORT OF COMMITTEE OF AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS

THE Committee submits the following report:

The General Committee met in the Treasury Building at Washington on Oct. 6 and 7, 1919. At the first meeting, Cornelius P. Kelley was appointed chairman, and Paul Armitage, secretary of the Committee.

Subcommittees were appointed on Invested Capital, on Mine Valuation, and on Depletion and Depreciation.

These subcommittees were directed to meet separately and to report on Oct. 7 to the General Committee. On Oct. 7, the reports of these committees were submitted to the General Committee and on motion were received and referred to the Executive Committee for its information to be used as it deemed wise.

A motion was carried that the Executive Committee be constituted a permanent committee to continue the work of the Committee, with full power to add to and increase its membership; that as thus increased it should constitute a permanent organization to act and coöperate with the Treasury Department in matters of mining taxation under the Federal Income and Excess Profits Tax Laws.

Thereafter the Executive Committee met and formed a permanent committee as follows:

Cornelius F. Kelley, chairman, 42 Broadway, New York City.

J. L. Darnell, Office of the Commissioner of Internal Revenue, Washington, D. C.

J. Parke Channing, 61 Broadway, New York City.

Walter Douglas, 99 John St. New York City.

R. V. Norris, 542 Second National Bank Bldg., Wilkes-Barre, Pa.

J. E. Spurr, care of *Engineering & Mining Journal*, 10th Ave. and 36th St., New York City.

W. G. Swart, 808 Selwood Bldg., Duluth, Minn.

J. P. Finley, 45 Cedar St., New York City.

W. A. Williams, Bartlesville, Okla.

F. S. Peabody, care of Peabody Coal Co., McCormick Building, Chicago, Ill.

Matthew C. Fleming, 71 Broadway, New York City.

H. B. Fernald, 54 Wall St., New York City.

William G. Mather, Cleveland, Ohio.

Walter Fitch, Eureka, Utah.

Arthur Thacher, 900 Security Bldg., St. Louis, Mo.

John P. Gray, Coeur d'Alene, Idaho.

Paul Armitage, Secretary, 233 Broadway, New York City.

At this meeting the Executive Committee appointed chairmen of subcommittees representing the following: (1) Coal, (2) Iron, (3) Copper, (4) Lead and Zinc, (5) Precious Metals, (6) Law, (7) Accounting, each one of the said chairmen to appoint and select a subcommittee either from the members of the Executive Committee or otherwise, said subcommittee to take up the special questions relating to the particular subject and report to the General Committee.

The chairman thereupon appointed the following:

R. V. Norris, chairman of the Subcommittee on Coal.

W. G. Swart, chairman of the Subcommittee on Iron.

J. Parke Channing, chairman of the Subcommittee on Copper.

J. P. Finlay, chairman of the Subcommittee on Lead and Zinc.

J. E. Spurr, chairman of the Subcommittee on Precious Metals.

Matthew C. Fleming, chairman of the Subcommittee on Law.

H. B. Fernald, chairman of the Subcommittee on Accounting.

The Committee thereupon adjourned to meet at Atlantic City on Oct. 20, 1919.

The Executive Committee met at Atlantic City pursuant to adjournment on Oct. 20 and 21, 1919.

The various subcommittees made their reports, and, after a full discussion, the following were adopted as the recommendations of the Committee to the Bureau of Internal Revenue:

I. INVESTED CAPITAL

On the subject of invested capital, the Committee recommends as follows:

Revenue Act of 1917

That Art. 63 of Regulations 41 be amended by striking out the following lines:

The adopted value shall not cover mineral deposits or other properties discovered or developed after the date of conveyance, but shall be confined to the value accurately ascertainable or definitely known at that time.

And by adding to said Art. 63 of said Regulations 41 the following paragraph dealing with surplus:

But in the case of mines and mineral deposits where legitimate expenditures have been made for the purpose of developing known ore-

bodies, or mineral deposits, and ascertainable values have been added to the property, or where as a result of development undertaken, exploration conducted or the adaptation of improved processes, deposits or portions thereof unknown or without value at the date when the mining property was acquired, or which were not then susceptible of most efficient beneficiation have been developed and given a value or an additional value which can be ascertained with reasonable accuracy, such value shall be regarded as surplus and shall be included in invested capital.

Art. 63 as amended would therefore read as follows:

When tangible property may be included in surplus: Where it can be shown by evidence satisfactory to the Commissioner of Internal Revenue that tangible property has been conveyed to a corporation or partnership by gift or at a value, accurately ascertainable or definitely known as at the date of conveyance, clearly and substantially in excess of the cash or the par value of the stock or shares paid therefor, then the amount of the excess shall be deemed to be paid in surplus.

Evidence tending to support a claim for a paid-in surplus under these circumstances must be as of the date of conveyance, and may consist, among other things, of (1) an appraisal of the property by disinterested authorities, (2) the assessed value in the case of real estate, and (3) the market price in excess of the par value of the stock or shares.

But in the case of mines and mineral deposits where legitimate expenditures have been made for the purpose of developing known orebodies, or mineral deposits, and ascertainable values have been added to the property, or where as a result of development undertaken, exploration conducted, or the adaptation of improved processes, deposits or portions thereof unknown or without value at the date when the mining property was acquired, or which were not then susceptible of most efficient beneficiation, have been developed and given a value or an additional value which can be ascertained with reasonable accuracy, such value or additional value shall be regarded as surplus and shall be included in invested capital.

The Committee understands that the above regulation as to paid-in surplus applies irrespective of the date at which the surplus was paid in, whether before or after Jan. 1, 1914.

It understands that Art. 55 of the Regulations 41 has no bearing whatever on the question of "Paid-in surplus," but was inserted in the regulations solely for the purpose of making clear the fact that property acquired before Jan. 1, 1914, at a value less than the par of the stock issued therefor, may be valued as of Jan. 1, 1914, it being, as the article states, one of the few cases in which the law permits allowance to be made for appreciation.

Your Committee is of the opinion that, if proper allowance be made for surplus as above defined, many of the objections heretofore raised to Section 207 of the Act of 1917 on the ground of inequality and discrimination in taxation will be obviated.

Revenue Act of 1918

We suggest one amendment which we regard as important, and which if adopted will bring the definition of invested capital under the Act of 1918 into line with that suggested above under the Act of 1917.

We, therefore, suggest that Art. 838 of Regulations 45 be amended by adding at the end thereof the following:

But in the case of mines and mineral deposits where legitimate expenditures have been made for the purpose of developing known orebodies, or mineral deposits, and ascertainable values have been added to the property, or where, as a result of development undertaken, exploration conducted, or the adaptation of improved processes, deposits or portions thereof unknown or without value at the date when the mining property was acquired, or which were not then susceptible of most efficient beneficiation, have been developed and given a value or an additional value which can be ascertained with reasonable accuracy, such value or additional value shall be regarded as surplus and shall be included in invested capital; such earned value not being "value appreciation" within the meaning of the last paragraph of Art. 844.

II. VALUATION OF MINES

The Committee arrived at the conclusion that it would be desirable to divide mineral properties into two classes, Class I and Class II.

In Class I are included mineral properties in which the tonnage or other unit has been determined with reasonable accuracy.

In Class II are to be included all other deposits.

As to Class I, the Committee considered methods of arriving at the present value of mineral property, and methods of depletion, and has arrived at the following conclusions:

A proper value of a mining property is the present value of the prospective net earnings taking into account probable variations in output and value discounted by recognized sinking fund methods at a fair rate of interest with sinking fund at 4 per cent. interest, or by calculations by standard annuity methods. But other recognized methods of valuation acceptable to the Department may be used.

In lieu of estimated net earnings, where mining on a royalty basis is customary, royalty prices may be used in valuation taking into consideration the trend of such prices.

No mine shall be valued on an estimated operating life exceeding 45 years.

Ores of different grades, location, and probable time of extraction in a mining property may be classified separately and valued accordingly.

Nothing herein contained shall be understood to prescribe a method of valuing separately the equities of lessor and lessees in a mining property.

Mines in Class II may be valued in the manner prescribed for Class I, but there will be a difference in the manner of determining the principal underlying factors, namely, the quantity and quality of ore and the life of the mine.

In Class II, sole reliance cannot be placed in the development of ore on the date of valuation, but concurrent evidence, such as the habit and type of orebodies in the mine itself, the characteristics of the district in which it occurs, the rate of development through exploration, the strength of mineralization, the stage of the operating life of the mine and any other satisfactory evidence may be used to establish a reasonable estimate of the required factors.

III. DEPLETION

On the question of depletion, the Committee makes the following recommendations:

Depletion should be a sum calculated to return to the owner, free of tax, the cost or value, as the case may be, of his mineral property.

This should be calculated either on a unit basis by dividing the estimated value or cost by the estimated units, or as a percentage of the annual income before deducting depletion, the said percentage to be the ratio of the cost or value of the property to the total estimated earnings.

The Committee is of the opinion that in view of the fact that the equitable apportionment of depletion between lessor and lessee depends in large measure on the terms and construction of the contract of lease, and of the law, no recommendation should be made by the Committee until the legal questions have first been satisfactorily and authoritatively determined.

The Committee recommends that distributions to stockholders made from depletion reserve are liquidating dividends, and do not constitute taxable income to a stockholder; and that said distribution under the law may be made from such reserve irrespective of the condition of the surplus and undivided profits account of the corporation.

The Committee is of the opinion that to prohibit such distribution until the surplus and undivided profits of the corporation have been first distributed, works a grave injustice to stockholders of mining corporations and is contrary to the letter and spirit of the revenue law.

IV. DISCOVERY

On the question of discovery, the Committee recommends as follows:

1. That Art. 219 of Regulations 45, as revised, be approved, and that it be amended by inserting after the words "proving and development" at the end of the first paragraph thereof, a new subdivision to be known as subdivision (c), to read as follows:

(c) The proving by the taxpayer of the commercial value of a mineral or ore deposit by the development, refinement or perfection of known methods or processes of mining or metallurgy, or both, or by the discovery and application of new methods of mining or metallurgy at a cost materially less than the commercial value of the deposit thus proven or created. The estimation of the value of the deposit must be made as of a date not later than thirty days after the commercial value of the deposit has thus been proven.

2. Further ore discovered either by further development or exploration, whether this ore be an extension of a previously known orebody or a new orebody or by improved processes of treatment, and not included within the previously estimated value or estimated life of the mine, may be valued for depletion purposes following such discovery or discoveries.

Respectfully submitted,

CORNELIUS F. KELLEY, Chairman.
PAUL ARMITAGE, Secretary,
233 Broadway,
New York City.

Oct. 21, 1919.

Colloid Chemistry and Metallurgy*

By Wilder D. Bancroft, Ithaca, N. Y.

Albert Sauveur, Cambridge, Mass. (written discussion).—In his introductory paragraph, the author states that he has come to bury Beilby, not to praise him. The paper, however, gives the impression that while he may have dug the grave, Beilby is still at large. I am not one of those whom the author calls enthusiastic supporters of Beilby's amorphous theory and in support of this claim quote from a book I am writing: "Attractive as the amorphous theory is, and notwithstanding the ability with which it has been presented and the renown of its sponsors and supporters, it should not be accepted without reservations as a demonstrated fact. . . . So long as it explains more satisfactorily than other theories a greater number of phenomena in which it may play a part, we should use it in preference to others as the best available. Let us avoid the danger, however, into which too many fall, of accepting as indisputable truths, statements and theories emanating from those whom we hold in great esteem. It is not what those men themselves would wish us to do. I am entitled, I think, to occupy a position in the middle ground. Neither should Howe be classified with the enthusiastic supporters for he has written that the theory should be classed rather among the precious working theories than among those firmly established," a very wise and prudent statement. Jeffries, on the other hand, I would not hesitate to describe as an enthusiastic supporter. The author reproaches the advocates of the amorphous theory with having taken whatever properties they needed and with having assigned them arbitrarily to the hypothetical amorphous phase. He then applies their method to small crystals. If the existence of an amorphous phase is assumed, its greater tenacity and greater hardness is satisfactorily explained in the words of Howe: "This greater strength is natural, because the amorphous state lacks the cleavages and slip planes which weaken crystalline substances." Can the greater strength and hardness claimed for small crystals be as satisfactorily explained? Nor do I think that the cold deformation of metals through successive slips within the crystalline grains can be accounted for as acceptably by the "small crystals theory."

* Discussion of paper printed on pages 604–617.

The argument that we know that small crystals exist while the existence of amorphous metal is a mere assumption does not suffice to reject the amorphous theory. Following that line of reasoning, many valuable theories would disappear, including the atomic theory. If metals behave as if the amorphous state was assumed under the conditions described, that is the essential thing. We need not be greatly disturbed by the possible non-existence of that state. If matter behaves as if it were made of atoms, we may accept with enthusiasm the atomic theory without excessive worry as to the possible non-existence of atoms. So long as things take place in the physical universe as if matter existed, we need not be alarmed by the claims that it has no real existence.

It does not seem to me that the author's claim that metallographists have been too uncritical in their attitude toward the amorphous theory is justified. The majority of them have accepted it with great caution and will continue to do so. They will use it as a tool so long as a better one is not at hand. What the author offers in its stead will not, I believe, cause many to discard it.

BIOGRAPHICAL NOTICES

Henry Marion Howe

HENRY MARION HOWE died at his residence in Bedford, N. Y., on May 14, 1922, of a malady from which he had been suffering for more than fifteen months.

Doctor Howe was prominently identified with the Institute, which was the first technical society he joined, and in which he held office at several different times, from 1879, when he was vice-president, to 1893, when he served as president.

He was a loyal and active member of the Iron and Steel Committee, and did more than any one else to enrich the publications of the Institute in the field of iron and steel metallurgy. At the meeting of the directors on May 19, the following resolution was passed:

The Directors of the American Institute of Mining and Metallurgical Engineers place upon record their profound sorrow at the death, on May 14, 1922, of Past President Henry Marion Howe. Mingled with their regret at his loss is their realization that the honor and credit reflected upon the Institute by the eminent attainments of their Past President, the acknowledged leader among the scientific men of the world who have collaborated in developing the science of the structure and properties of iron and steel, is an undying memory in which the Institute and its members may justly take pride, even though the distinguished author is no longer in their midst.

The Directors are happy to record that Doctor Howe used this Institute as the principal means of distributing to the world the results of his great work in research, investigation and interpretation. He was loyally devoted to the Institute; he joined it the year that he graduated from college, which was also the first year of the Institute's life; he was a frequent attendant at the conventions in all parts of the country, and enriched the TRANSACTIONS not only by his original contributions but by his constructive and valuable discussions of the papers of others, and by his skill as a presiding officer at many of the sessions. Eminent son of eminent and distinguished parents; possessor of a mind greater than that of most men for its ability to grasp scientific truth and interpret the results of investigation; gifted with a command of the English language which was equaled by few men of his time; expert organizer of research; the American and foreign engineering societies, universities and governments have honored you as few engineers have been honored, and you in return have reflected honor upon them by your membership and your contributions.

IT IS RESOLVED, That this minute be spread upon the records of the Institute, be published in the magazine, and be sent to the family of our honored colleague.

A record of Doctor Howe's professional career, of his principal published works, of his more than three hundred technical monographs, and a very long list of the honors conferred on him by American and foreign technical societies, governments and universities, was given

in the Bulletin of the Institute for July, 1917, in connection with the award of the John Fritz Medal to Doctor Howe on May 10, 1917.

On that occasion, Dr. Rossiter W. Raymond, in his address, spoke of Doctor Howe and his professional achievements in part as follows:

"Henry Marion Howe was born March 2, 1848, at Boston, Mass. His father was Dr. Samuel G. Howe, famous for his service to Greece
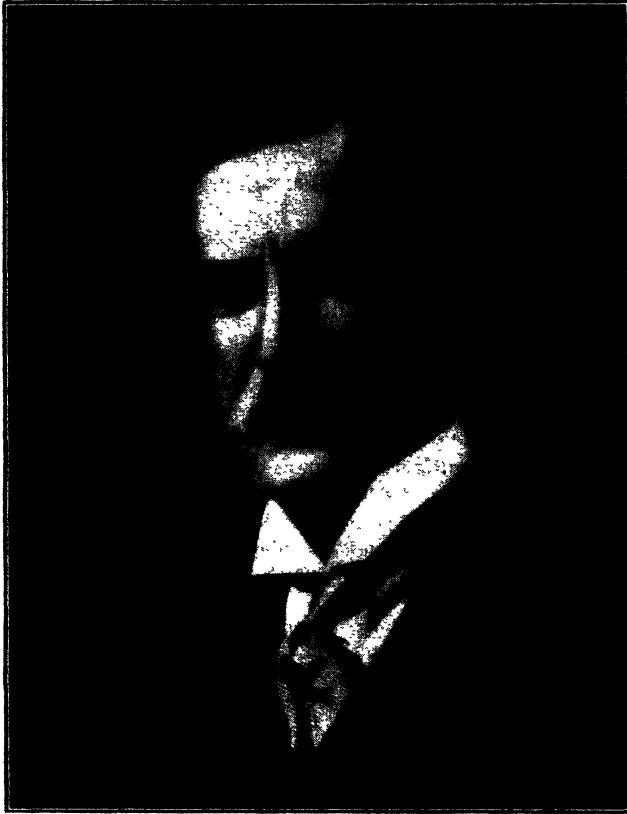
HENRY MARION HOWE

in her war for independence (from 1824 to 1830) and later for his labors in the instruction of the blind. His mother was Julia Ward Howe, author of the Battle Hymn of the Republic, and leader in many reforms. It was a good stock on both sides, making him heir to intellectual keenness and refinement, the capacity for both enthusiasm and perseverance, a passion for the pursuit of knowledge, and a gift of clear and felicitous statement.

"This inheritance was improved by a liberal education . . . . We may be sure that Professor Howe's easy command of his field in technical literature owes much to the circumstance that he was graduated

in 1865 from the famous Boston Latin School, and, four years later, received his degree of Bachelor of Arts from Harvard College. Thus equipped, he entered the Massachusetts Institute of Technology, which gave him in 1871 the degree of 'Graduate in the Department of Geology and Mining Engineering'—a cumbrous title for which the institution substituted, a few years later, that of 'Bachelor of Science.' Harvard made him Master of Arts in 1872, and Doctor of Laws in 1905. But upon this basis of wide and liberal culture, it was necessary to his future achievements that he should lay another foundation of acquaintance with practice; and this he did during the next dozen years, as a student in the steel works at Troy, manager of works at Pittsburgh, and designer and builder of the works of the Orford Nickel and Copper Co. at Capelton and Eustis in the province of Quebec, and at Bergen Point, New Jersey. His experience in the metallurgy of copper bore legitimate fruit in the publication at a later period of his first book, "Copper Smelting;" but before this appeared (in 1885), he had already become known through his technical papers as an acute observer and reasoner, by no means averse to friendly controversy.

"Since it is my present duty to offer, not so much a symmetrical and complete account of Professor Howe's activities as a description of the particular labors in recognition of which he receives today from his brother engineers the John Fritz gold medal, I shall pass lightly over the general features of his career, merely observing that from 1883 to 1897 he resided at Boston, and, besides his private practice as a consulting metallurgist and expert witness in metallurgical patent suits, was lecturer on metallurgy at the Massachusetts Institute of Techonology; that in 1897 he was called to the professorship of metallurgy in Columbia University, from which position he retired in 1913 with the title of Professor Emeritus; and that for the last ten years he has declined as far as practicable all professional business, in order to devote himself exclusively to the completion of what had clearly become the great scientific mission of his life. To this end, he has maintained at his own expense a special laboratory of research.

"It gives me pleasure to fancy, whether or no the fancy be also fact, that I remember the beginning of that mission, forty-odd years ago. In 1871, the first year of the existence of the American Institute of Mining Engineers, Mr. Howe, then just graduated from the Massachusetts Institute of Technology, became a member of the new organization. His two earliest contributions to its TRANSACTIONS, Blast-furnace Economy, in Vol. III, and Thoughts on the Thermic Curves of Blast-furnaces, in Vol. V, indicated already that he was making a scientific study of practice. But between the dates of these papers a famous controversy was inaugurated in the forum of the Institute by the brilliant paper of Alexander L. Holley, primus inter pares, entitled

"What is Steel?" In this controversy, eminent metallurgists like Wedding, chemists like Prime, and expert captains of industry like Metcalf, vigorously took part; but of all the knights of the tourney, none rode a straighter course or laid in rest a sharper lance then Howe, whose paper on The Nomenclature of Iron ardently advocated a scientific as distinguished from a commercial nomenclature.

"The science of metallography, at least so far as it relates to iron and steel, may be said to have begun with the observations of Sorby on the microstructure of iron, reported in 1864 and 1868. Martens published independently in 1878. But already in 1868, Tschernoff had enunciated the chief laws that govern the metallography of iron. These were supplemented by the appearance in *Stahl und Eisen*, 1885, of Brinell's laws. All these creditable steps of progress were rendered more or less uncertain and incomplete by the imperfection of the apparatus and methods of precision, by consequent errors of observation, and by gaps in the data—though the aggregate quantity of material was already overwhelming. Then came, in 1887, and subsequent years, the remarkable investigations and intuitions of Floris Osmond, who discovered that metals frequently combine to form definite chemical compounds, and that these compounds frequently form solid solutions. He discovered also the thermal critical points of iron, and, interpreting these changes in the cooling-curve as indications of some molecular change, propounded the brilliant hypothesis of the allotropy of iron which furnished the acceptable allotropic theory of the hardening of steel. To him, as Professor Sauveur remarks in his Biographical Notice of Osmond [TRANS. (1913) **45,** 274], we owe likewise the discovery of austenite, the non-magnetic solid solution of iron and carbon existing above the thermal critical range, and the transition-constituents, martensite, troostite and sorbite, marking as many distinct and important steps in the transformation, on cooling, of the solid solution austenite into the ferrite-cementite aggregate.

"This brings us to the date of Howe's first book on the subject; and the history of the new science, as I have already sketched it in general terms, is epitomized in the essays and books of Professor Howe from the appearance of his "Metallurgy of Steel" in 1888 to that of his "Metallography of Steel and Cast Iron" in 1916. The first of these books was an amazing accumulation of reported facts, tabulated, verified and explained as far as was then practicable. The last is an equally amazing array of facts, but now sifted, tested, logically arranged and luminously interpreted, exhibiting not uncomprehended differences, but significant similarities and relationships. The first was a heap, parts of which had been sorted; the last is an edifice. To produce the first required intelligent and inexhaustible industry and critical discernment. The second exhibits the creative genius of an architect. Between

the two lies the history of a science to every state of which this builder has made some important contribution. Let me mention a few of these, under the heads of invention of improved methods; discovery of new facts; testing of data; and correlation and interpretation of observed phenomena. I shall not pretend to comprehensive completeness in this survey. I must be content with the exhibition of typical samples.

"One of the new methods invented by Professor Howe is that of determining the microstructural and other conditions which exist in steel at high temperatures by fixing those conditions through the process of quenching—a method which has been generally adopted, and by the use of which much clearer and more trustworthy results have been obtained than were formerly possible.

"Among his discoveries of new facts may be mentioned the isotropy of the effects of plastic deformation (announced in 1888, and supported in 1914); the possibility of effacing blowholes in soft steel by welding (1909); the relations of graphite flakes in cast iron, representing, not complete discontinuities of the metal, but only the filling by graphite of the interstices in ferrite skeletons; and the crystallography of the slip-planes ("Metallography of Steel and Cast-iron," 327). . . .

"In a remarkable paper on the structure of steel, contributed to the proceedings of the American Society for Testing Materials for 1911, Professor Howe collected into the form of propositions or laws the observed relations of heat treatment to the microstructure and mechanical properties of steel. In that paper, he enunciated twenty-three of these laws; and had he then ceased from his labors, his work would have followed him as an imperishable record indelibly carved upon the history of science. But he was destined to go still further during the five years which followed. His latest book, "The Metallography of Steel and Cast-iron," published in 1916, occupies intellectually a higher plane than its predecessors, by which I mean that it commands a wider outlook, permitting grander generalizations, and the recognition, in due perspective, of causes, effects and relations. This book, in short, exhibits that simplification which I have described as following the stage of maximum complexity in the development of a science. Its very title, coupling steel with cast iron, expresses the maturer view which includes both in one continuous series, subject to the same laws—a significant conclusion of the controversy which began with the contention that steel was not cast iron. Not that the difference has been disproved, but that a higher unity has been shown to embrace it.

"That my estimate of the work of our great American colleague, as summed up and set forth in this book, is not merely an expression of personal affection or patriotic pride, let me prove by one or two opinions from eminent authorities.

"I would quote first a few paragraphs translated from a review

of the book by Professor Le Chatelier in the *Revue de Métallurgie* (March–April, 1916):

> The new work of Mr. Howe is entirely original. It has nothing common with the numerous treatises on metallography which have appeared during the last ten years. Whoever has read one of them has read them all. Here we have to do with the personal observations and views of the author. As he indicates in his preface, he has written this treatise, not to state the solidly established principles of a young science, but to cause the creators of that science themselves to think; to open before them new horizons. Hence he has not feared to announce theories sometimes a little hazardous. . .
>
> This study will make an epoch in the history of the progress of science. It represents many years of uninterrupted research, but for a much longer period it will certainly be consulted by investigators with fruitful results.

"The following remarks are taken from a similar review by H. C. H. Carpenter, Professor of Metallurgy at the Royal School of Mines, South Kensington, England.

> It is quite safe to say that Professor Howe's book will at once take its place as an authoritative, and indeed, classical, exposition of the field of knowledge with which it deals. From whatever standpoint it is judged, it is a great book—great in its power, lucidity, balance, comprehensiveness, and pre-eminently scientific character.

"In a private letter, written with reference to the announcement that the John Fritz medal was to be conferred upon Professor Howe, the same distinguished author says that he believes no award could have given greater pleasure to the metallurgists of Great Britain, who look upon Professor Howe not merely as the *doyen* of metallographists in America but as their most distinguished representative, a worker whose single-mindedness in the pursuit of truth is an example to us all, and whose conscientiousness gives to his publications a character of their own."

Doctor Howe is survived by his wife, to whom he was married on April 9, 1874. She was Miss Fannie Gay, of Troy, N. Y., and to her wise guidance and actual assistance and companionship in his work, Doctor Howe always justly attributed a large measure of his success.

<div align="right">BRADLEY STOUGHTON.</div>

Gardner F. Williams

GARDNER F. WILLIAMS, authority on diamond mining, died at San Francisco, Cal., on Aug. 22, after an illness of nearly a year.

Mr. Williams, who was in his eighty-first year, was born in Saginaw,
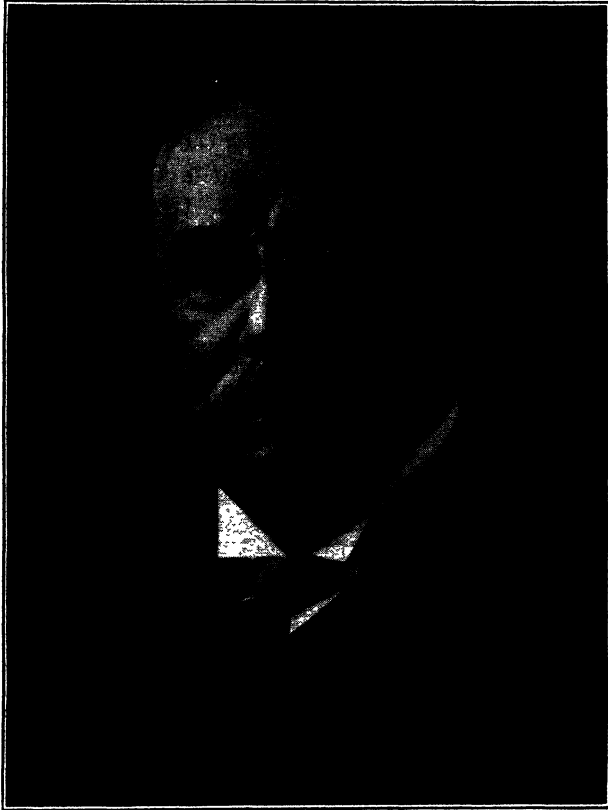
GARDNER F. WILLIAMS

Mich. He was graduated from the University of California in 1865, and afterward took a course at the Freiberg Mining Academy in Saxony. He was awarded the degree of LL. D. by the University of California in 1910, and Doctor of Engineering by the University of Michigan in 1917.

Beginning in 1870, Mr. Williams spent two years as assistant assayer at the United States Mint in San Francisco. He successfully developed

several mines in the West, and then went to South Africa. He was in charge of the gold mines in northern Transvaal during the years 1884–85, and later was engineer for the Exploration Company, Ltd., of London, in South Africa.

In 1887, he became general manager of DeBeers Company, which later became DeBeers Consolidated Mines, Ltd., organized by the late Cecil Rhodes and others, which produced 95 per cent. of the diamond yield of the world. He was in South Africa during the Boer war.

Mr. Williams was the author of several books, the one entitled "The Diamond Mines of South Africa" being especially well known.

He was a life member of the American Institute of Mining and Metallurgical Engineers, which he joined in 1886, and was also a member of the National Geographic Society; the Institution of Mining and Metallurgy, London; British Association for the Advancement of Science, and other scientific bodies. He was a member of the Metropolitan and Chevy Chase clubs of Washington and the Pacific Union Club of San Francisco.

When Mr. Williams returned from South Africa, he established his residence at Washington, D. C., where he took an active interest in civic affairs.

In 1871, Mr. Williams married Fanny Locke, of Oakland, Cal. Mrs. Williams died in 1911. A son, Alpheus Fuller Williams, now general manager of DeBeers Consolidated Mines, Ltd., of South Africa, and three daughters, Mrs. William Wallace Mein of New York City, Mrs. Frederick DeCourcy Faust of Washington, D. C., and Mrs. Monroe Eyre Pinckard of San Francisco, Cal., survive him.

Hjalmar Sjögren

THE cables brought the news last spring that the Institute had lost by death one of its most distinguished foreign members, Hjalmar Sjögren of Stockholm. For thirty-one years, Professor Sjögren had been one of our number. He was elected in 1891, joining the Institute at the time he was in the United States, in attendance at the Fifth International Geological Congress, which was held in Washington in the late summer of that year. Professor Sjögren was once more in America in 1906 and again in 1913, and on both of those visits his friends in New York had the welcome opportunity of extending courtesies to him. He was thus well known to a wide circle in our membership.

His keen interest in the geology of iron ores brought him in touch with the many Americans who have studied them and who have been active in the important productive districts which he visited. Our TRANSACTIONS contain (Vol. XXXVIII) one very important paper of seventy pages on the Geological Relations of the Scandinavian Iron Ores, which he contributed.

Sten Anders Hjalmar Sjögren, to give his full baptismal name, was born June 13, 1856, and was the son of Anton Sjögren, a mining engineer, at the time in charge of the iron mines at Persberg, in the east and west iron-ore belt of central Sweden, about 160 miles west from Stockholm. The Persberg orebodies yielded magnetite and specular hematite from schists, with which limestones were interstratified as well as the peculiar Swedish rocks called "hälleflinte." The mines were also well known to collectors and students of minerals.

While Hjalmar Sjögren was yet a lad, his father was advanced to the superintendency of a number of famous mining localities in this same district of Wermland. Paisberg, famous for its native lead, was one; and two others, Langban and Nordmarken, were among the most prolific sources of interesting minerals in all Sweden. In several localities, the pre-Cambrian limestones had obviously experienced the recrystallizing influences of intrusive igneous rocks and had been charged with the so-called "contact" minerals, which have found their way to the museums of the world. Anton Sjögren, the father, was a keen mineralogist and Hjalmar, the son, grew up in a favorable environment for receiving training and enthusiasms of the same sort. One is not surprised, therefore, to find Hjalmar Sjögren, in the long run, turning to mineralogy and geology as his final career.

In 1875, at the age of nineteen, young Sjögren matriculated at the Polytechnical School in Stockholm, where he spent three years. In 1878, he entered the University of Lund and after two years passed his examinations for the mining engineer's profession. During his student period he also had field experience in geological work and began the publication of his long series of papers on minerals, some old, some new and first named by him. On graduating, the young mining engineer entered the active work of the profession and continued it for two years; then for two years he gave the instruction in mineralogy at the famous old Swedish university of Upsala, where the traditions of Linnaeus and Berzelius still live as an inspiration to teachers and taught. Inevitably, Instructor Sjögren continued with increasing productivity his publications, both in mineralogy and geology, and what is of especial moment in connection with this tribute, he began his important series on iron ores.

BULLETIN OF UPSALA UNIVERSITY

In 1885, Hjalmar Sjögren became geologist for the Nobel oil wells, and refineries at Baku on the Caspian Sea, and for four years studied the geology of the oil fields and of the regions round about, acquiring thus data for a series of scientific papers. Returning to Sweden, he became, in 1890, professor of geology and mineralogy at Upsala, and was married to Anna Nobel, the daughter of Ludwig Nobel. At once the laboratories of the university were equipped with the best of apparatus and placed in a position to carry on researches with the necessary facilities. The well-known "Bulletin of the Geological Institute of the University of Upsala" was established and furnished a medium of publication. The first volume was completed in 1893, and has been followed by seventeen others, the entire expense of the series having been borne by Professor Sjögren. The series is one of the most valuable of all the publications in geology and mineralogy. The volumes are beautifully issued and are in all the large scientific libraries of the world.

In 1894, after four years of service, Professor Sjögren retired from the chair at Upsala and occupied himself primarily with the great business interests and other similar affairs with which he had necessarily become associated. In 1901, however, he took charge of the collections of minerals in the Royal Museum at Stockholm and reorganized the exhibits, both in materials and display, until they were able to supply to the delegates at the Eleventh International Geological Congress of 1910 one of the most striking and instructive features of Stockholm. To the mineralogical treasures of the state, Professor Sjögren added his own collection, which was of outstanding scientific and historical value. The great contribution of the above Congress was the two-volume work on the Iron Ore Reserves of the World. In its preparation Pro-

fessor Sjögren took a leading part; and at the distinguished symposium upon the question "What shall the iron industry do for ore in the future?" he followed the address of the Swedish Prime Minister, with a summary of the world-wide inquiry. During the Congress, and as one of its most enjoyable features, Professor and Mrs. Sjögren made their guests, for an afternoon and evening at their beautiful estate of Nynäs, an hour's ride from Stockholm, one hundred and fifty of the delegates and their wives. A sumptuous banquet was spread, after which a vast display of fireworks illuminated the grounds. As one of those present, the writer can bear witness to the great pleasure given to visitors from all parts of the world.

Since the year of the Congress in Stockholm, Professor Sjögren has been busied with scientific labors and administrative duties of many kinds and with journeys of world-wide extent. At the time of his death an important work entitled "A Handbook of Swedish Mineralogy" was in large part in manuscript, and a translation from the Latin into Swedish of Swedenborg's treatise on iron, in his "Opera philosophica et mineralia," was so far advanced that it can be carried to publication.

Aside from Professor Sjögren's many contributions to mineralogy, his numerous papers on the geology of iron ores possess the greatest interest for members of the Institute. His writings on the latter subject show remarkably well the gradual evolution through which our trusted views have passed, alike in Scandinavia and America. At first, even for the ancient magnetites and specular ores sedimentary explanations were exclusively set forth; but gradually the need of aid from igneous influences and contributions becomes more and more apparent and their importance is more fully recognized in his latest contributions. A full bibliography of Professor Sjögren's writings accompanies a beautifully written and appreciative sketch of his life by Professor A. G. Högbom, his successor in the chair at the University of Upsala. The memorial is published in Volume XVIII of the Bulletin of the Geological Institute of the University. The writer's personal recollections have been greatly aided by this memorial in preparing the present tribute.

JAMES F. KEMP.

R. C. Gemmell

ROBERT CAMPBELL GEMMELL, assistant managing director of the Jackling allied porphyry mining properties in Utah, New Mexico, Arizona and Nevada, died suddenly on Oct. 25, of heart trouble, while traveling from Hurley, N. M., to Los Angeles.
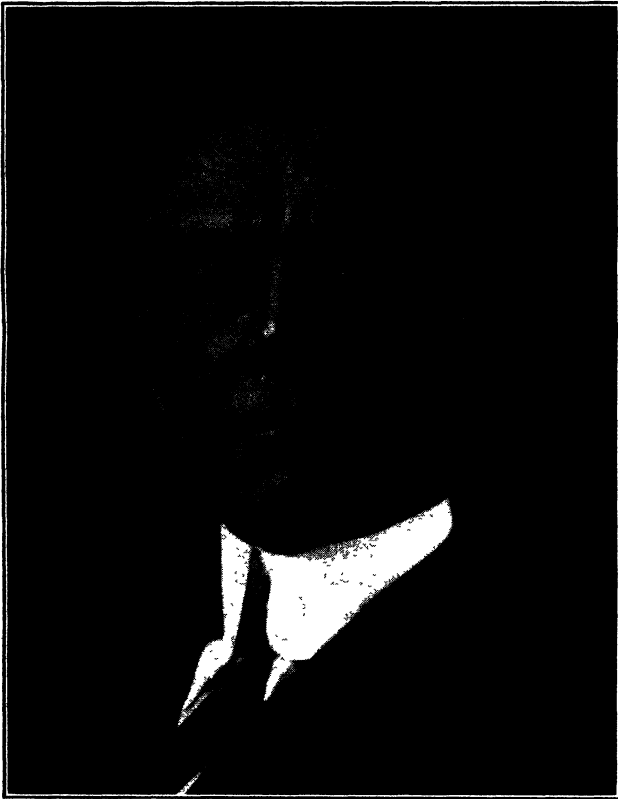
R. C. GEMMELL

To shape justly even an outline of the brilliant career and uniquely charming personality of Mr. Gemmell is well-nigh impossible, and any effort so to do can but demonstrate the inadequacy of language to sound the depths of soul-stirring sentiments. To touch upon his more important work, and afford even a limited conception of his general worth and character, is all I can hope to do.

Robert Campbell Gemmell, the son of Robert Brown Gemmell and Anna Eliza Campbell Gemmell, was born at Port Mathilda, Pa., on July 5, 1863. In the early 70's, his parents moved to Topeka, Kansas, in which city Mr. Gemmell secured a common school education and graduated from the Topeka High School. His father being in, and having made good at, the railroad business, Mr. Gemmell decided upon becoming a civil engineer, and to that end, in 1880, entered upon the study of his profession at the University of Michigan, from which he received his degree of B. S. in 1884.

From the summer of 1884 until the summer of 1890, he was employed in the engineering department of the Atchison, Topeka & Santa Fe Railway Co., rising to the position of division engineer in charge of construction work, with 2000 men under his direction. In 1891, he located in Pendleton, Ore., where he engaged in an extensive private practice of his profession extending over Oregon, Washington and Idaho. In 1893, he located in Salt Lake City, Utah, where, in addition to his private practice, he was employed in the engineering department of the city. He returned to his alma mater, at Ann Arbor, Michigan, and there received his degree of C. E., in 1895. Resuming his practice in Salt Lake City, he became engineer for the mines of Capt. Joseph R. DeLamar including the Golden Gate and the Consolidated Mercur properties at Mercur, Utah. Upon the admission of Utah to statehood in the Union, he became its first state engineer, and held that position from 1898 to 1901. He next went to Mexico as manager of the Mexican Syndicate Mining Co., followed by the superintendency of the properties of the Guggenheim Exploration Co. In Mexico, he acquired proficiency in the Spanish language. In 1905, he journeyed to Spain, where he examined gold-mining properties for a syndicate of New York capitalists.

Immediately upon his return, and in January, 1906, he became associated, in the capacity of superintendent of mine and mills, with the Utah Copper Co., of which D. C. Jackling was then general manager, and continued with that company, which through the joint efforts of those gentlemen has been brought to such a productiveness as to make its name known throughout the mining and financial world. Three years later he became assistant general manager of the Utah Copper Co. and on May 1, 1913, became its general manager, succeeding Mr. Jackling, whose various and extended interests obliged him to move to San Francisco, Cal., and who became its managing director and vice-president. At this time, in June, 1913, his university conferred upon Mr. Gemmell the honorary degree of Master of Engineers, the highest degree the profession affords. During these years, he became consulting engineer for the Ray Consolidated Copper Co., the Chino Copper Co., and the Nevada Consolidated Copper Co., as well as general manager of the

Bingham & Garfield Railway Co. In June, 1919, he was made assistant managing director of the Utah, Ray, Chino, and Nevada properties, in which capacity he had visited the Chino property, when, on Oct. 25 last, while en route to Los Angeles, Cal., on the Sunset Limited, he lay down to take a nap, and, while he was sleeping peacefully, passed away.

On Oct. 17, 1888, Mr. Gemmell married Miss Belle Evans Anderson of Salt Lake City, who was also a Michigan graduate, and a leader in the social and club life of this city, and who survives him.

He was fond of hunting and fishing, an ardent golf player, and a clubman of popularity and prominence. He was never too busy to concern himself with the uplift of boys, and it pleased him, beyond measure, to contribute to their happiness. The public good found him always responsive to the demands and duties of patriotic citizenship. All that tended to aid Utah in a material way, to develop her resources, add to her capital, population or industries, to encourage education, and to relieve the suffering of her needy, found in him a ready and enthusiastic champion.

He was a true man in all that the word implies. His genuine individuality never resorted to an artificial diplomacy, never pretended to an agreement for the sake of an advantage, and never made concessions to escape responsibilities. His broad and liberal views, his moral and upright character, his kind and gentle disposition, his polite and unassuming manner, his frank, generous, hearty, cheery good will to all, won men to him and gave him a power over human hearts which was both deep and strong. His career was one of magnificent development and broad usefulness. Seldom have men been endowed with such vigor of intellect. His was a sober judgment that measured opportunities, and the sagacity and self poise that enabled him to plan wisely and execute efficiently. The lofty heights reached by him in his chosen profession bear indisputable testimony of his untiring zeal, unconquerable energy and unquestionable ability.

The engineering profession can have no better thing to be proud of than such a character; in all her store, she has no richer jewel to display than the fame of such a son. Mr. Gemmell well earned the high position he held in our councils and in our hearts.

> A whiter soul, a fairer mind,
> A life with purer course and aim,
> A gentler eye, a voice more kind,
> We may not look on earth to find;
> The love that lingers o'er his name,
> Is more than fame.

Mr. Gemmell was a member of the American Institute of Mining and Metallurgical Engineers, the Mining and Metallurgical Society of America, the American Society of Civil Engineers, the Utah Chapter of the American Mining Congress (he was the prime organizer and the

first governor of this chapter), and the Sons of the American Revolution. He was honorary president and national representative of the Salt Lake Council of the Boy Scouts of America. He was also appointed by the Governor of Utah on the committee which, in 1916, prepared the draft of the Workmen's Compensation Act of Utah.

He was a member of the Alta, the University, the Country, the Commercial and Rotary clubs of Salt Lake City; of the Rudy Duck and the Bear River Duck clubs of Utah; of the Flat Rock Club of Idaho; and of the Sequoyah Country Club of Oakland, Cal.

D. C. Jackling, in a letter written to the Institute, expressed the feeling of Mr. Gemmell's many friends. He said: "Gemmell, personally, was the kind of man whom it was a rare privilege to know and call friend; as an engineer, he was capable and thorough to the degree of brilliance; as an operating official, he was patient and considerate as few have the character to be, but, at the same time, persistent to a degree that could not be satisfied by anything short of economic perfection. Altogether, he was a great man, a great engineer, and a great friend.'

Levi Holbrook

Levi Holbrook was born in Westboro, Mass., March 7, 1836. He was a descendant of John Holbrook, who came from England in 1660 and settled in the Massachusetts Bay Colony.

Mr. Holbrook was prepared for college at Williston Seminary, East-hampton, Mass., and after one year spent in the West, he entered Yale. He was a member of Phi Beta Kappa and of Linonia.

During the six months following his graduation he was in Boston, under medical treatment for his eyes; then he took a horseback trip across the Rocky Mountains to the Columbia River and back, and resided in Danville, Va., for a time. From 1860 to 1863, as his eyes permitted, he was a student of modern languages and literature in Cambridge, Mass., part of the time being a resident graduate at Harvard University. Finally, however, he was obliged to abandon the idea of a literary life and from 1864 to 1867 he was engaged in financial business in Boston. The following year, he spent in New York, and then traveled for a year in Europe, Egypt and Asia Minor. In 1871, he returned to New York City.

Mr. Holbrook was a member of the New England Historic Genea-logical Society, the Society of Colonial Wars, and the Sons of the Revolu-tion, was registrar general of the Order of the Founders and Patriots of America from 1915 to 1920, and had been governor of the New York society of that order. He was also a member of the American Fine Arts Society, the Order of the Cincinnati, the American Geographical Society (of which he was a councillor and secretary), and the Geographical Society of America. He joined the American Institute of Mining Engi-neers in 1878, and served as manager from 1895 to 1897.

Mr. Holbrook died on July 26, 1922, from organic heart trouble, at his residence in Center Harbor, N. H. He was married Dec. 27, 1871, in Newark, N. Y., to Viola, daughter of John A. and Elizabeth (Failing)Vowers, who died on Sept. 10, less than six weeks after her husband. His two children, Clark Holbrook of Red Bank, N. J., and Mrs. Julian P. Smith, survive him.

INDEX

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